TECHNOLOGY, EMPLOYMENT, AND OUTPUT PER MAN IN

COPPER MINING

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BUREAU OF MINES

FEDERAL WORKS AGENCY
WORK PROJECTS ADMINISTRATION

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Commissioner

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Assistant Commissioner

NATIONAL RESEARCH PROJECT
on
Reemployment Opportunities and Recent Changes
in Industrial Techniques

DAVID WEINTRAUB
Director

In cooperation with

UNITED STATES DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

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Mineral Technology and Output Per Man Studies
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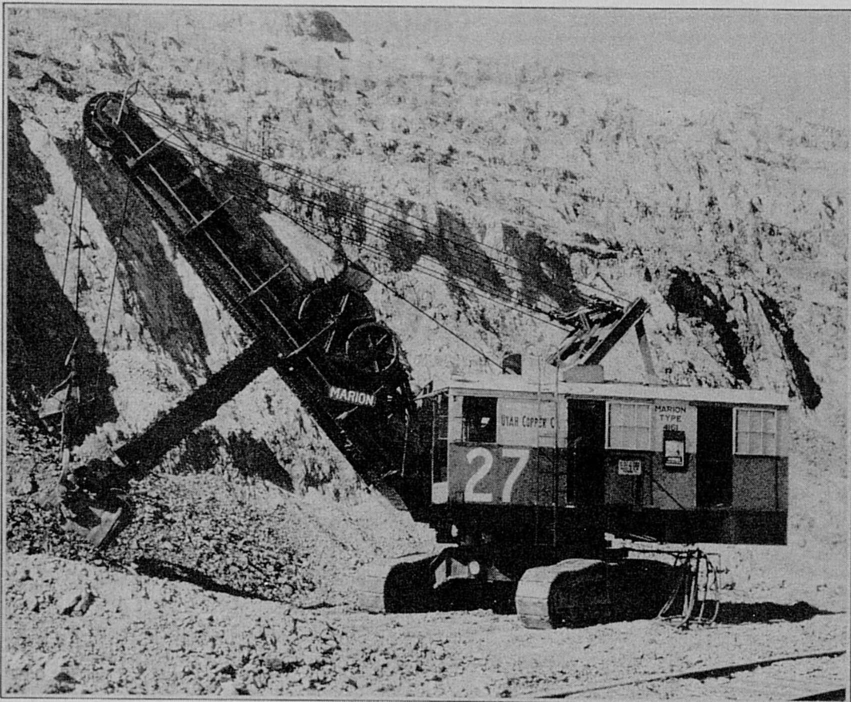

MINING COPPER ORE WITH A POWER SHOVEL AT BINGHAM CANYON, UTAH

The exploitation of large-tonnage, low-grade ore deposits and the evolution of low-cost surface mining techniques have been important developments in the copper industry during the past 30 years. Railroad-type steam shovels running on tracks were first used to strip overburden and to load ore, but they have been superseded by full-revolving, caterpillar-mounted electric shovels. Power shovels have been important in lowering labor requirements per ton of copper ore mined. The shovel itself has been steadily improved and now requires much less labor for maintenance than formerly. At underground mines there has been an increasing tendency to adopt smaller, special-type shovels as an aid to efficiency.

TECHNOLOGY, EMPLOYMENT, AND OUTPUT PER MAN IN COPPER MINING

by

Y. S. Leong, Emil Erdreich, J. C. Burritt,
O. E. Kiessling, C. E. Nighman, and
George C. Heikes

WORK PROJECTS ADMINISTRATION, NATIONAL RESEARCH PROJECT
In cooperation with
DEPARTMENT OF THE INTERIOR, BUREAU OF MINES

Report No. E-12

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THE WPA NATIONAL RESEARCH PROJECT ON REEMPLOYMENT OPPORTUNITIES AND RECENT CHANGES IN INDUSTRIAL TECHNIQUES

Under the authority granted by the President in the Executive Order which created the Works Progress Administration, Administrator *Harry L. Hopkins* authorized the establishment of a research program for the purpose of collecting and analyzing data bearing on problems of employment, unemployment, and relief. Accordingly, the National Research Program was established in October 1935 under the supervision of *Corrington Gill*, Assistant Administrator of the WPA, who appointed the directors of the individual studies or projects.

The Project on Reemployment Opportunities and Recent Changes in Industrial Techniques was organized in December 1935 to inquire, with the cooperation of industry, labor, and governmental and private agencies, into the extent of recent changes in industrial techniques and to evaluate the effects of these changes on the volume of employment and unemployment. *David Weintraub* and *Irving Kaplan*, members of the research staff of the Division of Research, Statistics, and Finance were appointed, respectively, Director and Associate Director of the Project. The task set for them was to assemble and organize the existing data which bear on the problem and to augment these data by field surveys and analyses.

To this end, many governmental agencies which are the collectors and repositories of pertinent information were invited to cooperate. The cooperating agencies of the United States Government include the Department of Agriculture, the Bureau of Mines of the Department of the Interior, the Bureau of Labor Statistics of the Department of Labor, the Railroad Retirement Board, the Social Security Board, the Bureau of Internal Revenue of the Department of the Treasury, the Department of Commerce, the Federal Trade Commission, and the Tariff Commission.

The following private agencies joined with the National Research Project in conducting special studies: the Industrial Research Department of the University of Pennsylvania, the National Bureau of Economic Research, Inc., the Employment Stabilization Research Institute of the University of Minnesota, and the Agricultural Economics Departments in the Agricultural Experiment Stations of California, Illinois, Iowa, and New York.

Since September 1, 1939, the Project has been sponsored by the National Resources Planning Board, Executive Office of the President, Washington, D. C.

FEDERAL WORKS AGENCY
WORK PROJECTS ADMINISTRATION
1734 NEW YORK AVENUE NW.
WASHINGTON, D. C.

F. C. HARRINGTON
COMMISSIONER OF WORK PROJECTS

February 20, 1940

Colonel F. C. Harrington
Commissioner of Work Projects

Sir:

There is transmitted herewith the report entitled *Technology, Employment, and Output per Man in Copper Mining.* Although the copper-mining industry employed fewer than 18,000 workers in 1938, its product is an essential, widely used industrial raw material and an indispensable war material. The industry's history affords, moreover, an exceptional picture of the impact of changing production methods on labor productivity and employment opportunity.

In terms of ore produced per man-shift, productivity increased from 0.6 ton in 1880 to 6.3 tons in 1928 and to 8.8 tons in 1936. Because the quantity of copper yielded per ton of ore has declined over the decades as poorer ore deposits have been tapped, the rise of productivity appears somewhat less rapid when measured in terms of recoverable copper.

Thus productivity has increased steadily despite growing natural handicaps. These productivity increases were made possible by technological advances in mining practice and in ore dressing. Production, however, has not kept pace, and employment has therefore declined. In 1929 the copper output was at practically the same level as in 1916, but only 37,000 men were employed as against 61,000 in the earlier year. In 1936 copper production was about 60 percent but employment was only about 38 percent of 1929.

The industry's development has involved the shifting of production centers in response, on the one hand, to depletion of deposits in the older producing areas and, on the other hand, to new mining and ore-concentration methods that made possible the exploitation of low-grade deposits in the West. Thus in Michigan, which once was the leading copper-producing

State, the output of a group of large mines declined from 225 million pounds in 1917 to 186 million in 1929 and to 96 million in 1936. Employment fell off even more rapidly, from 11,886 men in 1917 to 6,660 in 1929 and to 1,838 in 1936. In Arizona, now the largest copper producer, the 1929 production of 819 million pounds was 35 percent above its 1917 level, and 1929 employment was slightly lower than in 1917 - an average of 11,246 as against 11,909 men. In 1936, however, employment totaled only 3,696 men, for production was about half as great as in 1929 and productivity meanwhile had been greatly increased.

Something of what shifting centers of production have meant to the older centers can be seen in the familiar problem area of northern Michigan where copper resources have recently reached a point of serious depletion. In two such counties in the Lake Superior region population declined by almost 40 percent between 1910 and 1930. The migration has continued since then; yet it was estimated that in 1936 as many as 3,000 copper miners and their families were stranded in these two counties with little opportunity for reemployment. They have constituted an important relief problem.

The outlook for increased employment in copper mining, as in most mineral industries, is not bright. Intensification of the European war would be expected to increase our copper exports, and this country's armament program will call for an increased amount of copper. Output for other than armament purposes is not expected to be much above the 1936-37 level in the next 2 or 3 years. Over a 10-year period, however, the growth of population, the development of new uses for copper, and further growth of the electrical industries (which now account for about one-half of domestic consumption) point to some increase in copper production. However, productivity appears certain to increase further with continued adoption of improved techniques and with continuation of the shift in production to deposits having a higher output per man-hour.

Respectfully yours,

Corrington Gill

Corrington Gill
Assistant Commissioner

CONTENTS

CONTENTS

MAP, CHARTS, DIAGRAMS, AND ILLUSTRATIONS

MAP, CHARTS, DIAGRAMS, AND ILLUSTRATIONS-*Continued*

MAP, CHARTS, DIAGRAMS, AND ILLUSTRATIONS-*Continued*

TEXT TABLES

TEXT TABLES-*Continued*

PREFACE

The history of copper mining in this country affords a striking example of the manner in which the Nation's growing needs for important raw materials have in many instances been satisfied with a continually lower expenditure of labor per unit of output as well as with a declining total volume of employment. Many factors have contributed to these developments; the most important appear to have been changes in methods of mining copper, improvements in mechanical equipment, and increased application of mechanical power in mines and in ore-dressing plants.

The twentieth century has seen the rise of the open-cut method of mining copper. From less than 2 percent in 1907, the copper output of open-cut mines has grown to over two-fifths of the total. Large open-cut operations use mass-production methods of breaking, loading, and hauling ore that cannot be used in underground vein mines. In consequence, output of ore per man-hour is four or five times as great in open-cut operations as at underground vein mines, and copper output per man-hour is more than twice as great.

The rise of open-cut copper mining depended upon two principal technological developments: mechanical methods of handling large volumes of materials and improvements in ore-dressing techniques. The open-cut method of mining with power shovels and other large-scale mechanical equipment was borrowed from the iron-mining industry. Improvements in gravity concentration and the development of the flotation process of ore dressing have made commercially possible the recovery of copper from low-grade disseminated ores that 40 years ago were considered worthless by most engineers. It is these deposits that are now being worked by the open-cut method.

At underground mines, too, there have been significant advances in mining methods. Most important of these is the development of modern undercut caving methods that give to underground mines some of the advantages accruing to open-cut operations through mass-production methods of handling ore.

Over the several decades covered by this study, the most important influence on employment opportunity in copper mining has been the rising output per worker that has resulted from these changes in mining methods and from mechanical innovation and improvement of equipment and power utilization. Associated with these technological changes, which require large capital outlays for development work and for operating equipment, has been the trend toward a greater concentration of control in the hands of large producers and toward fewer and larger operating units. In 1922 the three largest producers accounted for about one-third of the copper production; in 1936 they produced about three-fourths of the total.

Centers of production have shifted as a result of depletion in some of the older areas and the increasing exploitation of low-grade ores in the newer western regions. Thus Michigan and Montana have declined in both absolute and relative importance while Arizona and Utah have become the most important copper-producing States. Since by and large the newer areas are those where labor productivity is highest, the growing relative importance of these areas has in itself served to decrease unit labor requirements in the industry as a whole. In addition, the southwestward migration of the industry, the shift to surface operation, and changes in underground methods have altered the character of copper-mining occupations and the locations of employment opportunity and thus have created adjustment problems for many of the copper miners.

This report is one of a series of studies conducted by the National Research Project in cooperation with the United States Bureau of Mines under the direction of O. E. Kiessling. It was prepared by Y. S. Leong, who was loaned for this study by the Central Statistical Board; Emil Erdreich, J. C. Burritt, C. E. Nighman, and George C. Heikes of the National Research Project; and O. E. Kiessling of the Bureau of Mines. The manuscript was edited and prepared for publication under the direction of Edmund J. Stone.

DAVID WEINTRAUB

PHILADELPHIA
February 19, 1940

ACKNOWLEDGMENTS

Acknowledgment is made of important aid given in the preparation of this report by officials of the United States Bureau of Mines. Data on ore production, yield, and depth of mines were supplied by Chas. W. Henderson, Supervising Engineer of the western field offices of the Mineral Production Division, and by J. P. Dunlop, C. W. Merrill, and T. H. Miller, respectively in charge of field offices of the same division at Joplin, Mo.; San Francisco, Calif.; and Salt Lake City, Utah. C. N. Gerry, Supervising Statistician of the last-named field office, also supplied valuable information and suggestions. Employment data utilized in this survey were made available by W. W. Adams, Supervising Statistician, Employment Statistics Section; M. E. Kolhos of the same section gave important help in the use of this material. C. F. Jackson, Supervising Engineer, and C. E. Julihn, Principal Mining Engineer, of the Mining Division, and E. D. Gardner, Supervising Engineer of the Tucson (Arizona) office of the same division, gave important aid in the classification of mining methods and in supplying information on mine ventilation. J. W. Furness, Chief Engineer; E. W. Pehrson, Asst. Chief Engineer; and H. M. Meyer, Senior Clerk, all of the staff of the Metal Economics Division, supplied valuable information and suggestions. The writings and suggestions of Daniel Harrington, Chief of the Health and Safety Branch, have been heavily drawn upon in preparing sections of the discussion concerned with the health and safety of miners; F. S. Crawford, District Engineer of the Duluth (Minnesota) office of the same branch, aided in the classification of mining methods. The preliminary examination of statistical data was conducted under the supervision of R. J. Lund, formerly Mineral Economist of the Bureau and now Editor of *The Mining Congress Journal*.

Grateful acknowledgment is also made to WPA National Research Project staff members who carried out major tasks in the conduct of the study. Responsibility for the accuracy of the statistical compilations fell chiefly upon Sidney Ginsburg and M. B. McIntyre, who also made important suggestions for improvement of the report. D. C. Athanas, E. M. Steuart, and P. J. Wester carried out special assignments in the preparation of the manuscript. Cooke Settle was responsible for the lay-out of the charts and furnished valuable aid in writing the technical sections of the report. R. E. Good, H. Cowles, and H. Tilford assisted in organizing and preparing the statistical material. W. S. Stanley prepared preliminary sketches of the illustrations of mining methods.

Valuable information on mining methods was supplied by John Norton and D. R. Rait, Mining Engineers in the Mining Section, Reconstruction Finance Corporation. Special thanks are also expressed to I. Alleman who completed the drawings presented in figures 29 to 33, inclusive, through the courtesy of Howard B. Myers, Director of the Division of Research, and E. G. Millison, Chief of the Graphic Section, Work Projects Administration.

Special data or photographs presented in this report were kindly supplied by the following concerns: Anaconda Copper Mining Company, 25 Broadway, New York City, N. Y.; Utah Copper Company, Kearns Building, Salt Lake City, Utah; Ingersoll-Rand Company, 11 Broadway, New York City, N. Y.; Gardner-Denver Company, Quincy, Ill.; Eimco Corporation, 634-666 South Fourth West St., Salt Lake City, Utah; and Worthington Pump and Machinery Corporation, Harrison, N. J. Thanks are also expressed for photographic material that was courteously furnished for use in this report by the United States Bureau of Mines.

CHAPTER I

INTRODUCTION[1]

The present study attempts to analyze the effect of changes in technology and in mining conditions on employment opportunities in one of the most important mineral industries - copper mining - and to determine the prospects it offers for employment in the next decade and, as far as possible, over a longer span of years.

Fundamentally, the number of units of labor (man-hours, man-days, or man-shifts) required depends on two main factors: the quantity of production and the output per unit of labor. If the output per unit of labor increases at a more rapid rate than the volume of production, labor requirements will be curtailed; and vice versa, if the volume of production grows at a faster rate than the output per unit of labor, labor requirements will be increased.

The level of output per unit of labor in copper mining and in practically all other mineral industries is influenced not only by advances in technology, as in manufacturing, but also by changes in mining conditions. In manufacturing, improvement in technology generally means a gain in the output per man; in mining, it may be counterbalanced by increasing physical difficulties in the forms of gradual depletion or less accessible ores. Growing physical handicaps and progressive depletion of irreplaceable resources are the most distinctive characteristics of copper mining in particular and of the mineral industries in general. As operators usually exploit the richest and most accessible deposits first, they must eventually turn to poorer and less-easily mined ores.[2] Migration to newly discovered areas having richer and more readily extracted ores or to known deposits made available by changes in market location or the growth of the transportation system may to some extent offset increasing difficulties. Other factors, such as increasing efficiency resulting from mass methods of production and improvements in management techniques, may join with technology in lightening the burden of accumulating handicaps of nature. The struggle, then, is between technology aided by these allies on the one hand and increasing physical

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[1]By Y. S. Leong, Senior Economist, Central Statistical Board and O. E. Kiessling, Chief Economist, Mineral Production and Economics Division, U. S. Bureau of Mines.

[2]It should be remembered that complete exhaustion of an ore body rarely occurs. What happens is that as the higher-grade and more accessible ore is exhausted, the operator must proceed to exploit the lower-grade and less-easily mined ore at increasing cost. As his cost increases in relation to that of his competitors who are more favorably situated - that is, less handicapped by nature - he must eventually abandon his mine even though the deposit may not be absolutely depleted.

difficulties on the other; the results are reflected in the index of productivity - "average output per worker per 8-hour day" - which, for the sake of brevity, may be designated as "output per man-day" or "output per worker."[3]

In view of the fact that the future grows out of the past and that industrial development is a continuous process, a study of past trends may throw some valuable light into the uncertain future. Thus this study goes as far back as 1880 for its starting point. The year 1880 marks the earliest date for which fairly reliable statistical data on copper mining are available.

The general method of approach may be outlined briefly. First, the long-time changes in production, employment, and output per worker are analyzed. Next, the fundamental factors affecting the output per worker, particularly advances in technology and the growth in physical handicaps, are considered. Finally, an attempt is made to ascertain the probable changes in employment in the copper-mining industry in the next decade, and over a longer period on the basis of the anticipated trend in copper consumption, the availability and character of the reserves, and the expected developments in technology, mining conditions, and other major factors.

As productivity in copper mining is determined largely by the methods employed in extracting ore, the analysis of factors that affect the output per worker will be made separately for the major mining methods. Unfortunately, the absence of accurate, detailed data for the earlier part of the period covered by this study does not permit such analyses to be carried back to 1880. However, a general survey covering the period from 1880 to 1936 is presented in chapter II, which discusses the long-time changes in technology, mining conditions, location of centers of production, corporate organization, and other factors and which summarizes the effects of the changes in these factors on the output per worker in the copper-mining industry as a whole.

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[3]The "average output per worker per 8-hour day" is a ratio of the total production of ore or copper in a given year to the total number of 8-hour man-days expended in producing that output. All mining labor and incidental surface labor necessary to carry on mining operations are included. Man-days worked by administrative and clerical employees are excluded. Indirect labor - that is, labor embodied in machinery and other equipment utilized and in materials consumed in mining - is also omitted unless the work is performed at the mine site, as for example, the work that is done in machine shops and carpenter shops attached to individual mines.

A man-day may be defined as 8 hours spent at work by one employee. The length of the workday throughout the greater part of the period covered by this study was approximately 8 hours. Where there was any appreciable variation from the 8-hour day, as in some districts during the early part of the period or as in the recent depression when employment was staggered, a correction was introduced to make the man-day equivalent to 8 hours.

More detailed data are available for open-cut mines beginning in 1914 and for underground mines beginning in 1917. For the more recent period, therefore, it has been possible to study the two large groups into which all copper mines may be conveniently classified.

Chapter III presents the analysis of open-cut mining. The expansion in open-cut operations and the rise in the output per worker are indicated; the evolution of open-cut technology and the changes in mining conditions are traced; and the effect of these factors on the output per worker in open-cut mining is appraised.

Chapter IV traces the development of underground mining as a whole, and of the several major methods employed in underground mines. Estimates are presented of the influences on output per worker of advances in technology, improvement in mining methods, and increasing mining difficulties.

Ore dressing does not properly come under a discussion of developments in mining processes. However, as the improvements in milling and concentration, particularly with the adoption of froth flotation, have enabled the mining industry to utilize lower and lower grades of ore, to improve the yield from ores milled, and thus to increase the output per worker, a brief discussion of the evolution of ore-dressing techniques is included in this study.[4] Accordingly, the trend of copper recovery in milling, the technologic advances in ore dressing, and the influences of these improvements in ore dressing on the trends of yield of ores are examined in chapter V.

Chapter VI considers some of the effects of technologic and other changes on the workers in the copper-mining industry.

Finally, chapter VII presents the outlook for productivity and employment in the copper-mining industry in the next decade[5] and over a longer period. In this chapter the future ore-reserve situation, the probable improvements in technology, the anticipated advances in ore dressing, the expected domestic consumption and foreign trade in copper, and the future supply of scrap copper are considered and evaluated.

A few explanations regarding the limitations of this study may be helpful at this juncture. The factors that influence the output per worker in copper mining are, of course, more numerous than those considered in this report. Among the more important of those omitted from the analysis

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[4]It should be noted that the employment afforded by the ore-dressing process is not included in the basic data of the report.

[5]Estimates of the probable trends of employment and other factors are projected a decade ahead beginning in 1937 since 1936 is the most recent year for which detailed statistics were available for consideration in this report.

are competition within the industry, competing products, availability of capital, and level of wage costs. These factors, however, affect the output per worker indirectly rather than directly; they induce the adoption of technologic improvements and influence the rate at which they are adopted through the desire to reduce costs and maximize profits.

It will be noted, too, that there is an apparent tendency in the discussion to relate the rise in productivity to technologic improvements as if they were the only factor affecting output per worker. Sight is not lost of the fact that other elements also influence the output per worker. Mention of these is purposely omitted in such instances because technologic advances are the most important factor in explaining the increase in the output per worker.

Another limitation of this study is the lack of statistical data to permit a more adequate analysis of the factors influencing the changes in labor productivity as measured by the output per worker. The discussion of technologic changes must be mainly qualitative. As has already been stated, the measure "average output per worker per 8-hour day" has been utilized in this study to register the effect of technologic advances and other factors on labor productivity. Reliance must be placed mainly upon this index for evidence as to whether or not technology is overcoming the burden of cumulating physical difficulties. In some instances, however, in the absence of requisite data, it has been necessary to show the effect of technology and other factors on labor productivity only in terms of a reduction in costs or operating expenses. Where such figures are employed, it is with the conviction that they largely indicate a corresponding decrease in the amount of labor expended.

It will also be observed that the description of the various technologic changes is rather general. Space does not permit discussion of each of the many important technologic improvements in technical detail. Moreover, most technologic developments have already been discussed at considerable length in the voluminous monographs and reports published by the United States Bureau of Mines and in the numerous articles appearing periodically in the mining and metallurgical journals. The report attempts, then, to give a bird's-eye view rather than a detailed technical exposition of the technologic changes and other factors that have had significant effect on the output per worker. Readers who are interested in greater detail are referred by footnotes to literature on the subject.

CHAPTER II

MAJOR TRENDS IN THE COPPER-MINING INDUSTRY, 1880-1936[1]

Although copper gave way to iron 3,000 years ago when the Bronze Age was supplanted by the Iron Age, it is still one of the most useful metals, second perhaps only to iron in importance. With the advent and development of electric energy copper has assumed a new significance. In fact, the expansion of the copper industry in this country largely paralleled that of the electric-power industry. Because of its high electric conductivity and comparative cheapness, copper has become practically a necessity for the growth of the electrical industry. In recent years about 50 to 60 percent of the copper consumed in this country has gone into manufacturing of equipment for the production, distribution, and utilization of electricity. If the price of copper should remain at the low level prevailing in recent years, substitutes are unlikely to take over this field.

The electrical industry is of course not alone responsible for the growth of copper production, for the automobile and building-construction industries have also been important users, taking from 20 to 25 percent of the total copper consumed in the United States from 1933 to 1937.

CHANGES IN PRODUCTION, EMPLOYMENT, AND OUTPUT PER WORKER[2]

Although copper was mined in this continent by white men as early as the seventeenth century, the birth of the copper industry in the United States is generally placed at about the middle of the nineteenth century when the Lake Superior region in Michigan first began to produce. In 1845, the year for which production figures first became available, this country produced 112 tons of copper. Of this amount Michigan contributed 13 tons. In the next 25 years the Lake Superior area gradually grew in importance, and by the early seventies it produced about four-fifths of

[1]By Emil Erdreich, Industrial Economist, WPA National Research Project; Y. S. Leong; and O. E. Kiessling.

[2]Lead, zinc, gold, and silver are produced as byproducts of copper production. These subsidiary metals are not considered in presenting the principal trends on production and productivity in this report. However, figures showing the output of the four byproducts are presented in table A-7 for a group of copper mines whose output of copper was in excess of 2,000,000 pounds in 1929 and whose operations are classified by the following mining methods: Open-cut, block-caving, open-stope, square-set, cut-and-fill, and shrinkage. For the period 1925-29 the copper equivalent of the output of the four subsidiary metals for the entire group of mines represents 7.5 percent of the copper production. (The method of converting the byproduct metals into their copper equivalent is shown in table A-7.) For the same period the output of copper per man-shift would have been increased from 170 pounds to 183 pounds had the copper equivalent of the accessory metals been included in its computation.

the total copper output of the United States. Meanwhile many ore bodies in other areas were discovered and mined. The deposits in Arizona, notably those in the Clifton-Morenci, Globe, Bisbee, and Jerome districts, which were first exploited in the seventies were destined to play an increasingly important part in the twentieth century. While the Lake Superior area was in the ascendant the vast deposits in Butte, Montana, were discovered. By 1887 Montana surpassed Michigan in production, becoming the principal copper-producing State and maintaining this position until 1907, when it was displaced by Arizona. In the first decade of the twentieth century an entirely new development occurred in copper mining - the exploitation of porphyry or low-grade disseminated deposits, which had long been known to exist but which had to wait until mining and metallurgical technology was advanced sufficiently to permit them to be mined commercially. Between then and the World War some of the largest porphyry mines were developed, namely, those at Bingham, Utah; Ely, Nevada; Santa Rita, New Mexico; Ajo, Arizona; and Bisbee, Arizona. The only other important development to be mentioned is the exploitation of fabulously rich deposits in the Copper River region in Alaska in about 1911. This roughly outlines the history of copper mining in the United States.

The changes in production, employment, and output per worker since 1880 are illustrated graphically in figure 1.[3]

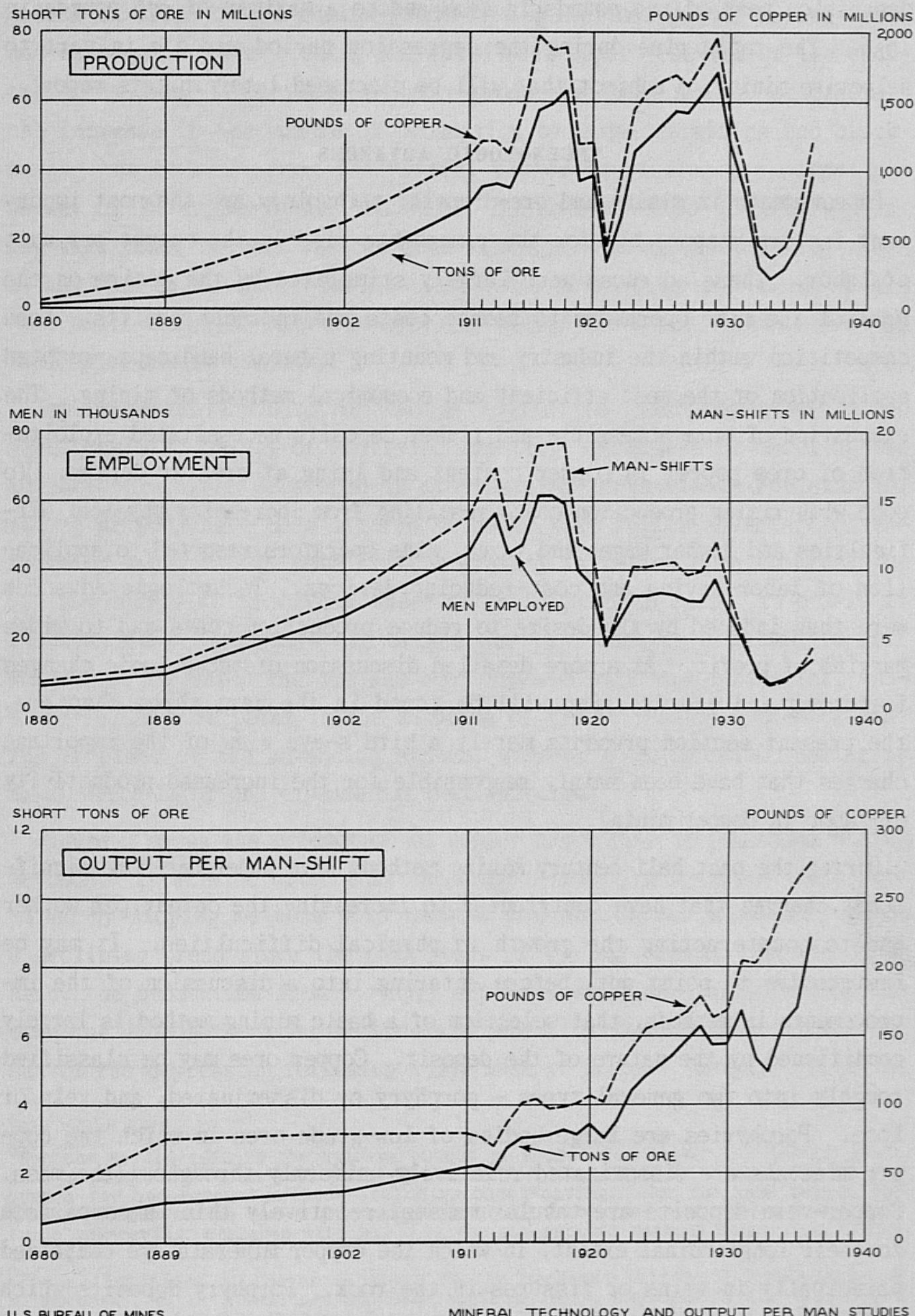
The output of primary or new copper increased from 60 million pounds in 1880 to 2 billion pounds in the peak years 1916 and 1929; the output of copper ore rose from 1 million tons in 1880 to 68 million tons in 1929. It will be noted that the production of both metal and ore did not expand at an even rate but fluctuated violently with the business cycles. The spectacular gain in copper production was due primarily to the expansion of the copper-using industries and secondarily to the general industrial growth.

Employment in the copper-mining industry increased less rapidly than production. The number of wage earners increased from about 6,000 in 1880 to a peak of 61,000 in 1916-17. Since 1917 employment has been declining. In 1929, when the magnitude of the copper output was the same as that in 1916, only 37,000 men were employed. The decline in employment, like that in production, was by no means uniform. The trough in employment was reached in 1933 when only 7,000 men were reported as engaged in mining copper.

During the 57-year period the output per worker as measured in terms of tons of ore per man-shift increased from 0.6 in 1880 to a predepression

[3]For discussion on comparability and coverage see table A-1, ftn. a.

Figure 1.- PRODUCTION, EMPLOYMENT, AND OUTPUT PER WORKER AT COPPER MINES, 1880-1936



U. S. BUREAU OF MINES
BASED ON TABLE A-1

MINERAL TECHNOLOGY AND OUTPUT PER MAN STUDIES
WPA-NATIONAL RESEARCH PROJECT E-188

peak of 6.3 in 1928 and to 8.8 in 1936. In terms of recoverable copper the rise was less rapid because of declining yield of ore. From 31 pounds per man-shift in 1880 the output of copper per worker increased to a pre-depression peak of 179 pounds in 1928 and to a maximum of 276 pounds in 1936. The rapid rise during the depression period was due in part to selective mining, a subject that will be discussed later in this report.

TECHNOLOGIC ADVANCES

Improvements in mining and ore-dressing technology are the most important factors responsible for the remarkable rise in the output per unit of labor. These advances were largely stimulated by the desire on the part of the mine operators to reduce costs and increase profits. Keen competition within the industry and mounting natural handicaps prompted application of the most efficient and economical methods of mining. The exhaustion of more accessible and richer deposits necessitated exploitation of ores poorer in copper content and lying at greater depths. To cope with rising production costs resulting from increasing physical difficulties and higher wages and taxes, mine operators resorted to application of labor-saving and cost-reducing devices. Technologic advances were thus induced by the desire to reduce production costs and to widen margins of profit. As a more detailed discussion of technologic changes in mining and ore dressing will be found in the next three chapters, the present section presents merely a bird's-eye view of the important changes that have been mainly responsible for the increased productivity of labor in copper mining.

During the past half century mining methods have undergone some significant changes that have contributed to increasing the output per worker and to counteracting the growth in physical difficulties. It may be instructive to point out, before entering into a discussion of the improvements in methods, that selection of a basic mining method is largely conditioned by the nature of the deposit. Copper ores may be classified roughly into two general types - porphyry or disseminated, and vein or lode. Porphyries are large bodies of low-grade ores in which the copper minerals are disseminated relatively uniformly throughout the mass. Copper-vein deposits are tabular masses, relatively thin in comparison to their longitudinal extent, in which the copper minerals are contained principally in veins or fissures in the rock. Porphyry deposits which occur near the surface are mined by the open-cut method; those at greater depth are mined by underground-caving methods. Vein deposits, as a rule, are exploited by underground methods.

The introduction of the open-cut and undercut block-caving methods in the twentieth century for exploiting porphyry deposits has been the most important event in the history of copper mining. It has permitted the mining of large bodies of low-grade ores hitherto regarded as non-ores, and consequently has greatly augmented the copper reserves of the country.[4] As these two methods are more productive than any of the others,[5] the increase in the volume of production by open-cut mining and block-caving has been the most influential factor in raising the output per worker in the copper-mining industry. The modern method of open-cut mining with power shovels was borrowed from the iron-mining industry where it had been in use since the eighties.[6] The development of modern, undercut, block-caving methods, however, is one of the major contributions of the copper-mining industry.

Improvements in mining methods as applied to underground-vein mines have consisted mainly of modifying existing techniques or adopting the advanced practices developed in other mining industries. For example, mining methods have been altered so as to take fuller advantage of the force of gravity. In some systems gravity is relied upon to crush the ore and convey it to a point where it is loaded into cars. In others it is utilized to transport the broken ore to chutes or bins and to fill workings from which the ore has been removed with waste material. Another instance in point is the adoption of the retreating system of mining in place of the advancing system, whereby a considerable saving in maintenance labor and expense has been effected.[7]

Figure 2 shows the production of copper classified by principal mining methods. It will be noted that the output from vein deposits which were mined by stoping methods with natural and artificial supports[8] was on a declining trend since the peak years of the war period. On the other hand, the production from porphyry deposits which were exploited by the open-cut and block-caving methods was generally increasing until the onset of the depression following 1929, when it dropped sharply. With the

[4]See table A-18.

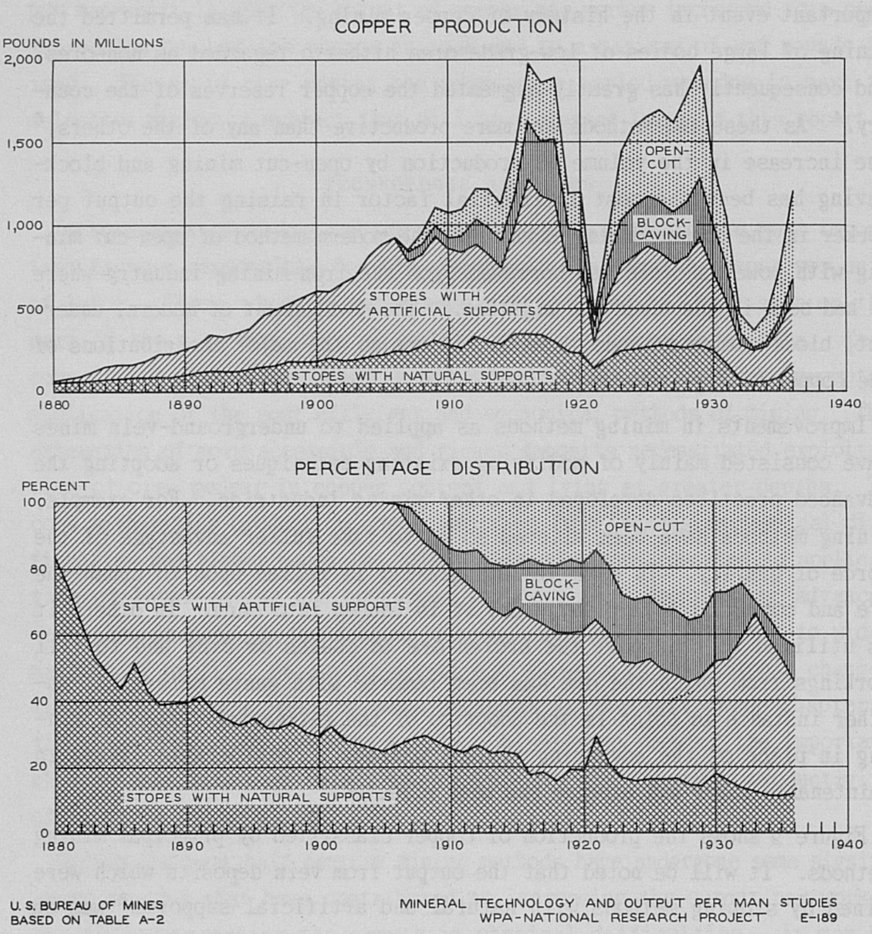
[5]For the period 1925-29 the average output of copper per worker per 8-hour shift is 319 pounds for open cuts and 188 pounds for block-caving, compared with 127 pounds for underground methods excluding block-caving. For the same period the average output of ore per 8-hour man-shift is 16.9 tons for open cuts and 9.5 tons for block-caving, compared with 2.2 tons for underground methods other than block-caving.

[6]For a discussion of the application of the open-cut method to iron mines see N. Yaworski, O. E. Kiessling, and Others, Technology, Employment, and Output per Man in Iron Mining (WPA National Research Project in cooperation with U. S. Department of the Interior, Bureau of Mines, in preparation).

[7]For a discussion of the advantages of the retreating system over the advancing system see chapter IV.

[8]A more detailed classification and discussion of underground mining methods will be found in chapter IV.

Figure 2.- COPPER PRODUCTION CLASSIFIED BY MINING METHOD, 1880-1936

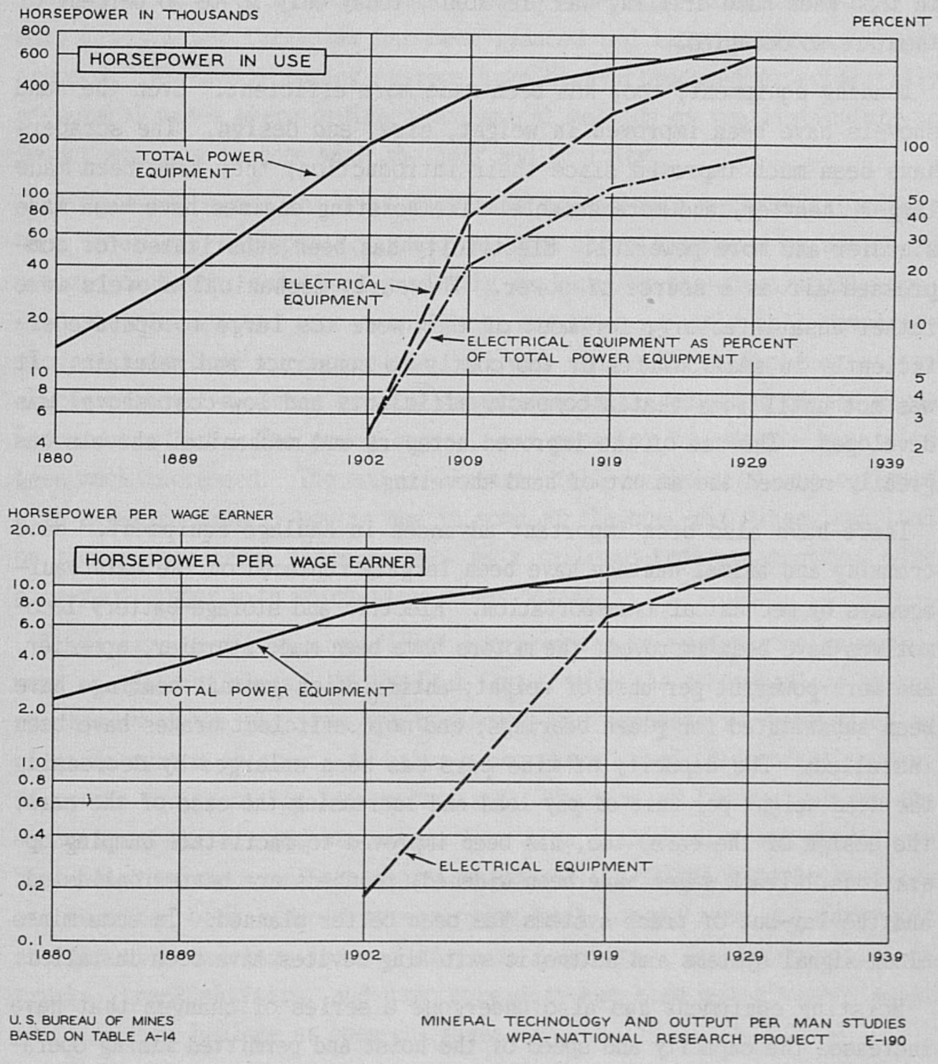


business revival in 1933, the output from this source was rising again, and by 1936 it contributed about half the primary copper production of this country. In this connection it is interesting to observe from figure 1 that the increase in the over-all output per worker was most rapid since the porphyry mines came into operation with the introduction of the two most productive methods of mining.

Technologic progress is reflected in the growth of mechanization. From figure 3 it will be seen that the total volume of mechanical equipment in use, as measured by the amount of horsepower in use in the copper-mining industry, increased from 13,500 in 1880 to 700,000 in 1929. The lower section of the chart shows the changes in the degree of mechanization as measured by the amount of horsepower in use per wage earner. In 1880 copper mines were mechanized to the extent of 2.2 horsepower per worker, and in 1929, 15.8 horsepower per wage earner - a more than sixfold increase. It is also interesting to observe the rapid growth in electrical equipment. In 1900 electrical machinery was practically nonexistent, whereas in 1929 there was 13.6 electric horsepower per wage earner. Were data available after 1929, they would undoubtedly reveal a further gain in electric horsepower per wage earner in the industry.

Figures showing an increase in the use of power equipment do not, of course, tell the whole story of technologic advances. Not only has there been rapid growth in mechanization during the past 50 years, but the efficiency of mining and ore-dressing equipment has been progressively improved.

Figure 3.- PROGRESS IN MECHANIZATION OF COPPER MINES, 1880-1929
(Ratio scale)



Drilling equipment, for example, has undergone some remarkable advances since 1880. The first piston drills, introduced in the eighties to replace drilling by hand, were heavy, clumsy, and slow. Frequently it cost as much to break a ton of rock by power drilling as it did by hand drilling. By reducing the weight, improving the construction, and increasing the speed of these machines, their efficiency was gradually advanced. Introduction of the stoper, drifter, and plugger types of drills in 1905-10 and the subsequent improvements of these machines have further increased the efficiency of drilling. In this connection mention should also be made of the substitution of dynamite and gelatins for black powder and the resultant improvement in efficiency and safety of blasting. As a result of these advances in drilling and blasting there has been a considerable reduction in labor requirements at underground mines. Whereas about two-thirds of the underground workers were engaged in breaking rock in 1880 when hand drilling was prevalent, today only 20 to 30 percent of them are so occupied.

Loading equipment, too, has been made more efficient. Even the hand shovels have been improved in weight, size, and design. The scrapers have been much improved since their introduction; they have been made larger, heavier, and more durable. The hoisting engines have been made sturdier and more powerful. Electricity has been substituted for compressed air as a source of power. The early mechanical shovels were rather unsatisfactory, for most of them were too large to operate efficiently in small drifts or too costly to construct and maintain. It was not until 1931 that a compact, efficient, and low-cost shovel was developed. The use of the improved scrapers and mechanical shovels has greatly reduced the amount of hand shoveling.

There have also been important advances in haulage equipment. Hand tramming and animal haulage have been largely replaced on the main haulageways by mechanical transportation. Electric and storage-battery locomotives have been improved; the motors have been made sturdier, speedier, and more powerful per unit of weight; antifriction journal bearings have been substituted for plain bearings; and more efficient brakes have been installed. The capacity of mine cars has been enlarged by decreasing the dead weight per unit of pay load and increasing the size of the cars; the design of the cars, too, has been improved to facilitate dumping operations. Track gages have been widened, roadbeds are better ballasted, and the lay-out of track systems has been better planned. In some mines block-signal systems and automatic switching devices have been installed.

Hoisting equipment has also undergone a series of changes that have increased the capacity and speed of the hoist and permitted mining operations to be carried to progressively lower levels. Animal power was displaced by steam power, which in turn was supplanted by compressed air or electricity. With the adoption of the Ilgner system and Ward-Leonard control, the efficiency of the electric hoist has been enhanced materially. Substituting the skip for the cage and thus obviating the necessity of hoisting the dead weight of the mine cars has increased the pay load of the hoist by as much as 40 percent. The use of lightweight materials for constructing the cage or skip has also lowered the dead weight and raised the capacity of the hoist.

With rising temperature and humidity, which accompany increasing depth of mines, it has become necessary to replace natural ventilation with mechanical ventilating systems. The installation of a series of electrically driven fans throughout the mine to force the air to the working places, with air courses and with provisions for controlling the direction of air currents, has so much improved working conditions that at many mines labor turn-over has been reduced and labor productivity increased. Air-conditioning systems have been introduced experimentally at several mines where ordinary mechanical methods of ventilation are no longer adequate to cope with the heat and humidity.

In open-cut mines the most important advances in mechanization have been in loading equipment. Railroad-type shovels powered by steam and mounted on trucks have been gradually replaced by the electrically powered shovels mounted on caterpillars, and more recently full-revolving shovels have been replacing the railroad type. The efficiency of the electric shovel has been greatly enhanced with the adoption of the Ward-Leonard control. The power, speed, and capacity of shovels have also been much increased. The modern electric full-revolving shovel equipped with a 5-yard dipper, now in use in some of the open-pit mines, can load on the average three times as much as a railroad-type steam shovel with a $3\frac{1}{2}$-yard dipper with which these mines began.

There have also been significant improvements in transportation equipment at open-cut mines. At the Utah Copper mine the steam-haulage system which served the pit from the beginning was completely replaced by an electric system in 1929. At other mines where steam transportation persists the equipment has been modernized. The size of locomotives and the capacity of the ore and waste cars have been increased, track conditions have been improved, and the starting and rolling friction of locomotives and cars has been reduced by equipping both locomotives and cars with antifriction bearings instead of plain bearings. The introduction of trucks, track shifters, and dump spreaders has also contributed to the efficiency of haulage at open-cut mines.

In the field of ore dressing improvements in gravity concentration and the development of the flotation process have made recovery of copper from low-grade ores commercially feasible and have, as a result, stimulated exploitation of vast porphyry deposits and multiplied the copper reserves of the Nation. Advances in crushing and grinding the ore have greatly enhanced the efficacy of both gravity concentration and flotation. The effect of the improvements in ore dressing may be observed from the fact that as late as 1900 only 65 percent of the copper was recovered, whereas at the present time well over 90 percent is obtained.

Mention has been made of the substitution of electricity for other forms of power in various branches of mining. The extent to which electrical equipment is replacing the machines powered by steam and other forms of energy is indicated in figure 3. In 1902 only 2.3 percent of the horsepower in use was electric, whereas in 1929, 86.2 percent was electric. More and more have underground mines come to depend on electricity as a source of power. The haulage systems, hoists, scrapers, pumps, air compressors, ventilation systems, and lighting systems have been extensively electrified. In open-cut mines, too, the use of electricity has been gaining ground. This increase in the application of electric energy has contributed immensely to the rising productivity in copper mining.

The general tendency in electrification has been to purchase electric energy rather than to generate it at the point of use. In the following tabulation the equipment at copper mines that was powered by purchased energy and the equipment that was driven by energy generated by the mining enterprises are enumerated separately:[9]

Year	Electric motors driven by purchased energy		Electric motors driven by energy generated by mining enterprises	
	Number	Horsepower	Number	Horsepower
1909	819	52,286	536	25,888
1919	3,647	135,968	3,252	161,024
1929	8,038	334,928	4,726	270,205
Percentage increase				
1909-19	345.3	160.0	506.7	522.0
1919-29	120.4	146.3	45.3	67.8

It may be seen that the trend in the period 1909-19 was in the direction of using electricity generated at the mine. The horsepower of the motors

[9] *Fifteenth Census of the United States: 1930*, "Mines and Quarries: 1929" (U. S. Dept. Com., Bur. Census, 1933), p. 301.

driven by electric energy so generated increased 522 percent, whereas that driven by purchased energy rose only 160 percent. In the period 1919-29 the situation was reversed. The horsepower of motors actuated by purchased energy increased 146 percent compared with a gain of only 68 percent for that of motors run by power generated at the mine. Complete figures after 1929 are not available, but fragmentary information indicates that the trend toward purchased energy has continued.

Many mining camps where fuel is costly or difficult to obtain have been brought within the range of profitable working through the utilization of hydroelectric power or energy generated by steam plants often situated a considerable distance from the mine. There are other reasons why many mining companies prefer to purchase energy rather than to generate it themselves. Some mines are in areas where power rates are low, and in such instances it is more desirable to save the cost of building a plant and eliminate the task of operating it. Moreover, where electricity is purchased it is usually generated by more than one station, hence the chance of shut-downs resulting from power failures is greatly reduced.

Progress in copper mining depends as much on managerial competence as it does on technologic efficiency. The rise in output per worker, sometimes ascribed solely to advances in technology, is certainly due in part to changes in methods of management. During the past half century mine management has achieved considerable success in increasing efficiency in mining and metallurgical technology through better planning, organization, and research. Greater care has also been taken to select and train workers for particular jobs. Bonus and contract systems have been inaugurated to induce workers to greater exertion. Working and living conditions have been made more attractive to promote contentment among employees, thereby reducing labor turn-over. All these have contributed to an increase of the productivity of labor.

PROGRESS IN SAFETY

Increasing attention has been devoted to the problem of accident prevention at copper mines. Several reasons may be ascribed to this. In the first place, extensive technologic changes and growing physical difficulties have multiplied the hazards of mining. In the second place, safety campaigns inaugurated by the United States Bureau of Mines have been bearing fruit. Finally, the enactment of workmen's-compensation laws has made accidents more costly to the operators. More and more operators have come to accept the view that prevention of accidents is profitable as well as humanitarian.

The more important steps taken to reduce accident rates in mines include training the workers in safety practices, thorough and frequent inspections of mine workings and equipment, the establishment of a safety organization at each mine, and observation of certain precautionary rules in blasting operations. Telephone and signal systems have been installed throughout mines to maintain contact with groups of workers at various sections. Protective apparel such as helmets, goggles, and special boots is worn by miners to prevent personal injury. Various measures for dust elimination have been adopted to lessen the danger of silicosis, and precautions have been taken against fire hazards.

Safety codes adopted by all copper-mining States make many of these safety measures compulsory. They provide for periodic inspections of mines by State safety inspectors, although in many instances the inspection staffs are handicapped in enforcing the code provisions effectively. However, as most of the mines are insured against accident risks, part of the burden of examination is carried by the insurance companies, whose inspectors exert considerable influence in improving the performance in mine safety practices.

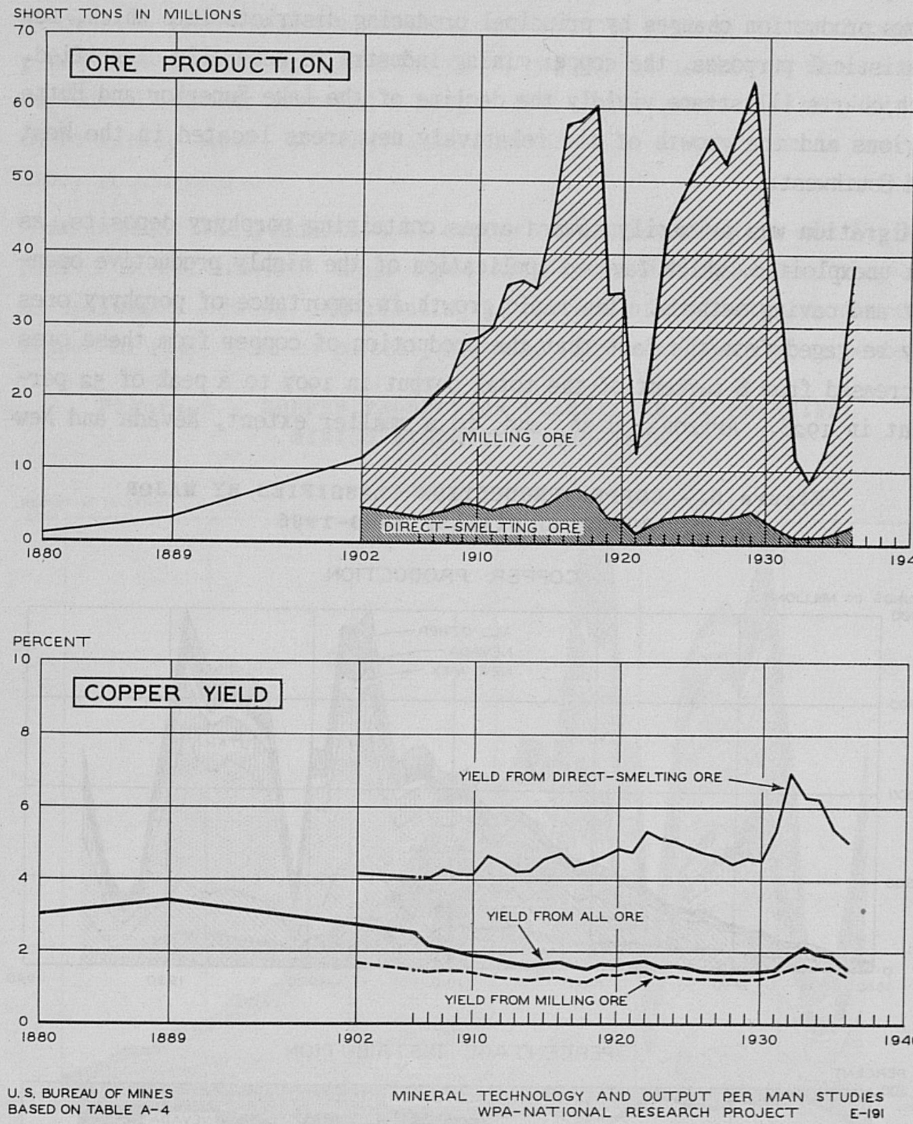
GEOGRAPHIC SHIFTS IN PRODUCTION

Factors other than technologic and managerial proficiency have contributed to increasing labor productivity. Among the more important of these are the shifts in centers of production. Progressive depletion of richer ores and steadily growing natural handicaps prompted the exploitation of deposits lying in newer and less developed areas that were made more accessible by the expansion of the railway network. Until about 1881 by far the greater part of copper was produced from the vein deposits of the Lake Superior region, which for many years had consistently accounted for over 80 percent of the total copper output of the United States. The next 25 years saw a rapid rise in production from the vein deposits in Montana, principally from the Butte district, which reached a high point of 314,750,620 pounds in 1905, a peak surpassed only in the war years 1916 and 1918. Since 1900 the vein deposits of Arizona have begun to play an important role as a source of copper.

In the eighties the greater proportion of copper was obtained from high-grade or direct-smelting ores.[10] This is indicated in figure 4. It may be seen that whereas the production of direct-smelting ores remained

[10] Direct-smelting ores are so rich in copper-bearing minerals that they go directly to a smelter instead of a concentrator. Ores having a lower copper content are sent to a mill or concentrator where the valuable minerals are separated from the gangue or waste. The concentrate, that is, the part containing the valuable minerals, is then shipped to a smelter.

Figure 4.- PRODUCTION AND YIELD OF DIRECT-SMELTING AND MILLING COPPER ORES, 1880-1936



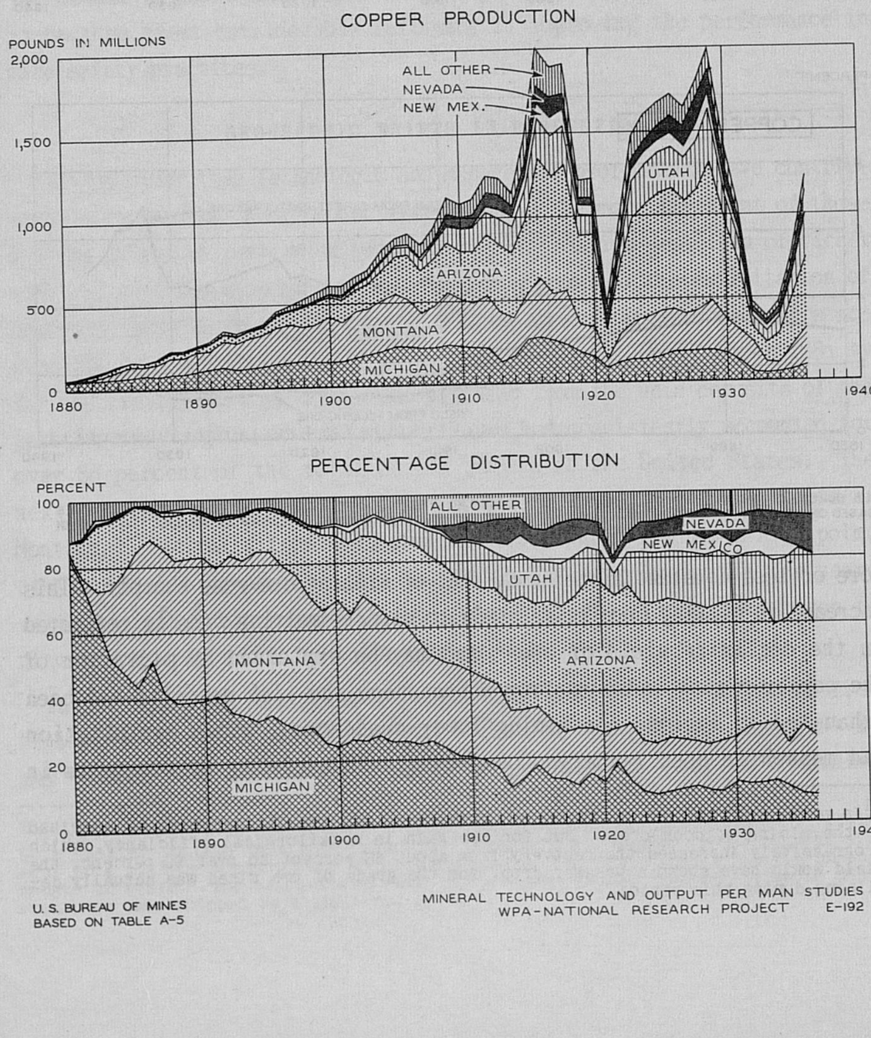
more or less constant since 1906, that of milling ore rose sharply. This increase in milling ores in relation to direct-smelting ores is reflected in the declining yield of ore[11] and may be explained by migration of the copper-mining industry from areas where the rich deposits had been exhausted to regions containing lower-grade ores, whose exploitation had meanwhile been rendered commercially feasible by improvements in

[11]In part, the low and declining yield in the last two decades must be ascribed to the mining of poor ores. But for the gain in metallurgical efficiency, which progressively increased the recovery from about 65 percent to over 90 percent, the yield would have shown a greater drop, for the grade of ore mined was actually declining during this period.

mining and metallurgical efficiency. Figure 5 shows the changes in copper production from 1880 to 1936 by major producing States; figure 6 shows production changes by principal producing districts into which, for statistical purposes, the copper-mining industry is generally classified. Both charts illustrate vividly the decline of the Lake Superior and Butte regions and the growth of the relatively new areas located in the West and Southwest.

Migration was primarily toward areas containing porphyry deposits, as yet unexploited, which favored application of the highly productive open-cut and caving methods. The rapid growth in importance of porphyry ores may be gaged from the fact that the production of copper from these ores increased from 8 percent of the total output in 1907 to a peak of 52 percent in 1928. Arizona, Utah, and, to a smaller extent, Nevada and New

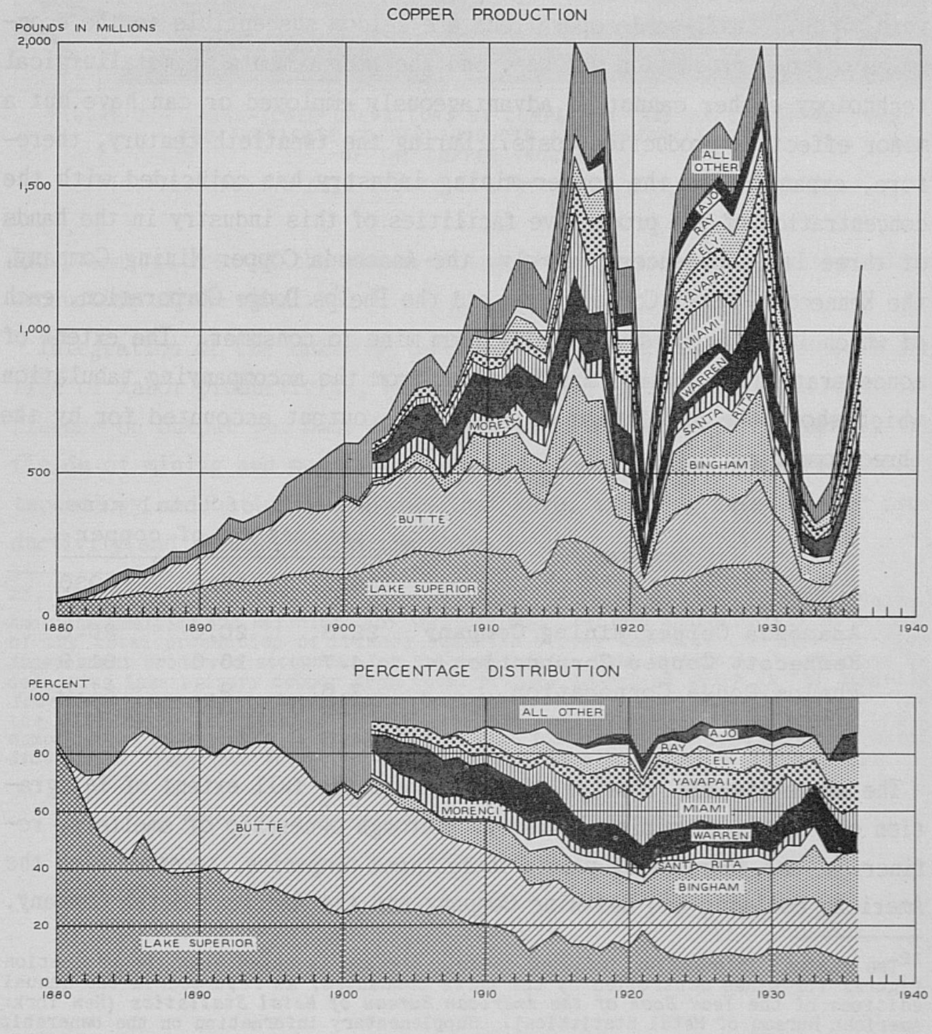
Figure 5.- COPPER PRODUCTION CLASSIFIED BY MAJOR PRODUCING STATES, 1880-1936



Mexico, produced the entire output from these low-grade disseminated deposits. From figure 5 it may be observed that the relative volume of production from these four States rose from 39 percent of total copper production in 1907 to a maximum of 71 percent in 1928. As these newly exploited regions have had a much higher labor productivity than the older mining areas, the shifts have resulted in a reduction in average labor requirements.

At present the production of copper is concentrated in six States. Ranked in the order of their importance as producing States, they are as follows: Arizona, contributing 34 percent of the total recoverable copper in 1936; Utah, 21 percent; Montana, 18 percent; Nevada, 11 percent;

Figure 6.- COPPER PRODUCTION CLASSIFIED BY PRINCIPAL MINING DISTRICT, 1880-1936



U. S. BUREAU OF MINES
BASED ON TABLE A-3

MINERAL TECHNOLOGY AND OUTPUT PER MAN STUDIES
WPA-NATIONAL RESEARCH PROJECT E-193

and Michigan, 8 percent. New Mexico, although a major producing State, accounted in 1936 for only 0.5 percent because the most important operation in the State, the Chino mine, was idle in that year. All other States and the territory of Alaska produced the remaining 7½ percent in 1936.

INTEGRATION OF THE INDUSTRY[12]

Rising output and the development of porphyry mines by mass-production methods were accompanied by the concentration of the industry in the hands of a progressively smaller number of producers. Steadily expanding demand for copper called for large-scale operations requiring huge capital outlays that the small producers could not afford. Furthermore, improved mining methods and other advances have been of much greater benefit to large producers of low-grade ores than to operators of small vein mines. Small-scale operations are seldom susceptible to the economies of mass-production methods, and the improvements in metallurgical technology either cannot be advantageously employed or can have but a minor effect on production costs. During the twentieth century, therefore, expansion of the copper-mining industry has coincided with the concentration of the productive facilities of this industry in the hands of three large producers, namely, the Anaconda Copper Mining Company, the Kennecott Copper Corporation, and the Phelps Dodge Corporation, each of which is integrated vertically from mine to consumer. The extent of concentration of control may be noted from the accompanying tabulation which shows the percentage of total mine output accounted for by the three companies:

Company	Percent of total mine production of copper		
	1922	1929	1936
Anaconda Copper Mining Company	22.3	20.6	20.3
Kennecott Copper Corporation	4.7	16.6	31.9
Phelps Dodge Corporation	7.6	9.1	21.8
Total	34.6	46.3	74.0

The above figures, however, do not reveal the full extent of integration of the copper-mining industry. Through ownership of smelters, refineries, and fabricating plants these three companies, together with the American Smelting and Refining Company and the American Metal Company,

[12]The factual information included in this section is based upon the production figures for mines controlled by the three companies, as reported in the annual editions of the *Year Book of the American Bureau of Metal Statistics* (New York: American Bureau of Metal Statistics). Supplementary information on the ownership of the individual mines was obtained from the companies' annual reports to stockholders and *Poor's Industrial Volume* (New York: Poor's Publishing Co.).

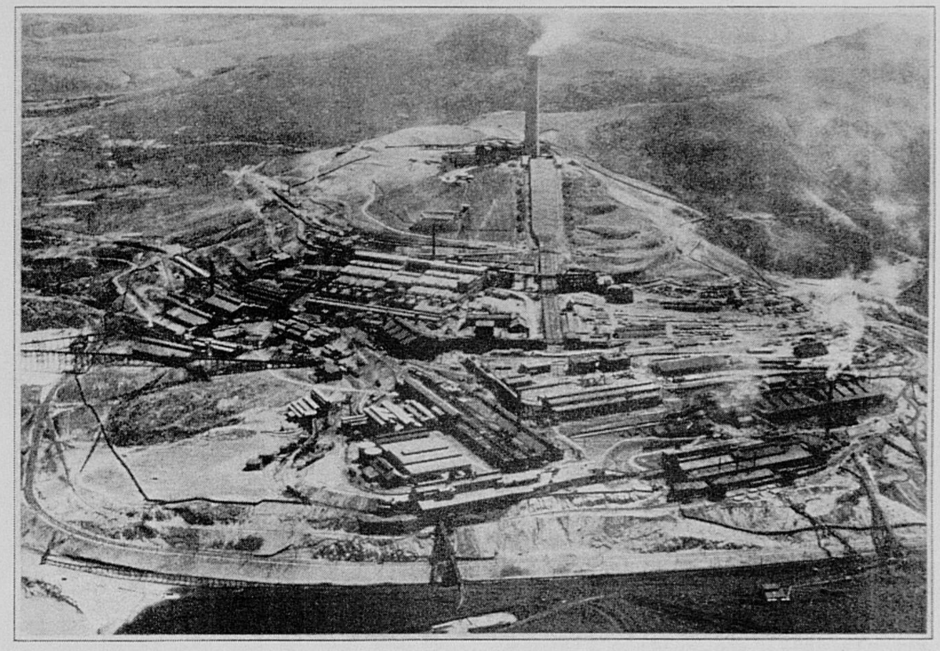


FIGURE 7.- LARGE-SCALE OPERATIONS BY COMPANIES THAT SMELT COPPER ORES AND FABRICATE COPPER PRODUCTS ARE CHARACTERISTIC OF THE COPPER INDUSTRY

This view shows a huge reduction plant that works Montana ores.

Ltd., control most of the copper-mining industry and virtually the entire production of the metal.[13]

Integration of the industry, although largely a passive factor in the rise of labor productivity, has nevertheless exerted a significant influence on output per man, for it has facilitated research work in the fields of mining and metallurgy and the adaptation of a more efficient technology and better techniques that have, in turn, raised labor productivity and reduced production costs.

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[13] It is estimated that in 1936 the refineries of Anaconda Copper, Phelps Dodge, American Smelting and Refining, and American Metal refined approximately 93 percent of the total production of primary domestic copper and that the Lake Superior independent producers accounted for the remainder. This conclusion is obtained by deducting Lake primary copper from total primary copper as given in the *Minerals Yearbook, 1937* (U. S. Dept. Int., Bur. Mines, 1937), p. 153, and considering the difference as produced by smelters and refineries of the aforementioned companies at such plants as are listed in *Year Book of the American Bureau of Metal Statistics: 1935* (New York: American Bureau of Metal Statistics, 1936), p. 22.

CHAPTER III

OPEN-CUT MINING[1]

The development of open-cut operations has marked a new era in the history of copper mining in the United States. From about the middle of the past century, when copper production of this country had first become of world importance, until about 1907 copper ore had been extracted by underground methods almost exclusively. At the beginning of the present century improvements in the methods of treating copper ores had made feasible commercial exploitation of the vast deposits of low-grade porphyry ores by modern methods of open-cut mining. Although a man with helmet and carbide or electric cap lamp still typifies the miner underground, his economic importance, so far as copper mining is concerned, has been gradually overshadowed by the surface worker in overalls who commands a power shovel.

The introduction of open-cut mining, with its mass methods and labor-saving devices, has had two major effects on the copper-mining industry: it has decreased the cost of producing copper, notwithstanding the low metal content of the porphyry ores, and it has reduced materially the labor requirements per pound of copper produced. Indeed, the open-cut method has been the most instrumental factor in the rapid increase in the productivity of labor in copper mining.

The growth of open-cut mining has affected the productivity of labor in two ways. As the open-cut is the most productive method, the steady increase in the proportion of the total output mined by open-cut operations has tended to raise the over-all output per man. Being particularly well adapted to mass production, this method is especially susceptible to technologic improvements that are most efficacious in labor saving. Moreover, increasing natural difficulties, which have had an adverse effect on the output per worker in underground mines, have been relatively insignificant at open-cut mines.

The output per man at open-cut mines is several times greater than that in underground mines.[2] As the labor cost constitutes a high proportion of total mining cost, the application of the open-cut method results in

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[1]By Y. S. Leong; Emil Erdreich; J. C. Burritt, Mining Engineer, C. E. Nighman, Consulting Mining Engineer, and George C. Heikes, Consulting Engineer, WPA National Research Project.

[2]For the period 1923-31 the output of ore per man-hour at open-cut mines was 4.7 times as large as that at underground mines; the output of copper per man-hour at open-cut mines was 2.3 times as large as that at underground mines. Compare figures 11 and 21.

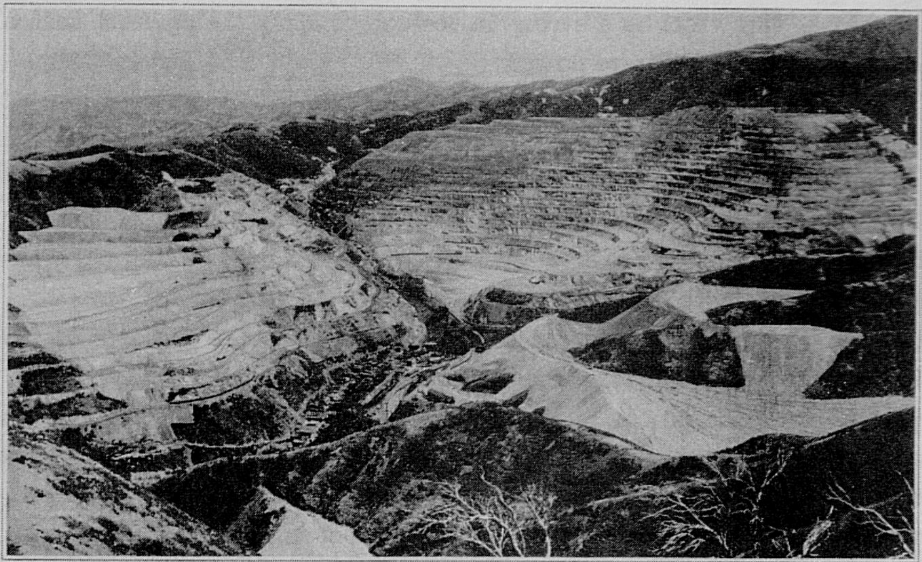

FIGURE 8.- OPEN-CUT COPPER MINE AT BINGHAM CANYON, UTAH

This is the largest copper mine in the United States. The mining of a tremendous tonnage of ore and waste has left a large amphitheater-like pit, the sides of which are giant steps averaging 70 feet in height and from 50 to 250 feet in width. Additional mining operations are carried on that are not shown in this view.

a considerable reduction in the cost of producing copper. The direct mining cost by the open-cut method, based on the 1926-30 average, is 2.26 cents per pound of recoverable copper compared with 4.10 cents by caving methods and 4.53 cents for all underground mines.[3] It is obvious,

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[3]The direct mining cost included in the computation embraces the following items: Labor in development and mining, materials and supplies, immediate supervision, and miscellaneous underground and surface expenses applicable to mining. These cost figures are for the 5-year period 1926-30 for most operations and for shorter representative periods within this period for all other operations, with the cost per pound weighted by the quantity produced at each mine. The mines included in the computation are as follows: Open-cut mines - Utah, Ruth Pit of the Nevada Consolidated, Chino of the Nevada Consolidated, and New Cornelia; block-caving mines - Miami, Inspiration, Ray of Nevada Consolidated, and Morenci of Phelps Dodge; other underground mines - Copper Queen, United Verde Extension, Verde Central, Walker, Engels, Beatson, Isle Royal, Burra Burra of Tennessee Copper, Mary of Ducktown Chemical, Mines B, C, and E at Michigan, Calumet and Arizona, and Magma.

Data on ore and copper are from reports submitted to the Bureau of Mines, and the cost of mining is from the following sources: B. Van Presley, "The Latouche System of Mining as Developed at the Beatson Mine, Kennecott Copper Corporation, Latouche, Alaska," *Transactions of the American Institute of Mining and Metallurgical Engineers*, Vol. 76 (1928), pp. 11-55; M. J. Elsing, "Cost of Surface Metal Mining," *Engineering and Mining Journal*, Vol. 134, No. 1 (Jan. 1933), pp. 17-21; M. J. Elsing, "Cost of Mining by Caving," *Engineering and Mining Journal*, Vol. 133, No. 11 (Nov. 1932), pp. 573-5; E. D. Gardner, C. H. Johnson, and B. S. Butler, *Copper Mining in North America* (U. S. Dept. Int., Bur. Mines, Bull. No. 405, 1938), p. 265; R. L. D'Arcy, *Mining Practice and Methods at the United Verde Extension Mining Company, Jerome, Ariz.* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6250, mimeo., Feb. 1930), 11 pp.; C. F. Jackson, *Summary of Ore-Mining Cost Data* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6785, mimeo., June 1934), 47 pp.; W. R. Crane, *Mining Methods and Practice in the Michigan Copper Mines* (U. S. Dept. Com., Bur. Mines, Bull. No. 306, 1929), 192 pp.; C. H. McNaughton, *Mining Methods of the Tennessee Copper Company, Ducktown, Tenn.* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6149, mimeo., June 1929), 17 pp.; V. L. Kegler, *Mining Methods of the Ducktown Chemical and Iron Co., Mary Mine, Isabella, Tenn.* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6397, mimeo., Feb. 1931), 9 pp.; and H. M. Lavender, *Mining Methods of the Campbell Mine of the Calumet & Arizona Mining Co., Warren, Ariz.* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6289, mimeo., Apr. 1930), 18 pp.

therefore, that there is a strong inducement to apply the open-cut method wherever it is feasible to do so.

DESCRIPTION OF OPERATIONS

Open-cut mining is best adapted for exploitation of ore bodies of large lateral extent that occur near the surface, that is, where the depth of overburden seldom exceeds 200 or 300 feet. This type of operation has been confined largely to the mining of porphyry ores that contain 1 to 2 percent copper, the sole major exception being the vein deposits of the United Verde mine which has employed both surface and underground methods. Because the grade of ore is low, porphyry deposits are not profitable to mine unless immense tonnages can be handled. In most underground vein mines an output of 1,000 tons of ore per day is considered high, whereas in open-cut porphyry mines a daily production of 10,000 to 50,000 tons is not unusual.[4]

There is relatively little variation in the general method of surface mining. The overlying waste or capping is stripped and ore is then mined by power shovels. Mining operations are usually conducted simultaneously on several levels or benches that range from 25 to about 80 feet in height and from 50 to 350 feet in width. The number of benches varies at different mines, depending largely on the topography. Where the topography is flat, as at the New Cornelia mine at Ajo, Arizona, the benches are rather wide and the required tonnage can be mined from three or four working terraces. On the other hand, at the Utah Copper mine at Bingham Canyon, Utah, where the surface slopes rather steeply, 22 relatively narrow benches are used.

Not only the waste or capping directly over the ore must be removed, but also that beyond the limits of the ore body at the edges of the pit, to permit the mining of boundary ore and to prevent the sides from sliding into the pit. That is, as the depth of the pit is increased, it must also be widened. The over-all slope ratio is generally about 1 or $1\frac{1}{2}$ to 1. That is, to increase the depth by 1 foot, it would be necessary to widen the pit by 1 or $1\frac{1}{2}$ feet. In time the volume of overburden that must be removed in relation to the volume of ore that can be mined may become so large, and the length of hauls and the track grade so much increased, as to make open-pit operations uneconomical. When this stage is reached, open-pit mining is discontinued and the remaining ore is recovered by underground methods.

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[4]The Utah Copper mine, for example, produced, in 1937, 23,000,000 tons of ore at an average rate of 64,000 tons per day.

Before the material, consisting of ore and waste, can be excavated and loaded by power shovels, it must be loosened and shattered into fragments small enough to be convenient for handling. This is accomplished by drilling and blasting the rock from the face of the bank. Subsequently boulders that are too large for the shovel dipper to handle are reduced in size by "secondary" drilling and blasting. The material is then loaded by power shovels and conveyed in cars on standard-gage railroad tracks to a concentrator or disposal dump. At one of the mines - United Verde - motortrucks are used instead of railroad cars to transport the ore to the ore passes.[5]

Long spirals or switchbacks are required at all surface mines. The length of tracks is an important factor in determining labor requirements at open-cut mines. Long trackage increases the time consumed in transporting the material and requires more rolling stock and additional labor to maintain, extend, and shift the tracks and trolley lines. Mining by the open-cut method thus consists of three major operations: breaking, loading, and transporting the ore.

Among the main advantages of surface mining are low labor requirements, safety, and flexibility of operations. The relatively small labor requirements of open-cut mining are due primarily to large-scale methods and greater adaptability to mechanical loading and transportation. The advantages of the open-cut over the underground methods from the standpoint of safety are obvious. Falls of roof and many other hazards of underground mining are absent and outdoor work under daylight conditions is not only conducive to safety but may also be more healthful. In times of fluctuating demand surface mines afford particular advantages because of the ease of varying the total output. Flexibility of open-cut operations makes it possible to reduce materially the volume of production or even to suspend operations entirely without incurring the high shut-down expense that attends maintenance of underground mines.

Among the factors that may have an adverse effect on open-cut mining are the necessity of removing the overburden, length of the development period, the large amount of capital tied up in development work, cost of dumping waste, and weather conditions. The amount of stripping that must be done before production can begin varies in different mines, depending on the thickness of the capping, the shape of the deposit, the topography, and the nature of the rock. Generally the time required to prepare approaches to the mine and to advance the stripping to a stage

[5]At the Clay ore body in Morenci, which is being developed for open-pit mining, motortrucks are utilized to convey the overburden to waste dumps.

where the ore can be excavated is much longer than that needed for preliminary development by underground methods. The capital required for such long-time and large-scale development work, too, must necessarily be greater. Moreover, dump space is needed for depositing the large volume of waste, and when none is available near a mine, mining cost is materially increased by the necessity of hauling waste over a longer distance. Finally, surface mining, unlike underground mining, is affected by weather conditions. Severe conditions may interfere with the haulage of material and compel the mine to cease operation. These adverse factors tend to offset some of the advantages of the open-cut method.

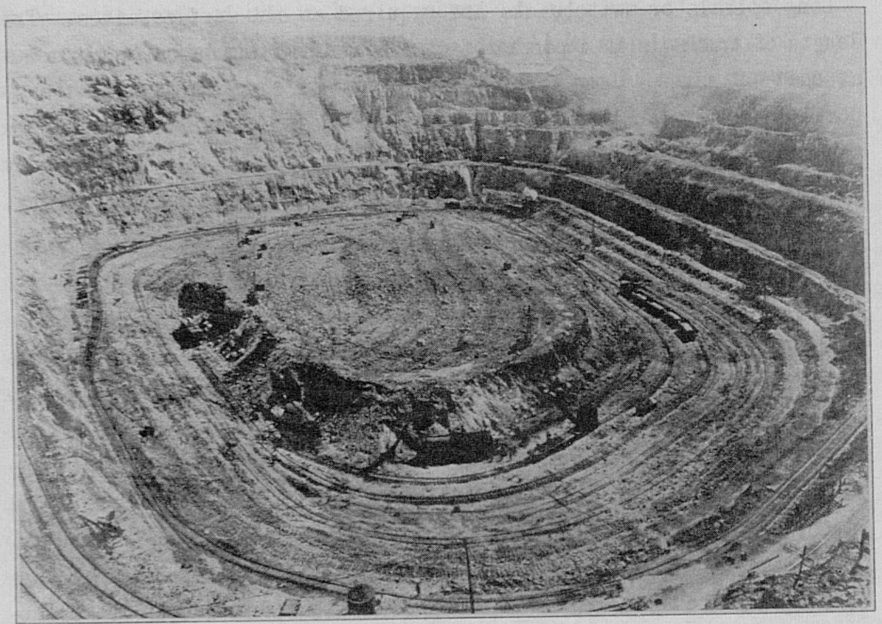

FIGURE 9.- A SHIFT FROM SURFACE TO UNDERGROUND MINING IS SOMETIMES ADVISABLE

Hampered by increasing hazards from rock slides and other difficulties, surface mining operations at the Sacramento Hill mine shown above, which had moved approximately 30,000,000 tons of ore and waste, were suspended in the latter part of 1929. The remaining ore was mined by underground methods.

GROWTH OF OPEN-CUT MINING

Surface mining of copper ores in the United States is a comparatively recent practice, having been introduced at the Utah Copper mine at Bingham Canyon in the latter part of 1906.[6] Open-cut mining was adopted shortly afterward by the Nevada Consolidated Copper Company at Ely, Nevada, where

[6]The low-grade porphyry deposits at Bingham and other areas had been known to exist for many years. When the mining of these ore bodies was proposed, many of the best mining engineers were rather skeptical about exploiting at a profit any deposit containing only 40 pounds of copper per ton of ore, even at the high price of 17 or 18 cents per pound. See, for example, the editorial comments in the *Engineering and Mining Journal*, Vol. LXVII, No. 21 (May 27, 1899), p. 614.

up to 1914 nearly the entire production, and since then about 80 percent, came from open-cut operations. By the end of 1910 these two mines were producing 24 percent of the total output of ore, representing 14 percent of the total recoverable copper in the United States. In 1911 the Chino mine at Santa Rita, New Mexico, began to mine ore by the open-cut method. These three mines have dominated copper production in their respective States to such an extent that the average output per worker for each State has not differed materially from the output per man at the respective open-cut mines. The fourth mine to use surface methods was New Cornelia at Ajo, Arizona. Although some preliminary operations had been carried on at this mine for several years previously, large-scale production did not begin until 1917. These four mines are the only open-cut mines now in operation that are likely to continue surface mining for any length of time in the future. It is probable that the Clay mine, now being developed for open-pit operations, may come into large-scale production in the near future.[7]

During 1922-31 two other mines, Copper Queen at Bisbee, Arizona, and United Verde at Jerome, Arizona, were operated partly as open-cut mines. Surface operations at Copper Queen were entirely discontinued early in 1930 and at United Verde in 1931. Although open-cut mining was resumed at United Verde in 1935, it is unlikely that it will continue for over 4 or 5 more years because of increasing depth of excavation.

Growth in Production

The growth of surface mining is shown graphically in figure 10. The major spurts in production by this method occurred in 1923 and in the post-depression years 1933-36. In the first instance the beginning of large-scale open-cut operations at the Copper Queen and United Verde mines was at least partly responsible for the increase in the volume of copper from open-cut mining from 19 percent of the total output of copper in 1922 to 29 percent in 1923. Low prices, falling demand, and accumulation of stocks during the depression[8] discouraged production from high-cost underground mines, thus increasing the relative share from surface operations. The output from open-cut mines consequently increased from 24 percent of the total copper output in 1932 to 44 percent in 1936. Even greater was the increase in output of ore; from 46 percent in 1932, production from surface operations increased to 62 percent of the total output of copper ore in 1936.

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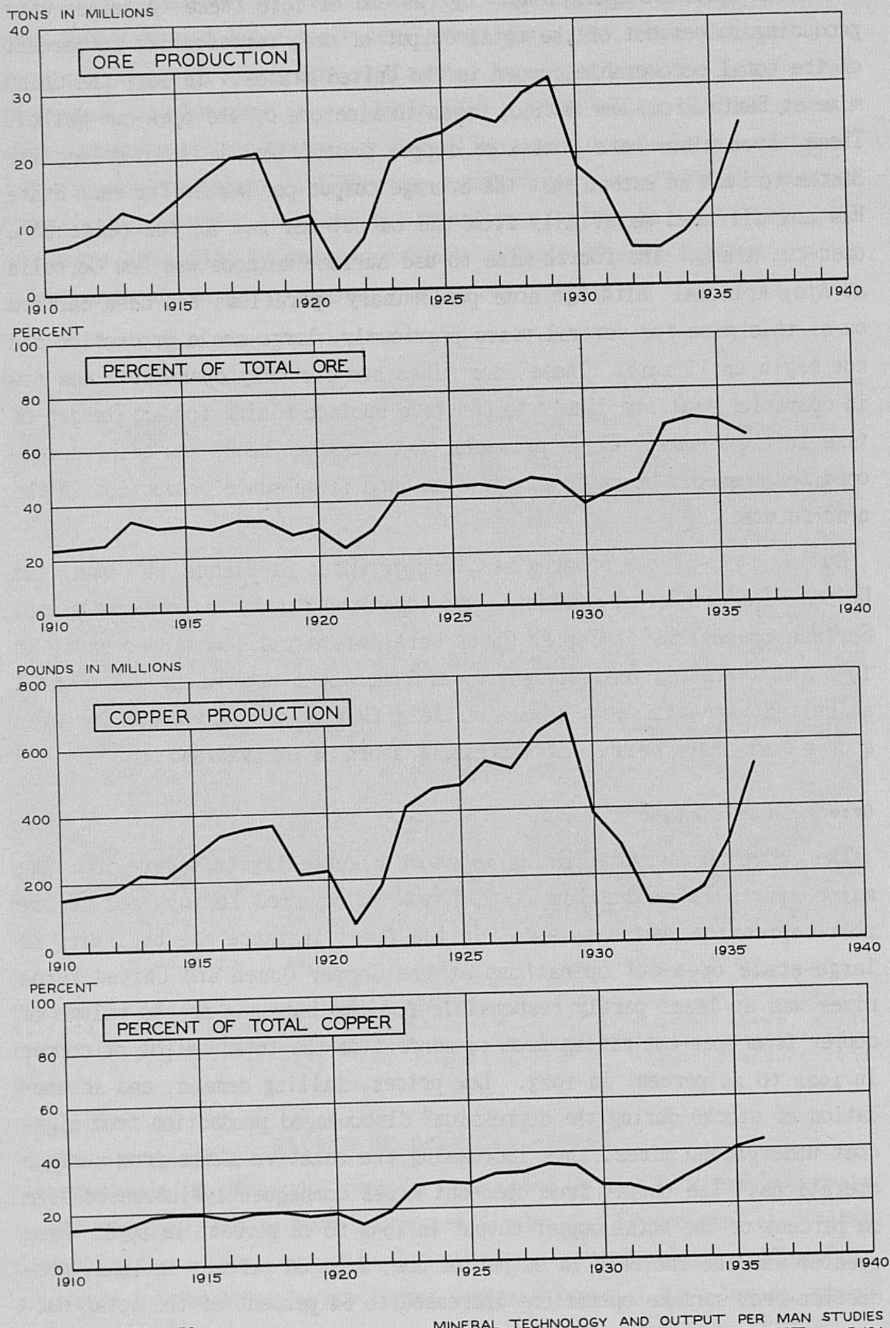
[7]See Phelps Dodge Corporation, *Annual Report: 1938*, p. 7.

[8]The price of copper declined to an all-time low in 1932, while the stock of copper reached an unprecedented peak in 1933.

Figure 10.- GROWTH OF OPEN-CUT COPPER MINING, 1910-36



U. S. BUREAU OF MINES
BASED ON TABLE A-2

MINERAL TECHNOLOGY AND OUTPUT PER MAN STUDIES
WPA-NATIONAL RESEARCH PROJECT E-194

This relative gain in output may be partly attributed to the flexibility of open-cut mining and inflexibility of underground mining. In general the underground mines, after a partial or complete shut-down in the early thirties, required a considerable time to get back to a production basis. On the other hand the open-pit mines began to operate or to increase their output as soon as the demand for copper improved.

Changes in Employment

As a source of employment, open-cut mines are far less important than underground mines. This is indicated in the accompanying tabulation showing the percentage that the employment at four open-cut mines, Utah, Chino, New Cornelia, and Copper Flat, comprised of total employment at all mines:[9]

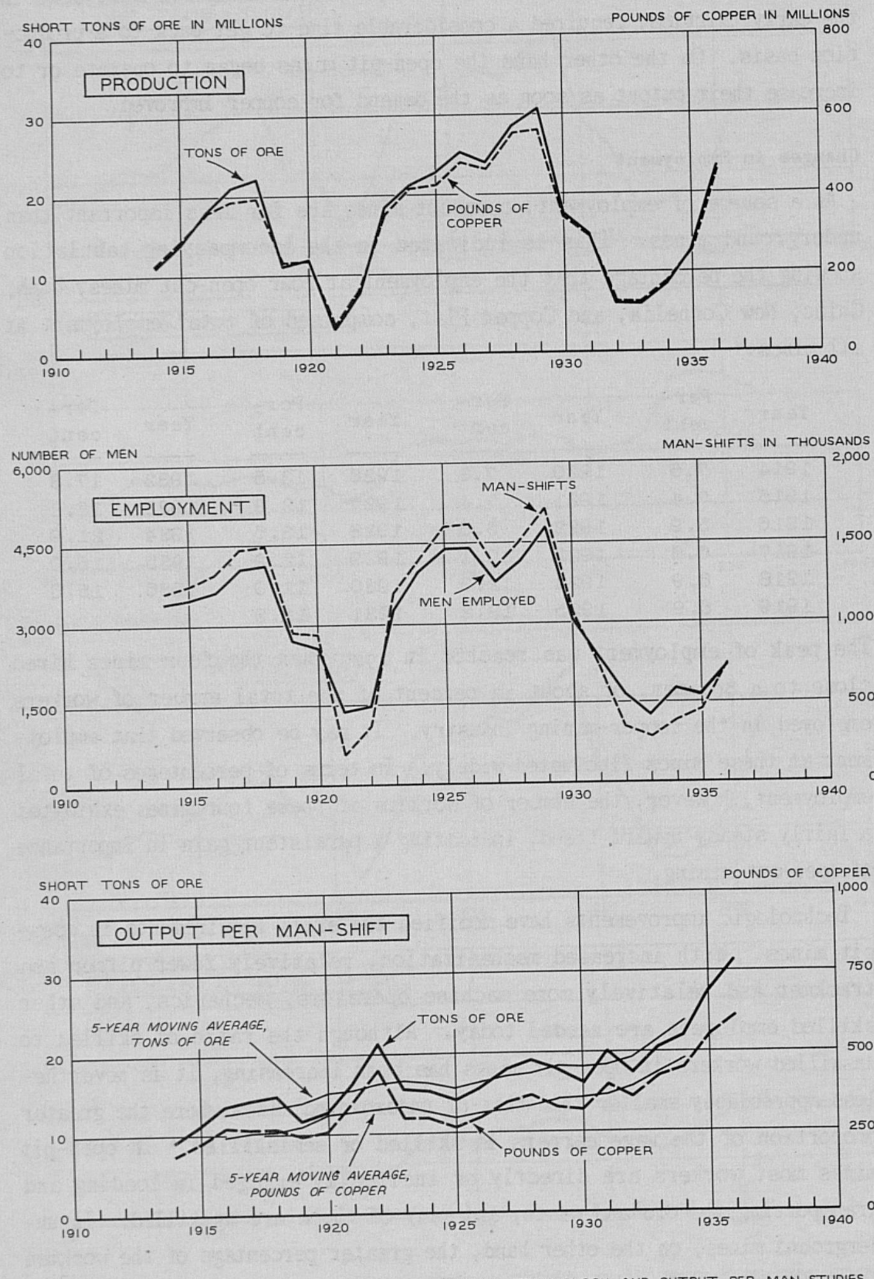
Year	Percent	Year	Percent	Year	Percent	Year	Percent
1914	7.6	1920	7.3	1926	13.5	1932	17.3
1915	7.4	1921	7.4	1927	12.3	1933	18.2
1916	5.9	1922	5.8	1928	13.6	1934	21.9
1917	6.6	1923	10.1	1929	12.9	1935	16.0
1918	6.9	1924	12.3	1930	11.0	1936	15.6
1919	6.9	1925	13.2	1931	11.9		

The peak of employment was reached in 1929 when the four mines hired close to 4,800 men, or about 13 percent of the total number of workers employed in the copper-mining industry. It may be observed that employment at these mines fluctuated widely. In terms of percentages of total employment, however, the number of workers at these four mines exhibited a fairly steady upward trend, indicating a persistent gain in importance of open-cut mining.

Technologic improvements have modified the skill requirements in open-pit mines. With increased mechanization, relatively fewer pitmen and trackmen and relatively more machine operators, mechanics, and other skilled employees are needed today. Although the ratio of skilled to unskilled workers in open-pit mines has been increasing, it is nevertheless appreciably smaller than that of underground mines where the greater proportion of the wage earners is skilled or semiskilled. In open-pit mines most workers are directly or indirectly engaged in loading and transporting the ore and waste, and many of these are unskilled. In underground mines, on the other hand, the greater percentage of the workmen is occupied with breaking ground and with timbering, and most of these

[9] Based on tables A-1 and A-7. It was not possible to segregate with reasonable accuracy the workers engaged in open-cut operations from those who were mining underground at the other two mines, Copper Queen and United Verde, that operated partially as open-cuts.

Figure 11.- PRODUCTION, EMPLOYMENT, AND OUTPUT PER WORKER AT OPEN-CUT COPPER MINES, 1914-36



U. S. BUREAU OF MINES
BASED ON TABLE A-9

MINERAL TECHNOLOGY AND OUTPUT PER MAN STUDIES
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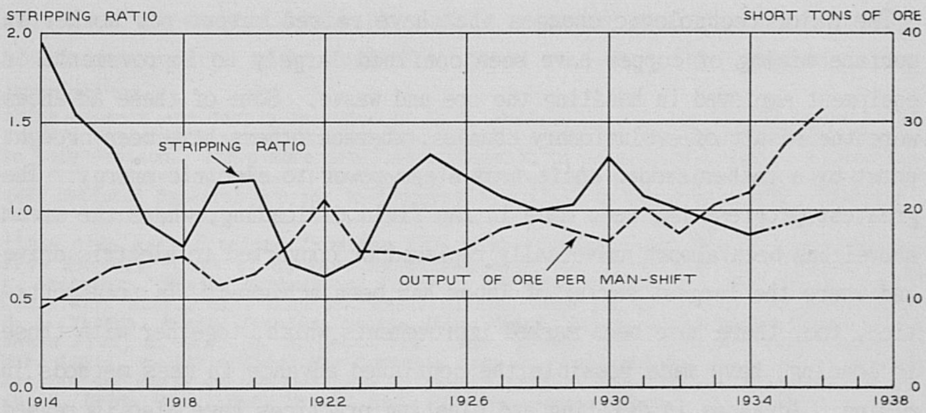
employees are skilled. It may be expected that as the open-pit mines become further mechanized the proportion of skilled to unskilled workers will increase.

Rise in the Output per Worker

The output per worker, in terms of ore and of copper, at the four open-cut mines is shown in figure 11 for the period 1914-36. Although the output per worker of both ore and copper varied appreciably, the trends of the two series were definitely upward. The 5-year moving averages superimposed on the respective output-per-man curves indicate what the trends of labor productivity might have been had the effect of purely temporary causes, such as fluctuations in business conditions, been eliminated and the development work spread over a period of years. This growth in labor productivity, in the absence of any other important factor that might explain it, must be attributed almost entirely to technologic improvements and better planning and supervision of activities at open-cut mines. The year-to-year oscillations of output per worker are due partly to variations in the volume of capping removed per ton of ore mined (that is, the "stripping ratio"), partly to several factors that were ascribable to business fluctuations, and partly to changes in the proportion of the output contributed by each of the four mines.

The relationship between the stripping ratio and the output of ore per man-shift is illustrated in figure 12. It may be observed that on the whole there is an inverse correlation between the two series; that is to say, when the stripping ratio rises the output per unit of labor generally declines, and vice versa.

Figure 12.- STRIPPING RATIO AND OUTPUT PER WORKER AT OPEN-CUT COPPER MINES, 1914-36



U. S. BUREAU OF MINES
BASED ON TABLE A-9

MINERAL TECHNOLOGY AND OUTPUT PER MAN STUDIES
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Changes in the proportion of output by each of the four mines under discussion have undoubtedly been responsible for some of the fluctuations in output per worker. As these mines differ considerably in their productivity, any variation in their relative volume of production is likely to cause deviations in the level of over-all labor productivity.

In times of business recession the number of days during which a mine is kept active is generally reduced and mine capacity is only partly utilized. In such periods two opposing groups of influences come into play, one tending to raise output per man-hour and the other tending to diminish it. The first consists of a relatively higher labor productivity, attained either through retention of only the more efficient workers or by placing greater emphasis generally on the individual performance. The workers themselves may be inclined to work harder. There is also a strong likelihood that mining operations are confined to sections where the ore can be extracted with a relatively smaller labor input or where a better grade of ore can be readily mined. Furthermore, development work often is reduced to a minimum to decrease production costs. The group of influences that tends to depress output per worker is also attributable to several factors. When the mines are operated at a fraction of their capacity, relatively much overhead labor (for maintenance, a skeleton force for the operation of basic equipment, etc.) must be retained. Another factor in the situation is that some mining companies may utilize their surplus labor force to do development work that otherwise would have to be done later, thus lowering the output per worker. The precise influence of each of the individual factors cannot be isolated or ascertained, but the over-all effect in most instances has been to raise the output per worker in depression years.

TECHNOLOGIC ADVANCES

The major technologic changes that have raised output per worker in surface mining of copper have been confined largely to improvements in equipment employed in handling the ore and waste. Some of these advances were the result of evolutionary changes, whereas others have been brought about by a rather sudden shift from steam power to electric energy. The greatest progress has been made in the field of loading, where the steam shovel has been almost universally replaced or converted to electric drive and where the largest saving of labor has been achieved. In transportation, too, there have been marked improvements which, together with those in loading, have made possible the continued advance in mass methods in mining. Advances in drilling and blasting practices have also increased the productivity of mine labor and, although they are perhaps of less significance in open-cut mines than in underground operations, they nevertheless have reduced the labor requirements and the cost of mining copper ore.

Drilling and Blasting[10]

In the open-cut mining of copper, drilling and blasting account for a substantial proportion of the total mining costs. The following tabulation, which gives examples of costs of drilling and blasting and the total direct-mining costs for some of the principal open-cut mines, shows that the average cost of drilling and blasting represents over 20 percent of the total direct-mining costs:[11]

Mine	Year	Cents per ton of ore: Cost of drilling and blasting	Cents per ton of ore: Total direct mining cost	Cents per ton of ore: Percent that drilling and blasting is of total
Nevada Consolidated	1923	4.2	21.4	19.6
Utah Copper	1928	2.7	11.7	23.1
Chino	1929	4.8	20.7	23.2
New Cornelia	1930	7.0	21.0	33.3

Figures showing the amount of labor expended in drilling and blasting operations are not available, but it seems probable that the labor cost comprises roughly 25 to 50 percent of the total cost of drilling and blasting. In the period preceding that covered by the tabulation, drilling techniques were in process of development and relative costs of drilling were doubtlessly considerably higher.

Drilling.- The drilling operation has three major functions: in prospecting, to delineate the ore deposit and to determine the grade of ore and the thickness of capping and ore; in primary drilling, for blasting rock from the face of the bank; and in secondary drilling, for breaking boulders too large to be handled by power shovels. Two types of drills

[10] The authors are deeply indebted to H. P. Sweeny, Consulting Mining Engineer of the WPA National Research Project, who supplied much of the material incorporated in this section. For a more detailed discussion on this subject see C. E. Nighman and O. E. Kiessling, *Mineral Technology and Output per Man Studies: Rock Drilling* (WPA National Research Project in cooperation with U. S. Department of the Interior, Bureau of Mines, Report No. E-11, Feb. 1940).

[11] The data were obtained from the following sources: for Nevada Consolidated from Elsing, "Cost of Surface Metal Mining," pp. 17-8; for Utah Copper from A. Soderberg, *Mining Methods and Costs at the Utah Copper Co., Utah* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6234, mimeo., Feb. 1930), p. 22; for Chino from H. A. Thorne, *Mining Practice at the Chino Mines, Nevada Consolidated Copper Co., Santa Rita, N. Mex.* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6412, mimeo., Mar. 1931), p. 27; and for New Cornelia from G. R. Ingham and A. T. Barr, *Mining Methods and Costs at the New Cornelia Branch, Phelps Dodge Corporation, Ajo, Ariz.* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6666, mimeo., Oct. 1932), p. 16. Costs for the New Cornelia mine are based on total material handled and assumed similar costs for ore and waste.

are used in prospecting - the diamond drill and the churn drill. The diamond drill, which is actuated by steam, by internal-combustion or compressed-air engines, or by electrical motors, and which can bore a hole at any angle, produces a core that shows the rock formations throughout the depth of the hole and provides a sample for assay. This type of drill is adequate for drilling holes up to 5,000 feet or more in depth. The churn drill, which is powered by steam, or internal-combustion engines or electric motors, and which drills only vertical holes, does not produce a core and is therefore not well suited for determining the rock structure. It is useful for procuring samples of homogeneous ore deposits; it is therefore employed rather widely in prospecting porphyry ores. The sample obtained by using churn drills is much larger than that obtained by means of diamond drills. The diameter of the hole may be as large as 20 inches, whereas diamond-drill holes are seldom more than 3 inches in diameter. Large samples are more adequate for assaying, for they are more representative of the mass of the deposit.

In primary blasthole drilling, three types of drills have been in use - the churn drill, the piston drill, and the hammer drill, the last two being actuated by compressed air. The hammer-type drifter is a vast improvement over the piston machine, which it began to displace rapidly at about 1924;[12] however, the piston drill is still used at some of the mines.[13] Like the piston machine, the drifter may be mounted on a tripod. The wagon-mounted drifter, which is also in use, is more efficient than the tripod-mounted machine, having a greater mobility and a higher drilling speed than the latter. The churn drill is used for drilling vertical holes generally 6 to 9 inches in diameter and from 25 to 100 feet in depth. For shallower holes either one of the other machines may be utilized. For holes other than the vertical ones the hammer-type drill is employed.

For secondary drilling, the hand-held jackhammer is generally used. This drill is characterized by its light weight and is operated by one man. Like the drifter type, which is a much heavier machine, it consists essentially of a reciprocating piston which delivers a rapid succession of light blows to the shank end of the drill steel, held loosely in the machine. It is also actuated by compressed air. The development of this

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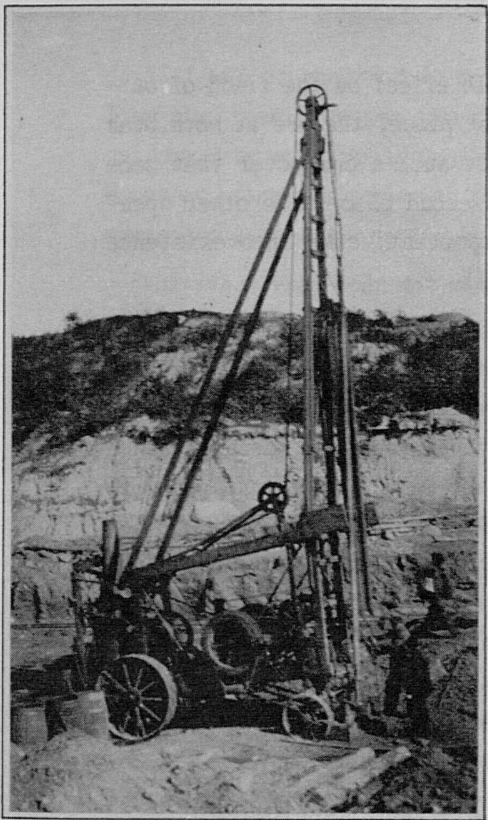
[12] F. E. Cash and M. W. von Bernewitz, *Methods, Costs, and Safety in Stripping and Mining Coal, Copper Ore, Iron Ore, Bauxite, and Pebble Phosphate* (U. S. Dept. Com., Bur. Mines, Bull. No. 298, 1929), pp. 164-5.

[13] The piston drill is still widely used at the Utah Copper mine, where it has been found to be the most suitable type for drilling the desired 22-foot "toe" holes. Modern hammer drills have been tried repeatedly without success, mainly because the water lines required to keep the sludge from the point of the drill bit become frozen during the winter. Soderberg, *op. cit.*, p. 12.



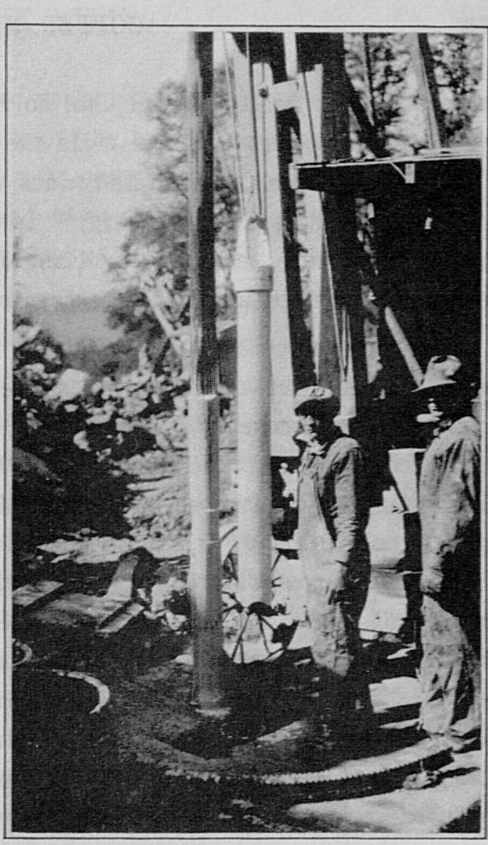


FIGURE 13.– DRILLING AT OPEN-CUT COPPER MINES

The upper left view shows an old-type steam churn drill which was widely used for prospecting porphyry copper ores as well as for blasthole drilling. The upper right view shows some of the tools used in churn drilling; the bailer (or sand pump) which removes the cuttings from the hole is at the right of the bit in the center of the picture. The lower illustration shows a tripod-mounted wet hammer-type drifter drill used for drilling toe holes at a southwestern surface copper mine.

type of hammer drill, however, had no visible effect on the trend of output per man at open-cut mines. In the first place, the ore at both Utah Copper and Nevada Consolidated mines was of such a character that secondary drilling was unnecessary;[14] in the second place, the other open-cut mines, where secondary drilling is important, came into existence after the development of the jackhammer drill.

No major improvements have been made in either the diamond drill or the piston drill since the beginning of open-cut copper mining. The more recent machines, however, are speedier, sturdier, and more economical in power consumption. The hammer drill has undergone some advances since its introduction at open-cut mines. Earlier models were equipped with weak steel rotating mechanisms that impaired their utility. It was not until a more powerful rotation was substituted that the hammer drill began to displace the piston type. The speed and mobility of the drill have been increased with the substitution of a wagon mounting for the tripods. The use of the modern wagon-mounted drill in place of the old-style drill mounted on a tripod has increased the speed of drilling at the New Cornelia mine by about 10 percent and at the United Verde mine by 50 percent.[15]

The churn drill has been greatly improved. The electric drive has been substituted for steam and internal combustion; wire rope has replaced manila rope; the cutting bits have been better designed and made more durable; the size and power of the drill and the length and speed of the stroke have been increased; and the mobility of the drill has been greatly improved with the utilization of caterpillar treads that permit the machine to move rapidly from hole to hole and from bench to bench under its own power. The introduction of the large-diameter churn drill, in particular, has enhanced the efficiency of drilling at open-cut mines.[16] This machine can bore 9-inch or even larger holes at virtually the same speed as other drills cutting 6-inch holes. Its high efficiency is due to the heavy string of tools it carries, the greater distance the tools travel during each stroke, and the higher speed of each stroke. The shock absorber with which it is equipped enables the drill to impart the sharper, quicker blow required to shatter and penetrate the rock more effectively. Larger holes make it possible to space them farther apart and hence to

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[14] E. D. Gardner, *Drilling and Blasting in Open-Cut Copper Mines* (U. S. Dept. Com., Bur. Mines, Bull. No. 273, 1927), p. 16.

[15] Gardner, Johnson, and Butler, *op. cit.*, p. 134.

[16] At the Copper Flat mine, where four new electric churn drills replaced nine old-style steam machines, it was found that the drilling speed of the former was about 2½ times that of the latter (*ibid.*). It should be pointed out that the old steam drill required two or more men to run it, whereas the new electric machine calls for only one, with one helper serving several drills.

drill fewer of them and to move and set up the machine less frequently. They also permit better distribution of the powder and consequently more uniform fragmentation with a minimum of oversize and secondary blasting. Moreover, they are easier to load and do not need to be sprung. All this means a considerable saving in labor and other costs.

Blasting.- When open-cut mines first began to operate, practically all of the present types of explosives were already available.[17] However, some improvements have been made which render the explosives less hazardous to handle and use. When dynamites freeze they become highly sensitive and therefore unsafe. Because the earlier types of dynamites were inclined to freeze at relatively high temperatures, serious accidents frequently resulted from their use. By adding nitro-substitution compounds it has been possible to lower the freezing points of these explosives and thus to minimize the danger of accidental explosions. There have also been some improvements in the adoption of types, sizes, and grades of explosives that have been found by experiments to be particularly efficacious under conditions existing at each mine. Moreover, a better balance has been maintained between the amount of drilling required and the amount of explosive used, a good part of which may, in turn, be ascribed to better planning and supervision.[18]

In the modern open-pit mines the ground to be broken is carefully studied by the engineering staff to determine the amount and kind of drilling and blasting required. The workers engaged in drilling and blasting operations are closely supervised; the size, depth, and spacing of the holes and the amount and kind of explosives to be used are prescribed. The important role played by the manufacturers of explosives in improving the efficiency of blasting should not be overlooked. Working in cooperation with the mine operators, they have made explosives available in a wide range of strengths, sizes, and densities suitable for specific purposes. Moreover, the manufacturers have accomplished much in increasing the certainty of detonation and in reducing the volume of noxious gases produced by explosion.

In considering the advances achieved in blasting, mention should be made of the development of the Cordeau detonating fuse. Effective blasting at open-cut mines requires that a large quantity of powder and numerous holes be exploded simultaneously. The Cordeau, which consists of a lead tube filled with trinitrotoluene, commonly known as TNT, is well adapted

[17] Two types of explosives have since been developed - liquid oxygen and starch dynamites. The former has not been utilized in open-cut copper mines. Although the latter were tried at some of the open-pit mines, their use was discontinued after a number of serious accidents. See Gardner, *op. cit.*, p. 40.

[18] See section on "Planning" later in this chapter.

to blasting operations at open-pit mines, for it detonates so rapidly that all charges connected to it explode almost instantaneously.

The effect of the advances in blasting on productivity of labor in drilling and blasting operations has been small. The combined influence of the increase in efficiency of power drilling, the improvements in the selection and application of explosives, and better planning and supervision is, however, fairly large, although, because of the lack of statistical data, it is not possible to state it quantitatively.

Loading

The most significant improvements in mechanization, insofar as the increase in the output per worker and reduction in mining costs are concerned, have been in loading machinery. The railroad-type shovels,[19] powered by steam, were practically the only loading machines in the open-cut copper mines of this country until about 1922, when electrically powered shovels mounted on caterpillar treads were introduced. The earlier railroad-type shovel required tracks on which to advance as it dug its way along the bench. A shovel crew of three men, an operator, fireman, and craneman, was needed to run the shovel, and a pit crew of four to six men had to be on hand to lay short sections of track ahead of the shovel, shift the track, release the jack-arms, set the rail clamps, and reset the jack-arms each time the machine was moved. Hours or even days were lost in transferring the shovel from one working place to another. Moreover, the old railroad-type shovel could dig in only one direction; it could not turn, and therefore it had to back up and start over for each new cut. With the adoption of caterpillar traction the necessity for laying the shovel tracks was eliminated and the ground crew was reduced by at least half. Equally as important as the direct saving in labor was the increase in mobility which permitted the shovel to advance or retreat at a moment's notice. In the event of threatened slides or cave-ins of the bank above, it can quickly back out of danger. With a shovel on rails, however, it requires from 10 to 15 minutes to move it even a short distance; this is more than enough time for a slide to bury it.

Electric shovels were available at the time the open-cut copper mines were commencing large-scale operations, but the operators did not feel that these machines were reliable enough to warrant their installation. Even when they were developed to a point where their efficiency was equal to that of the steam machines, their general adoption was retarded for

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[19]The original railroad-type shovel may be described as a machine whose power units are mounted on a car frame structure. The boom is mounted on the "swing circle", by means of which the boom, stick, and dipper revolve through an arc of 180 to 240 degrees.

two reasons. In the first place, the operators were cautious, believing in playing safe with the steam equipment that had given them dependable service for many years. In the second place, they had by that time made considerable outlays for the steam equipment which was still in usable condition. It was not until 1917 that one of the American copper companies installed an electric shovel in its South American open-pit mine.[20] This machine was equipped with a 4-yard dipper and a.-c. motors and was mounted on tracks, which were later discarded in favor of caterpillar traction. Although it had not reached the level of development of modern shovels, its performance was nevertheless an improvement over its steam competitors. From then on electric shovels gradually began to replace the steam machines at this mine.[21] In 1922 the first electric shovel was introduced in one of the open-cut mines in the United States.[22] This shovel was mounted on three caterpillar tractors and was equipped with a $4\frac{1}{2}$-yard dipper. It proved to be so much more efficient than the steam-driven type that other purchases were soon made, and the machines that were already in service were converted from steam to electric drive. The other open-cut mines have also begun to acquire electric shovels.

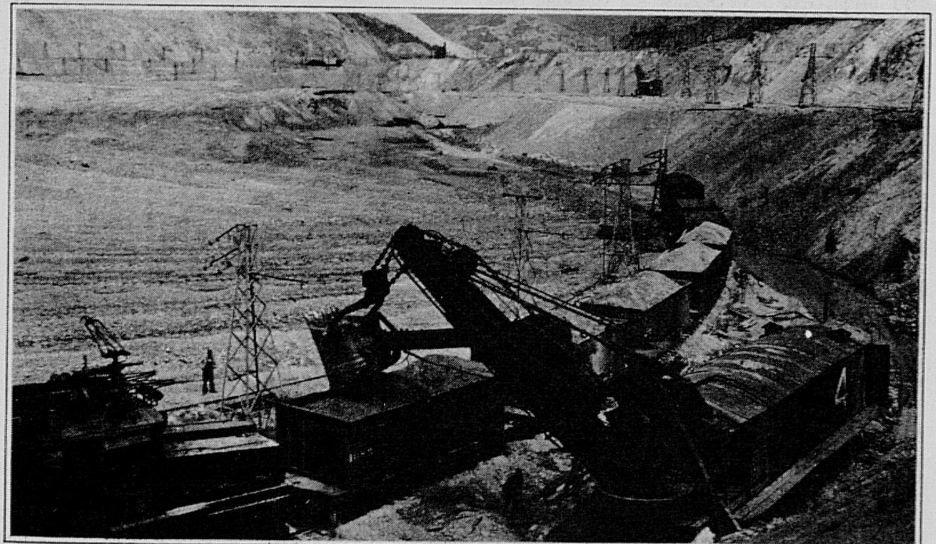

FIGURE 14.- A REMODELED SHOVEL LOADING ORE AT AN OPEN-CUT COPPER MINE

This is a railroad-type shovel that has been in service for more than a quarter of a century. Originally powered by steam, this shovel has been improved by electrification and by the addition of caterpillar trucks. The remodeled shovel, however, is less flexible in operation and has a smaller loading radius than the modern full-revolving shovel.

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[20]Chile Copper Company's mine at Chuquicamata, Chile.

[21]A. B. Parsons, *The Porphyry Coppers* (1st ed.; New York: The American Institute of Mining and Metallurgical Engineers, 1933), p. 382.

[22]The first copper mine in the United States to use an electric shovel was that of the Utah Copper Company. See C. W. Corfield and R. J. Corfield, "Electric Shovel Operation at Utah Copper Mine," *The Mining Congress Journal*, Vol. 13, No. 9 (Sept. 1927), p. 674.

With the adoption of shovels powered by electricity and mounted on caterpillar tractors a further economy in labor was achieved. The application of electric energy has dispensed with the services of the fireman and the labor engaged in delivering fuel and water to the shovel.[23] But apart from reducing the labor requirements directly, the electrically operated shovel has many advantages over the steam-driven machine. It minimizes the problem of water supply, which is of considerable importance at mines where good boiler-feed water cannot be had, or where it is difficult to keep the water mains open in severe winter weather. Its operation is quieter, steadier, faster, and cleaner. It excavates and loads more per unit of time than the steam shovel, for its power supply does not fluctuate as does steam. The fact that it does not emit smoke or steam makes it safer not only for the shovel crew but for other workers in the pit. The heightened visibility permits the crew to operate trains at faster speed with greater safety. It also has a longer life; because its moving parts are mostly rotary rather than reciprocating, the wear and tear on them are less than on those of steam machines. The outstanding advantage of the electric shovel, however, is its low operating and maintenance costs in comparison with those of steam excavators. It does not, for example, consume power when it is standing by, whereas a steam shovel consumes steam even when it is idling.

That the electric shovel is more economical than the steam shovel is evident from the following tabulation, which shows the cost of handling a ton of material by steam and electric machines at a mine which excavates about 10,000,000 tons of rock per year:[24]

Cost item	Cents per ton			
	Steam shovels	Electric shovels	Saving	Percent saving is of steam-shovel cost
Operating labor	2.186	1.411	0.775	35.5
Repair labor	0.364	0.289	0.075	20.6
Material	2.402	0.609	1.793	74.6
Shop orders	2.188	1.496	0.692	31.6
Miscellaneous	0.046	0	0.046	100.0
Fuel or power	4.091	0.272	3.819	93.4
Total	11.277	4.077	7.200	63.8

[23] For example, at the Chino mine of the Nevada Consolidated Copper Company the crew of a steam shovel equipped with railroad trucks consists of an engineer, a crane-man, a fireman, and six pitmen; that of a standard electric shovel is composed of an engineer, a craneman, an oiler, and a pitman (Thorne, *op. cit.*, p. 23). At the Utah Copper mine the crew of an electric machine is made up of an engineer, a craneman, a pitman, and an electrician who serves two or three shovels (Soderberg, *op. cit.*, p. 15). The reduction in the number of pitmen is attributable in [*Con.*]

Footnote 24 appears on following page.

The difference between the cost of loading a ton of material by steam shovels and that by electric shovels is 7.2 cents - a substantial saving indeed.

Virtually the same results were obtained at other open-cut mines where both steam and electric shovels are employed. For example, it may be observed in the following tabulation that at the Chino mine in 1929 it cost 7.6 cents to load a cubic yard[25] of material by electric shovels and 18.2 cents by steam machines:[26]

Cost item	Cents per cubic yard			
	Steam shovels	Electric shovels	Saving	Percent saving is of steam-shovel cost
Operating labor	6.172	2.583	3.589	58.1
Repair labor	1.367	0.774	0.593	43.4
Materials	3.799	2.021	1.778	46.8
Shovel track and general overhauling	2.436	1.137	1.299	53.3
Fuel or power	4.464	1.121	3.343	74.9
Total	18.238	7.636	10.602	58.1

By utilizing electric instead of steam shovels, this mine saved 10.6 cents per cubic yard of materials excavated, or 58 percent of the cost

23 [*Con.*] part to the use of caterpillar tractors and in part to the change from steam to electric power.

The application of electricity to power shovels has not only done away with the labor required to transport the fuel to the mine, thereby reducing the labor cost, but also has lowered the haulage costs by avoiding any interference by coal trains on switchbacks or on benches with ore and waste trains.

24 Compiled from W. Jurden, "Development and Application of Energy in the Mining and Metallurgical Treatment of Copper Ore," *Transactions of the World Power Conference, Sectional Meeting, Scandinavia, 1933* (Stockholm, Sweden: Kungl. Boktryckeriet. P. A. Norstedt & Söner, 1934), V, 150. Although the name of the mine for which the cost figures were computed is not revealed, it seems probable that they pertain to one of the South American mines. The figures under the column caption "Saving" represent the difference between steam-shovel costs and electric-shovel costs.

25 A cubic yard of material is equivalent to approximately 2 tons in weight. See Gardner, Johnson, and Butler, *op. cit.*, p. 122.

26 The figures presented in the tabulation are compiled from Thorne, *op. cit.*, pp. 27-8. The figures under the column caption "Saving" represent the difference between steam-shovel costs and electric-shovel costs. Thorne does not say whether all the shovels used were equipped with caterpillar traction, although Parsons is of the opinion that all were so equipped (Parsons, *op. cit.*, p. 385). At the time the paper was written (1931), the Chino mine possessed the following shovel equipment powered by steam and electricity: One Marion Model 350 electric, full-revolving shovel mounted on trucks; seven Marion electric Model 92 shovels mounted on caterpillars; three Marion steam Model 92 shovels mounted on caterpillars; one Marion steam Model 92 mounted on railroad trucks; and one Model 30-B Bucyrus Diesel shovel mounted on caterpillars. All Model 92 shovels were equipped with 4-yard dippers and the Model 350 with 8-yard dippers. The steam shovel mounted on railroad trucks was reserved for special work in soft wet ground or on switchback tracks where continual moving from one level to another was required (Thorne, *op. cit.*, pp. 22-3). If the shovel equipment enumerated above was in use in 1929, it would seem that the superiority of the electric machines, as indicated by the lower-cost figures, was due to a large extent to the application of electricity, for all shovels, with the exception of one electric and one steam, were mounted on caterpillars. The use of the electric shovel Model 350, which moved 1.7 times as many cubic yards per shift as the average loaded by the Model 92 electric shovels, would of course tend to increase the over-all efficiency of the electric shovels.

for steam shovels. It is interesting to note that the greater efficiency of the electric shovels is also reflected in the cost of haulage. For an average haul of 2.7 miles it cost 21.8 cents when the material was loaded by electric shovels and 27.7 cents when loaded by steam shovels.[27] This saving is due mainly to the fact that the electric machines loaded material more regularly and rapidly, thus facilitating the movement of traffic.

The experience of the Utah Copper mine may also be cited. As indicated in the accompanying tabulation showing the relative cost factors for the Utah Copper mine, the cost of loading a ton of material by electric shovels was 37 percent of that by steam shovels, or a saving of 63 percent as a result of the use of electric instead of steam machines:[28]

Cost item	Cost per ton	
	Steam shovels	Electric shovels
Operating labor	100	76
Operating materials	100	20
Coal and water, electric power	100	18
Packing and oil	100	34
Repairs	100	22
Total	100	37

It is significant that in every instance the cost items for electric shovels are lower than those for steam shovels. The great saving in labor cost was due largely to reduction in labor requirements brought about by the application of electric drive to power-shovel operations. The lower maintenance cost for electric shovels is explained in part by the fact that delays caused by choked boilers, leaky steam pipes, and frozen water lines are avoided and in part because of less wear and breakage of moving parts, as already noted. The enormous saving in power cost is largely to be attributed to the fact that no current is consumed when the electric shovel is not in actual operation, whereas steam must be maintained whether the steam shovel is active or temporarily idle as, for

[27]Thorne, *op. cit.*, pp. 27-8.

[28]Compiled from Corfield and Corfield, *op. cit.*, p. 679. It is not clear whether both the steam and electric shovels compared were mounted on caterpillar traction. The Corfields stated that about half the savings was attributable to electrification and the other half to the substitution of caterpillar tractors for railway trucks. Apparently, from this statement, the electric shovels were equipped with caterpillar traction and the steam shovels were not. Goodrich, who made a similar comparison of costs at Utah, reported practically the same results: a saving of 23.7 percent of the cost of operating labor and 79.6 percent of operating material. See H. C. Goodrich, "Shovel Operations at Bingham, Utah Copper Co.," *Transactions of the American Institute of Mining and Metallurgical Engineers*, Vol. LXXII (1925), p. 583. However, he indicated that both electric and steam shovels were similarly equipped with tractors (*ibid.*, p. 582; also pp. 569-72 for the list of shovel equipment).

instance, at night or on Sundays and holidays. To some extent it is due to the relatively high cost for fuel, as in the case first cited.[29]

Many important improvements in power shovels have occurred since the introduction of the electric shovel. These, like the application of electric energy and the adoption of caterpillar traction, have increased the output per worker and counteracted some of the difficulties of the growing depth of pits and the exhaustion of the richer ores. For these benefits credit should be accorded to the manufacturers of power shovels and of electric equipment who, working in cooperation with the mine operators and their engineering staffs, have been supplying the copper-mining industry with ever more efficient shovels that can meet the special and severe requirements of mine service.

One of these significant advances was the adoption of the full-revolving feature of power shovels. Only one man is required to operate a full-revolving shovel mounted on a single caterpillar base and powered by electricity, whereas two persons are needed to run the railroad type - the engineer to work the swinging and hoisting mechanisms and the crane-man to operate the crowding engine and to dump the load. The labor of the craneman is eliminated in this advanced type of shovel. As all the power and digging elements revolve on an underframe around a central axis, the engineer is always facing the dipper and can dig and deliver material at any angle from the digging face.[30] He does not need a craneman to signal to him. Not only does he alone run the shovel, but he runs it at a higher speed than he and a craneman could do with a railroad-type shovel.

The application of the Ward-Leonard control to electric shovels has been another important forward step in mining copper by the open-pit method. This system had been in use for several years in other mine services, particularly in underground hoisting, before its adoption by electric shovels. It consists of a separate generator for each of the direct-current motors employed in hoisting, swinging, and crowding the loading

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[29] The cost of fuel oil was exceptionally high, being 10 cents per gallon, and that of electrical energy was 0.7 cents per kilowatt-hour. The combined effect of the economical use of power by the electric shovel and the high cost of fuel oil was a saving of over 90 percent. In a year this saving amounted to $720,000, or 36 percent of the investment cost of $2,000,000. As the equipment installed was capable of handling 15,000,000 tons yearly instead of the actual 10,000,000 tons, the savings (had the equipment been worked at capacity) would have amounted to about $1,000,000, or, roughly, 50 percent of the investment cost. (See Jurden, *op. cit.*, p. 151.)

[30] In a full-revolving shovel the boom is fixed to the frame, which forms the base for the engines or motors and other machinery. The frame, which rotates on several large rollers, revolves in a full circle in either direction. This type differs from the railroad-type in that the frame revolves in the former through 360 degrees, whereas only the boom swings in the latter and only through an arc of 180 to 240 degrees. The less flexible railroad-type shovel cannot, for example, deliver boulders and other materials directly behind the shovel. The full-revolving shovel is also better balanced, being so designed that it does not need jack-arms to keep it from overturning, as does the railroad-type shovel.

mechanism, with the speed of each motor regulated by the generator voltage.[31] The flexibility, speed, power, and economy of the electric shovel have been materially advanced by the introduction of this innovation.[32]

The motors that drive the shovel and supply the power have been continually improved. Equipped with dustproof bearings and made more compact, more rugged, and more powerful, they can withstand severe service and high temperatures and require but little space and a minimum of repairs. Control has been simplified by the use of push buttons and magnetic devices that increase the ease as well as the security and speed of operations.

The shovel itself, apart from its power units, has been greatly improved. The weight of the dipper, for example, has been reduced by the use of alloy steel instead of cast steel.[33] Because steel alloys are considerably stronger than cast steel in withstanding certain stresses, lighter sections can be used without sacrifice of strength. This has permitted an increase in dipper capacity without adding to the power requirements. Thus, by replacing a 4-cubic-yard dipper which weighs 15,150 pounds and handles 12,300 pounds of material with a 5-cubic-yard dipper which weighs 12,400 pounds and holds 15,400 pounds of material, the pay load of the same shovel has been increased by 25 percent.[34]

The use of welded steel alloys for the fabrication of dipper handles and booms has further reduced the weight and at the same time has strengthened the construction of the front end of the shovel. Reducing the weight of the loading end of the shovel has not only raised the pay load but has also accelerated the swing-cycle speed. The utilization of antifriction bearings, together with the installation of adequately powered electrical

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[31] For a brief description of the Ward-Leonard control as applied to power shovels see H. R. Johnson, "High Speed Electric Shovels Compact in Design," *Engineering and Mining Journal*, Vol. 138, No. 12 (Dec. 1937), p. 49.

[32] It was found at the Utah Copper mine that, on the whole, the direct-current type of shovel is more economical, efficient, and flexible than the alternating-current type. The a.-c. shovel costs about 18 percent less than the d.-c. machine. The maintenance cost of the a.-c. equipment was approximately 30 percent lower than that of the d.-c., but the power factor of the d.-c. was 54 percent higher and power consumption 15 percent lower than that of the a.-c. shovel. (Goodrich, *op. cit.*, p. 584.) The d.-c. shovel gave a better performance, loading 25 percent more material than the a.-c. shovel with the same power (Soderberg, *op. cit.*, p. 17). Apart from better performance and the economy accruing from lower power consumption and cheaper electric rates that go with a higher load factor, the d.-c. machine is more flexible in its control than the a.-c. shovel. Equipped with the Ward-Leonard control device it is capable, among other things, of limiting the motor torques automatically and also of obtaining high motor torques with a low wattage input to the motor without sacrifice of high light-load motor speed. (R. W. McNeill, "The Electric Power Shovel," *Engineering and Mining Journal*, Vol. 122, No. 20 [Nov. 13, 1926], pp. 764-8.)

[33] See the following articles in *The Mining Congress Journal*, Vol. 24, No. 7 (July 1938): F. C. Ruhloff, "Increased Speed and Flexibility Characterize Improvements in Open Pit Machinery," pp. 14-6 and L. C. Mosley, "Developments in Loading Units at Strip Mines Just as Important as Publicized Advances in Stripping Shovels," pp. 16-7.

[34] Ruhloff, *op. cit.*, p. 14.

equipment, has likewise speeded up the swing cycle and contributed further to the efficiency of the modern electric shovel.

The construction of the other parts of the shovel has also been improved. The dipper teeth have been strengthened and made more durable by the use of forged steel, manganese steel, and molybdenum-chrome steel. The modern shovel frame, too, is fabricated of rolled steel instead of steel castings. Moreover, the wearing parts of the entire shovel are so designed that replacements may be made with a minimum of delay.

As the power, speed, and stability of shovels have been increased, it has become feasible for operators to adopt larger ore-loading machines with bigger dippers and longer booms. In the early years of open-pit mining most of the shovels were equipped with $3\frac{1}{2}$-cubic-yard or smaller dippers. Later the dippers were enlarged to 4 and $4\frac{1}{2}$ cubic yards, and in recent years some mines have acquired 5-cubic-yard shovels. When larger shovels are used, fewer cuts need be made and a larger volume of material may be loaded than when smaller shovels are employed. With longer booms, having two to four times the reach of the smaller shovels, less moving of load track is necessary for the transportation of the ore, a larger volume of material may be shot at one time, and less time is lost in moving the shovel forward. All these improvements have resulted in a material saving in labor and operating costs.

It should be pointed out, however, that there is always a time lag between the development of a new invention or the improvement of an old one and its general adoption. Thus there are only a few power shovels in service at open-pit copper mines that embody all the advanced features of electric shovels. The huge outlays already made for older types of loading equipment naturally limit the rate at which it appears economical to replace such equipment with more modern machines. In many instances operators have deemed it advisable to improve their existing machines by adopting some of the latest features instead of discarding the old for new equipment. With the advent of caterpillar traction, the old steam railroad-type shovels were modified by placing one tractor under each jack-arm and another in the rear. When the electric shovels proved their superiority over the steam machines, steam engines were replaced by electric motors. Similarly, with the development of new and better electric motors, old machinery was dismantled when it required extensive repairs and the latest equipment was installed in its place. The prolonged depression of 1929-33, the recurrence of recessions that followed, and the uncertainty of the demand for copper during the past decade have tended to deter mine owners from making large investments in shovels and other equipment. Eventually, of course, some of the old renovated shovels will become so worn or obsolete that they will have to be replaced by completely new equipment.

In table 1 the power shovels at open-cut mines, approximately as of 1935, are described. Many of these machines were purchased before 1923. From time to time they were remodeled and improved. For example, of the

Table 1.- POWER SHOVELS AT OPEN-CUT COPPER MINES, 1935[a]

Mine	Number	Power	Traction	Dipper capacity (cubic yards)
Utah Copper	23	Electric	Caterpillar	$4\frac{1}{2}$
Copper Flat	1	Electric	Caterpillar	$4\frac{1}{2}$
	3	Electric	Caterpillar	4
Chino	1	Electric	Caterpillar	8
	8	Electric	Caterpillar	4
	4	Steam	Caterpillar	4
	1	Diesel	Caterpillar	$1\frac{1}{4}$
New Cornelia	2	Electric	Caterpillar	$4\frac{1}{2}$
	10	Steam	Caterpillar	4
Sacramento Hill	6	Steam	Railroad	$3\frac{1}{2}$
	1	Steam	Caterpillar	$3\frac{1}{2}$
United Verde (lower pit)	2	Electric	Caterpillar	4
	1	Electric	Caterpillar	3
	4	Electric	Caterpillar	$1\frac{3}{4}$

[a]Compiled from E. D. Gardner, C. H. Johnson, and B. S. Butler, *Copper Mining in North America* (U. S. Dept. Int., Bur. Mines, Bull. No. 405, 1938), p. 128. For the most part, they were for the year 1935. Since 1935 there have been some new acquisitions (not included in the table). In 1937, for example, Utah Copper purchased two new full-revolving electric shovels, the Copper Flat mine one new one, and the New Cornelia mine two new ones and obtained two used ones from the United Verde mine. See *Twenty-Second Annual Report of Kennecott Copper Corporation, 1936*, p. 6; *Twenty-Third Annual Report . . . . 1937*, p. 6; and Phelps Dodge Corporation, *Annual Report: 1938*, p. 6. A. S. Walter, in "Copper in New Mexico," *The Mines Magazine*, Vol. XXVII, No. 11 (Nov. 1937), p. 46, states that the Chino mine also installed a new electric shovel in 1937, but no mention of the purchase is made in the *Twenty-Third Annual Report of Kennecott Copper Corporation, 1937*.

23 electric shovels at the Utah Copper mine 15 were originally powered by steam and placed on railroad tracks but were later converted to electric drive and caterpillar traction.[35] Other mines, like Chino, also modernized their old loading equipment.[36] On the other hand, the Copper Flat mine purchased four modern, full-revolving, electrically powered and caterpillar-mounted shovels between 1931 and 1935 to replace the old-style

[35]Soderberg, *op. cit.*, pp. 14-5.

[36]Thorne, *op. cit.*, p. 22.

steam equipment.[37] Since 1935 some of the other mines have been acquiring new shovels.

The gain in loading efficiency resulting from the introduction of new and better shovels and the constant modification and improvement of existing machines is vividly illustrated by the shovel performances at three of the mines for which figures are available. At the Utah Copper mine the railroad-type steam shovels with 3½-yard dippers loaded, on the average, 2,350 tons of ore per 8-hour shovel-shift in 1923, compared with 5,200 tons loaded, on the average, in 1934 by electric shovels mounted on caterpillar crawlers and using 4½-yard dippers. The new full-revolving shovels equipped with 5-yard dippers which this mine has recently acquired have a maximum capacity of 8,000 tons per 8-hour shift and an average capacity well over 6,000 tons.[38] At the Chino mine the cubic yards of material loaded per 8-hour shovel-shift increased gradually from 825 in 1923, when the old-type railroad steam shovels were in use, to 1,280 in 1931, when the loading machines were modernized.[39] It was found that replacing two steam shovels with two full-revolving electric shovels with caterpillar tractors and 4-yard dippers at the Copper Flat mine in 1931 increased the amount of ore loaded per shovel-shift 10 percent over the preceding year. In 1935 two more full-revolving shovels with caterpillar crawlers were installed, one being equipped with a 4-yard and the other with a 4½-yard dipper. The four electric shovels displaced all nine of the old-style steam machines and they handled as much material in one shift as had the nine steam excavators in two.[40] To some extent this apparent increase in shovel efficiency should be credited to improvements in drilling and blasting and also in transportation.[41] It is obvious that, unless the shovels are continually served with cars and provided with broken material, interruptions in loading operations will result and shovel performance will suffer. In part, this gain in efficiency might also be attributed to changes in mining conditions; at the Chino mine, for instance, a smaller proportion of harder cap rock was removed in the later than in the earlier year.

The effect of the increased efficiency of power shovels on the labor productivity is clearly indicated in figure 15, which shows the relative output of ore per man for one of the mines plotted against the relative

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[37]Gardner, Johnson, and Butler, *op. cit.*, p. 136.

[38]D. D. Moffat, "Mining Utah Copper," *The Mines Magazine*, Vol. XXVII, No. 11 (Nov. 1937), p. 51.

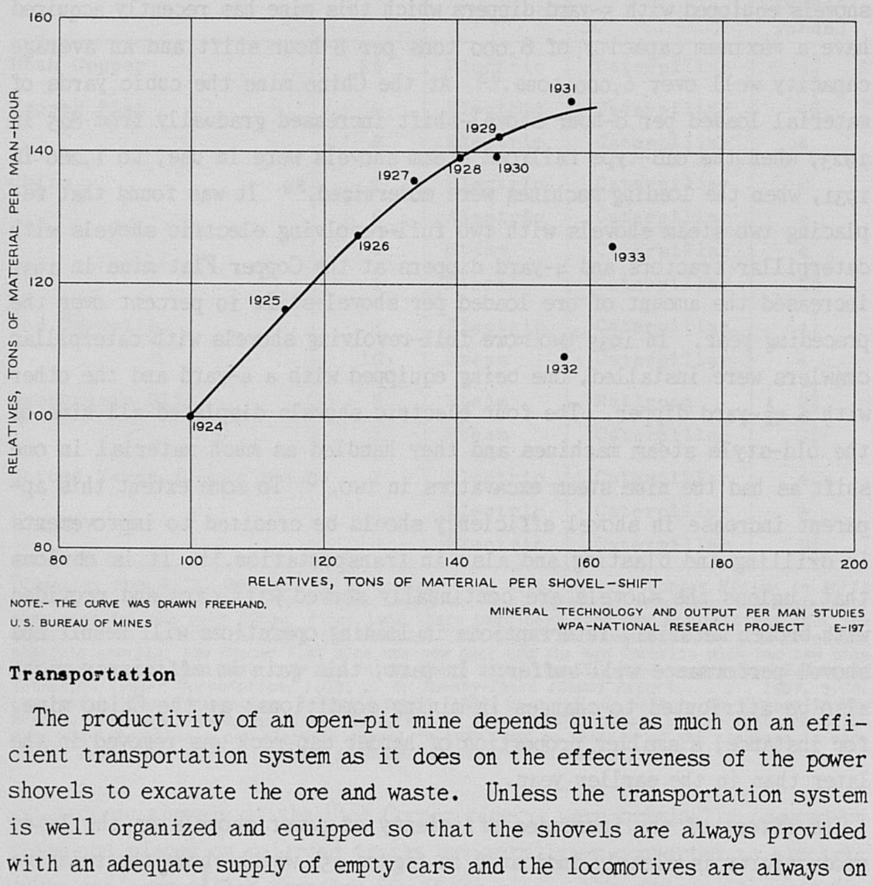
[39]Thorne, *op. cit.*, p. 23.

[40]Gardner, Johnson, and Butler, *op. cit.*, p. 136.

[41]For example, a part of the improvement in shovel performance at Copper Flat in 1931 should be attributed to better transportation, for the track conditions were improved during the year when the new shovels were acquired.

output per 8-hour shovel-shift for the same mine for the period 1924–33.[42] The correlation between the two series (suggested by the freehand curve) is striking throughout the entire period except 1932–33, when the copper-mining industry was severely depressed. In the latter period shovel performance continued to improve while output per man declined materially, mainly because of the stagger system of employment then practiced and the low degree of utilization of mine capacity.

Figure 15.- RELATIVES OF SHOVEL PERFORMANCE AND OUTPUT PER MAN AT ONE OPEN-CUT COPPER MINE, 1924-33
(1924=100)



NOTE.- THE CURVE WAS DRAWN FREEHAND.
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Transportation

The productivity of an open-pit mine depends quite as much on an efficient transportation system as it does on the effectiveness of the power shovels to excavate the ore and waste. Unless the transportation system is well organized and equipped so that the shovels are always provided with an adequate supply of empty cars and the locomotives are always on hand to haul the loaded cars to the mill, costly interruptions of mining operations will ensue. With the adoption of better power shovels having higher speeds and larger capacities, haulage facilities have generally been correspondingly improved.

[42]Complete data on output per shovel-shift for other mines for the same period are not available.

Among the major advances in the field of transportation at open-pit mines are the electrification of mine haulage, the adoption of larger, heavier, and better equipment, and the introduction of new and improved auxiliary devices. These technologic changes have enhanced the efficiency not only of the transportation system but also that of all mining functions.

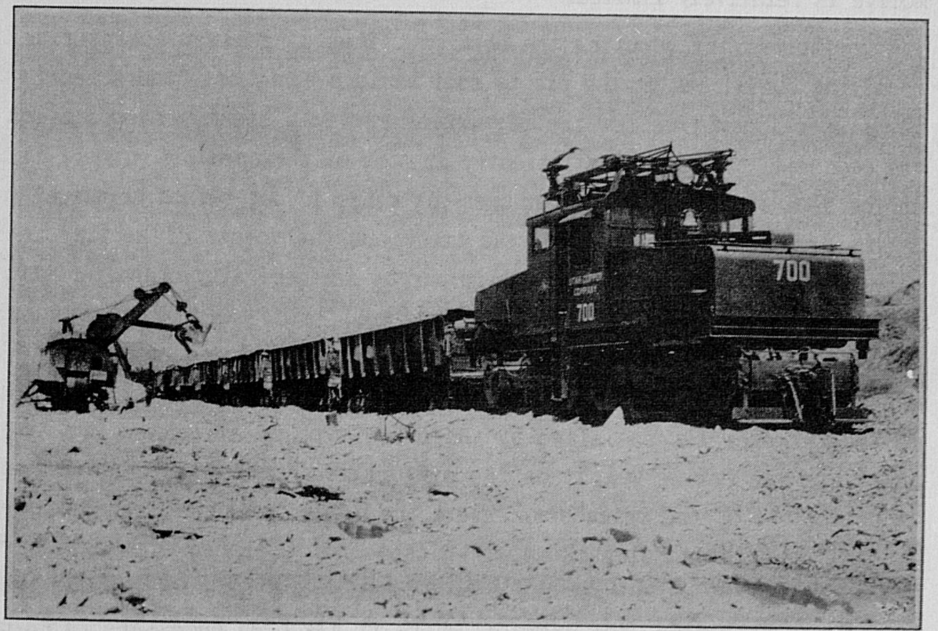

FIGURE 16.- TRANSPORTING WASTE AT AN OPEN-CUT COPPER MINE WITH A COMBINATION TROLLEY STORAGE-BATTERY LOCOMOTIVE

Although the standard trolley electric locomotive is generally the most effective electric haulage unit, the combination trolley storage-battery locomotive is useful where the use of an overhead or side trolley wire is not feasible.

The steam locomotive, despite its relative inefficiency, high cost of operation, and poor adaptability to mining, has still retained its dominant position as a source of motive power at open-cut copper mines, largely because of the high installation cost of alternative haulage systems and of the reluctance on the part of mine operators to scrap their present costly equipment.[43] As far as open-cut copper mining is concerned, the disadvantages of steam locomotives are many. Most of the open-cut porphyry mines are situated in mountainous districts where cheap fuel is not available. It is therefore necessary to transport it to the mine at considerable expense. Scarcity of water and often the poor

[43]The operators of the Nevada Consolidated mine, for example, have been reluctant to make the large outlays required to replace the steam with the more efficient electrical equipment because of the belief that open-cut operations at this mine would be superseded by underground methods before many years (Parsons, *op. cit.*, p. 387).

quality of water for steaming purposes likewise present serious problems in the operation of locomotives. The load factor of the locomotive in the pit is low, for on the average it spends about one-tenth of its time hauling a load. The rest of the time it is spotting cars for the shovel or simply idling and consuming expensive fuel and water. Finally, because of steep grades in the pit, the hauling capacity of the steam locomotive is relatively limited.

Nevertheless, by adopting the best practices of railway engineering, modifying conditions at the pit to suit haulage equipment, and selecting the proper locomotive for the particular service, steam transportation has been made to function efficiently at most of the open-pit mines. The choice of a steam locomotive for open-pit mines is determined principally by the grades and curvature of the track and the volume of hauling required. Since steep grades and sharp curves are generally characteristic of the pits and since the track is seldom well laid, the mine locomotive must have adequate tractive effort and must be articulated enough to negotiate the switchbacks and spirals without derailment. These requirements are generally satisfied by the 0-6-0 type of switch engine, one which has the entire weight on the six driving wheels, with no wheels under the cab or the cylinders. Where the grade or curvature is exceptionally bad the saddleback type, which is not equipped with a tender but which carries water over the driving wheels, may be selected. At the Sacramento Hill mine, for example, where it has been necessary to have curves as abrupt as 40 degrees, saddle-tank engines with four driving wheels instead of six are employed. At the New Cornelia mine, because of its distance from the source of coal, oil-fired locomotives are used instead of the coal-burning type. Because of the severe service, locomotives with surplus power are generally chosen. In the long run an engine with reserve power is more economical than one without any; although its initial cost is higher, its maintenance expenditure is lower and its working life longer.

Conditions at the mines have been constantly modified to permit the most efficient use of existing steam equipment. Wherever possible the steepness of grades is lowered and the abruptness of curves reduced. Delays in spotting trains are avoided by providing passing tracks and spurs on benches as close to the shovels as is practicable. Coaling stations and water tanks are so situated that the locomotives are coaled and watered at the same time.

More important are the many improvements in the steam locomotives themselves. The size of locomotives has been gradually increased. At the

Utah mine, for example, the early steam engines weighed only 50 tons; in 1916, 75-ton types were acquired; and before the complete electrification of the haulage system in about 1930, modern 85-ton locomotives equipped with superheaters were in service.[44] At the Chino mine the largest locomotives weigh 90 tons.[45] The installation of superheaters, which raise the temperature of steam and minimize condensation in the cylinders, has greatly increased the power of the locomotive. A considerable saving in cost has been realized with the substitution of piston valves for the D-valve. Among the improvements which have greatly increased the efficiency of the steam locomotives are the adoption of cab-controlled cylinder cocks which enable the engineer to eliminate the water condensed in the cylinder by remote control; the development of the cab-controlled flange oilers which make it possible for the engineer to reduce the resistance of the curves by oiling the flange of the wheels from the cab while the locomotive is rounding the sharp bends; and the introduction of the power reverse mechanism which permits the engineer to reverse the locomotive quickly and easily, thus facilitating the spotting of cars and the movement of traffic.

The electrification of the haulage system at open-pit copper mines has been slow, partly because of the improvements in steam locomotives which have delayed the necessity for a change in motive power and partly because of the high cost of installing the electrical equipment and the many difficulties inherent in the use of electricity for this purpose. One of the main difficulties lies in the necessity of frequently moving the trolley lines and tracks as the ore is excavated and the face of the bench recedes. Moreover, as blasting at open-pit mines is generally heavy, trolley wires situated near the blasts are likely to be damaged. The ore cars must be moved as soon as possible after each blast, allowing little or no time for repairing electric circuits.

In attempting to avoid some of these difficulties the Utah Copper mine, the only open-pit copper mine in this country to have had its pit haulage system electrified, experimented with a Diesel-electric locomotive and a combination trolley and storage-battery locomotive in 1926-27.[46] The Diesel-electric engine was found to be ill-adapted for open-cut work. With a load factor as low as 10 percent or less and with fuel oil at relatively high prices the Diesel-electric locomotive is at a competitive

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[44] Goodrich, *op. cit.*, p. 569 and Soderberg, *op. cit.*, p. 17.

[45] Thorne, *op. cit.*, p. 24.

[46] *Twenty-Second Annual Report of the Utah Copper Company for the Year Ended December 31, 1926*, p. 9 and *Twenty-Third Annual Report . . . . 1927*, p. 9.

disadvantage with the electric or even steam locomotives at open-cut mines. The conclusion reached was that the simple trolley type was the most effective but that the combination of trolley and storage-battery was suitable for special service and emergencies. In 1928, 11 85-ton electric locomotives were purchased;[47] in 1929, 30 more were acquired.[48] Of the total of 41, 7 were of the combination type. These 41 units replaced a similar number of steam locomotives. Some steam engines were retained for use at the mine on some of the upper levels to handle the short-haul stripping.[49] Gradually, however, the upper levels were also electrified; by 1932 there were about 65 miles of electrified track in and about the pit.[50] No new locomotives were acquired until 1937, when 12 new electric units were purchased.[51]

The electric locomotives at the Utah mine are equipped with a conventional pantograph, two side-arm collectors, a reel containing 2,000 feet of cable, and, in the combination type, a storage battery. The use of spreaders to clear snow from tracks makes it necessary to place all poles on the bank side of the track. On the working benches or levels where the tracks must be shifted from time to time, portable steel towers are utilized to carry the feeder lines to both the shovels and trolley wires. The dump tracks on filled-in areas that are settling continually require that the trolley support be made a part of the track so as to maintain a fixed clearance between the rail and trolley wire.[52]

Electric locomotives were highly developed and efficient when the Utah Copper mine began to adopt them in 1928-29. They were equipped with simple but rugged controls and compact, powerful, and speedy motors having a relatively high horsepower rating per ton of locomotive weight. Nevertheless, there have been some important improvements in their design and construction in recent years. One of these is the application of blowers to cool the motors, an innovation that has materially reduced the frequency of winding the armature. Another is the improvement of the motor bearings of both the plain and antifriction types. The use of dust- and moisture-proof bearings has lowered maintenance expenses. The

[47] *Twenty-Fourth Annual Report of the Utah Copper Company . . . . 1928, p. 9.*

[48] *Twenty-Fifth Annual Report of the Utah Copper Company . . . . 1929, p. 9.*

[49] *Ibid.*

[50] *Twenty-Sixth Annual Report of the Utah Copper Company . . . . 1930*, p. 9; *Twenty-Seventh Annual Report . . . . 1931*, p. 8; and *Twenty-Eighth Annual Report . . . . 1932*, p. 5.

[51] *Twenty-Second Annual Report of Kennecott Copper Corporation, 1936*, p. 6 and *Twenty-Third Annual Report . . . . 1937*, p. 6.

[52] For a more detailed discussion of the electrification of the transportation system at the Utah Copper mine see R. J. Corfield, "Electrification of Utah Copper Mine Haulage System," *Mining and Metallurgy*, Vol. 10, No. 267 (Mar. 1929), pp. 142-4 and Soderberg, *op. cit.*, pp. 17-9.

locomotive, apart from its motors, is also better constructed. The frame, for example, has longer life, for it is made of alloy steel which is welded together instead of cast steel, and the chassis is equipped with better springs.

The substitution of electric for steam locomotives has reduced labor requirements by dispensing with the services of the fireman, watchman, and part of the enginehouse crew and by making it unnecessary to maintain coaling, watering, and other facilities. Further advantages are derived from higher speeds, greater hauling capacity, economy in fuel and power, and safer operating conditions (particularly better visibility) attributable to the absence of smoke. The increase in track capacity resulting from higher speed and greater hauling capacity and the possibility of employing steeper grades also tend to diminish the need for additional track and hence for the labor required for its construction and maintenance. The electrification of the mine haulage system at Utah Copper has made it feasible to increase the number of cars that can be hauled per train from 8 to 12 and the speed by 25 percent; moreover, it has been responsible for some of the improvements in the performance of the electric shovels. The saving, in terms of costs, may be seen from the following tabulation, which presents Utah Copper figures for the average cost of electric haulage for 1930 and 1931 and the average cost of steam haulage for 1927, which was the lowest for any year when steam equipment was used:[53]

Item	Direct-haulage costs	
	Ore (cents per ton)	Waste (cents per cubic yard)
Steam haulage, 1927	4.55	8.25
Electric haulage, 1930-31	2.70	6.07
Saving	1.85	2.18
Percent saving is of steam-haulage cost	40.7	26.4

"Adjusted to equal stripping ratio of waste to ore, the actual saving per ton of ore mined is 2.80 c., which represents a decrease of 34.3 per cent compared with steam haulage."[54]

Another example of the saving in costs resulting from the replacement of steam by electric haulage may be cited. The following tabulation gives the average costs of operating steam locomotives and electric locomotives

[53] A. H. Hubbell, "Economies and Efficiencies in Mine and Plant," *Engineering and Mining Journal*, Vol. 133, No. 1 (Jan. 1932), p. 8.
[54] *Ibid.*

per ton per mile, including rolling stock, at a large, unnamed open-cut mine:[55]

Cost item	Cents per ton per mile			
	Steam locomotives	Electric locomotives	Saving	Percent saving is of steam-locomotive cost
Operating labor	0.201	0.138	0.063	31.3
Repair labor and materials	0.387	0.174	0.213	55.0
Fuel or power	0.579	0.066	0.513	88.6
Total	1.167	0.378	0.789	67.6

It may be observed that by substituting electric for steam locomotives a saving of 68 percent in total operating cost was obtained. The largest saving was in power, amounting to 89 percent of the cost of fuel for the steam locomotives.

Table 2.- LOCOMOTIVES AT OPEN-CUT COPPER MINES[a]

Mine	Number	Kind	Size (tons)
Utah Copper	53	Electric	75
Copper Flat	10	Steam aparejo tank	80
	4	Steam saddleback	80
Chino	11	Steam	90
	8	Steam	85
	4	Steam	$47\frac{1}{2}$
New Cornelia	16	Steam	67
Sacramento Hill	15	Steam saddleback	55
United Verde (lower pit)	3	Steam	85
	2	Steam	55

[a]Compiled from E. D. Gardner, C. H. Johnson, and B. S. Butler, *Copper Mining in North America* (U. S. Dept. Int., Bur. Mines, Bull. No. 405, 1938), pp. 128, 138-40; *Twenty-Second Annual Report of Kennecott Copper Corporation, 1936*; and *Twenty-Third Annual Report . . . . 1937*.

The number, kind, and size of locomotives at the several open-cut copper mines are shown in table 2. Except for 12 electric locomotives acquired by Utah Copper in 1937, these engines were at the mines before the depression which began in 1929. As a result of the many improvements made

[55]Compiled from Jurden, *op. cit.*, p. 148. Although the author does not reveal the source of his data, it seems probable that the figures were for one of the open-cut mines in South America. The figures, originally in terms of cents per long ton per kilometer, have been converted into cents per short ton per mile. The item "Repair labor and materials" includes "labor repairs", "materials", "shop orders", "miscellaneous items", and "renewal repairs."

on them, however, most of them are relatively efficient notwithstanding the fact that they were purchased some years ago.

The substitution of antifriction bearings for plain bearings has had an important effect on the efficiency of haulage at open-pit copper mines.[56] One of the main factors limiting the load a locomotive can haul is the starting friction, which is three times greater for cars with plain bearings than for those with modern bearings. By equipping the cars with antifriction bearings, therefore, a heavier load can be moved by any given locomotive. There is, moreover, a considerable saving in operating costs and maintenance expenses. The consumption of fuel or power and of grease has been reduced. Modern antifriction bearings are so well designed and constructed that they are practically wearproof; they are so carefully housed in oiltight and dustproof enclosures that they need but a minimum amount of lubrication and maintenance. Break-downs due to poor bearings, which have confined many a car in the repair shops, have been greatly reduced.

The ore and waste cars have also been improved and their capacity increased. At the Utah Copper mine, for example, the original ore cars were of 70-ton capacity; they were supplanted by the 80-ton cars in 1922. Since then 90- and 100-ton cars have come into use.[57] The original stripping dump cars were of 6-cubic-yard capacity; in 1910 they were replaced by 12-cubic-yard cars which were in turn superseded by the 30-cubic-yard units in 1916. Recently those of 36-cubic-yard capacity have been adopted as standard equipment.[58] The cars have been better constructed; being heavier and more rugged, they require less repairs. Some of the mines have improved the design of their cars and have thereby increased the efficiency not only of haulage but also of other mining functions. The United Verde mine, for instance, has raised the speed of its stripping operations by the use of a new type of car with drop doors that permit the dumping of larger fragments of rock than the old-type cars with lift doors.[59]

Heavier trainloads require not only more powerful locomotives and larger and sturdier cars but also heavier rails and a better track system. At the Utah Copper mine, for example, the 65-pound rails first used

[56] See the following articles in *The Mining Congress Journal*, Vol. 24, No. 7 (July 1938), pp. 23-5: P. C. Poss, "Standardization and Ease of Assembly Vital in Roller Bearing Design"; R. C. Byler, "Mining with Antifriction Bearings"; and H. G. Dillon, "Speeding the Wheels on Ball Bearings."

[57] In 1937 the Utah Copper mine purchased 240 ore cars of 100-ton capacity. See *Twenty-Second Annual Report of Kennecott Copper Corporation, 1936*, p. 6 and *Twenty-Third Annual Report . . . . 1937*, p. 6.

[58] Goodrich, *op. cit.*, p. 569; Soderberg, *op. cit.*, p. 18; and Gardner, Johnson, and Butler, *op. cit.*, p. 138.

[59] Gardner, Johnson, and Butler, *op. cit.*, p. 140.

on the switchbacks have been replaced by 90-pound rails.[60] Similarly, the weight of the rails at some of the other mines has been increased to 75 and 90 pounds.[61] The lay-out of the track has generally been improved. Care is taken to keep the grade as gentle and the curves as easy as possible, to provide the track with proper ballast, and to maintain the rails in alignment. The use of signals, direct-telephone connections, and semaphores to route the trains has increased the speed of haulage and at the same time reduced the occurrence of accidents.

In open-cut mining the track must move along with the face of the bench as the ore is extracted. The more rapidly the ore is loaded and transported the sooner must the track be moved. With modern loading and haulage equipment operating continuously, a mine must shift its track frequently and must maintain a considerable labor force to perform this function. The adoption of mechanical devices by some of the mines has greatly reduced the labor required for track shifting. At the Utah Copper mine, for instance, the tracks on the benches are moved by nine track-shifting machines, each requiring a crew of only six men.[62] Had it not been for these devices, five to six times as many workers would have been employed to do this labor-consuming task. At some of the other mines bulldozers, or heavy tractors with a road blade, have been utilized to grade the ground and move the tracks, not only on the benches but also at the dumps.[63]

Trucks have been adopted by some mines for transporting ore and waste from the shovels to the ore or waste passes. At the United Verde mine lightweight trucks were first used. They experienced difficulty in overcoming the steep grades maintained in the pit, and they were supplanted by 10-ton-capacity, gravity-side-dump, 6-wheel, 4-wheel-rear-drive trucks that could easily overcome 10-percent grades. In wet weather, transportation was effected by using 10-ton tractor trucks equipped with caterpillar traction on the rear wheels. In 1936 most of this equipment was replaced by 20-ton trucks equipped with hydraulic side-dump body and pneumatic tires.[64] The Clay mine, which has been undergoing development work during the past few years, has been utilizing $22\frac{1}{2}$-cubic-yard trucks to transport the capping.[65]

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[60] Soderberg, *op. cit.*, p. 18. The weight of rails is expressed in pounds per yard.

[61] The New Cornelia mine, for example, improved its track system by increasing the weight of its rails from 75 to 90 pounds in 1938. See Phelps Dodge Corporation, *op. cit.*, p. 6.

[62] Soderberg, *op. cit.*, p. 19.

[63] Bulldozers are used at Copper Flat, United Verde, and New Cornelia mines. See Gardner, Johnson, and Butler, *op. cit.*, pp. 138-40 and Phelps Dodge Corporation, *Annual Report: 1937*, p. 6.

[64] Gardner, Johnson, and Butler, *op. cit.*, p. 140.

[65] Phelps Dodge Corporation, *Annual Report: 1938*, p. 7.

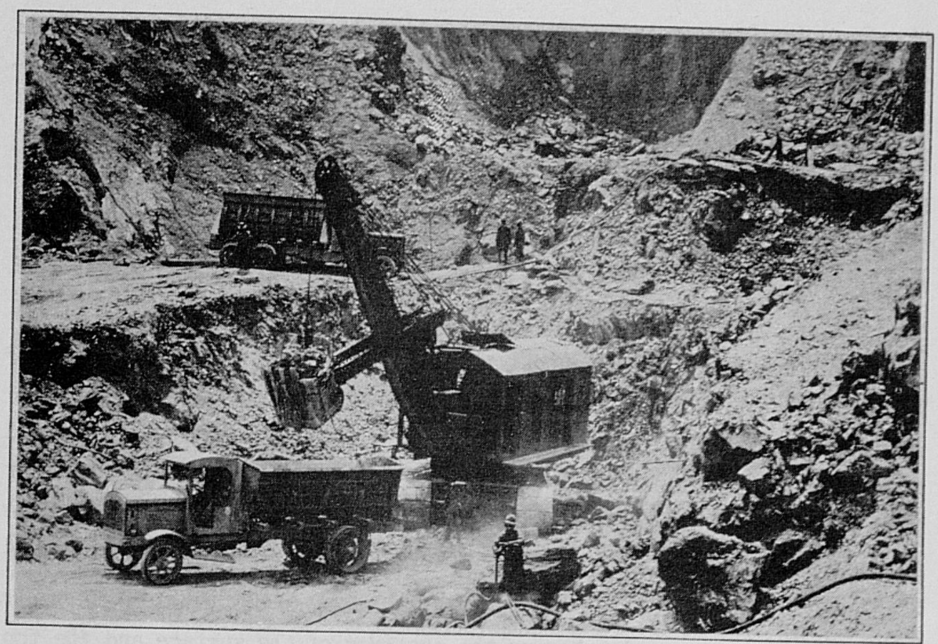

FIGURE 17.— TRUCK HAULAGE IN AN OPEN-CUT COPPER MINE

In surface mining at the United Verde mine specially designed motortrucks transport the material loaded by the electric shovels to transfer raises located at the edge of the pit.

All these advances in the field of transportation have greatly enhanced the effectiveness of power shovels, which are the centers around which all other activities at open-cut mines revolve. They have had a considerable effect on the increase in the output per worker at open-cut mines by enabling the shovels to dig almost continuously during each shift. The lack of statistical data, however, makes it impossible to determine quantitatively what part of the increase in the over-all output per worker may be attributed to these improvements in transportation

PLANNING

Important as are the technologic advances in breaking, loading, and transporting the ore, their effect on the productivity of labor would not have been so great had it not been for the progress in the planning of mining operations. A successful open-cut copper mine depends quite as much on technical proficiency as on the skill of the workers and the efficiency of the mechanical equipment. For this reason every mine maintains a staff of experienced engineers and other experts to plan, coordinate, and supervise operations.

Planning has from the beginning been an important function at open-cut copper mines. Long before any of them began actual operations a considerable amount of prospecting, exploration, and preliminary development had been done to determine the probable shape, size, structure, quality, and boundaries of the ore body. The use of the churn and diamond drills for prospecting greatly simplified the problem of delineating and sampling the porphyry deposits and facilitated the formulation of plans for mining these low-grade ore bodies. Before these drills were introduced development work was usually done by costly underground methods - sinking shafts and driving drifts, crosscuts, raises, and winzes.

Before starting actual mining operations the operators must make decisions about the mining rate, which is closely related to the size of the ore body, the ratio of the output to the ore reserve, the capital outlay for equipment, the lowest grade of ore than can be profitably mined, the amount of stripping that must be done to insure the desired output and grade of ore, the location of approaches to the mine, and other similar broad problems. Having resolved these more general questions of mine planning, they can proceed with the more specific ones.

Accurate maps and sections of the deposit based on drill holes and other information are prepared, showing the contours of the surface and the top and bottom of the ore body, the thickness of the ore and overburden, and an outline of the proposed pit including benches, banks, and slope lines. These are maintained currently as additional information is disclosed by actual mining operations and as the original plans are revised.

The height, width, and slope of the benches must be predetermined. Theoretically, the higher the benches the more economical is the removal of overburden and the more efficient are the loading and haulage operations. The higher the benches the smaller is the horizontal area that must be exposed to allow for the space occupied by the benches, and hence the smaller the amount of stripping required. Both loading and haulage operations are more effective because less time is consumed in moving the shovel ahead and less track work is necessary for each ton of material excavated. It is not always feasible to mine with high benches, however. If the material that is to be excavated is soft and shattered, high benches may be used provided the slope of the bank is not too steep. The rock or ore, which can easily be broken into small pieces, tends to feed down to the shovel. If, however, the material is hard and unbroken by natural fracture, it may not be desirable to work with high benches; in the first place, it is difficult to drill the long holes required and, in the second place, the rock is almost sure to break with a large percentage of oversized fragments that may have to be blockholed or may cause delays in loading and milling operations. The hazard to workmen and equipment is, of course, greater in mining with high than with low benches. Where the material tends to crumble and slough off, dangerous slides may occur on high benches, particularly in seasons of heavy rain and snow. On the other hand, in hard ground the heavy blasting needed to shatter the rock from high benches scatters the broken material over a wide area and may bury shovels, tracks, and other equipment.

A bench should be wide enough to contain the loading track and the shovel course and to permit the bank to be blasted without causing the bench above to cave in or that below to be covered by broken rocks. Within these requirements it is kept as narrow as possible; the narrower it is, the higher is the over-all slope and the smaller is the amount of stripping needed. In soft, fractured ground which may be blasted principally by toe-holes, relatively narrow benches may be worked. However, where it is necessary to resort to heavy blasting because of the tendency of the rock to break coarsely, the width of the bench is increased to prevent damage to the equipment located on the benches above and below.

The height and width of a series of benches form the general slope of the pit.[66] To avoid an excessive amount of stripping, the ideal plan usually calls for a pit slope as steep as practicable at the close of open-cut operations. The maximum over-all slope of the sides of the pit must be determined before mining costs can be estimated, for obviously a slight variation in the angle of this slope may make a great difference in the amount of stripping necessary and hence in mining costs. In general, during the early stages of open-cut operations the bench slopes may be carried at a relatively flat angle, thus giving a slight over-all slope to the pit. At the final stages the width of the benches may be narrowed progressively and the height gradually increased by consolidating relatively low banks into higher ones until it is no longer safe to do so. Berms or narrow benches may be left at intervals to break up steep banks and catch rocks from the upper levels.

One of the objectives of planning drilling and blasting operations is to reduce secondary blasting to a minimum. The presence of a large proportion of boulders in the broken material not only greatly increases the cost of drilling and blasting but also results in costly delays in loading operations. As a general rule, the larger the number of holes that may be shot simultaneously, the better the fragmentation, the smaller the proportion of rock requiring secondary blasting, the less frequent the interruptions at the shovels, and the cheaper the cost of breaking ground.[67]

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66 At the open-cut copper mines the height of the benches ranges from 25 to 80 feet, the width from 50 to 350 feet, and the over-all angle of slope from 26 to 45 degrees. See Gardner, Johnson, and Butler, *op. cit.*, p. 122.

67 Ingham and Barr, *op. cit.*, p. 10.

Another objective is to reduce the total cost of drilling and blasting to a minimum. This is accomplished by spacing the blastholes as far apart as practicable and by using explosives as economically as possible. At each mine there is an economic balance between the spacing of holes and the quantity and quality of powder used. The management must find that happy medium which gives the desired degree of fragmentation at the lowest cost. It decides, largely by empirical methods, whether it would be cheaper to drill more holes or to use more powder; to use air drills to bore small holes rapidly, which must be chambered and which are more difficult to load; or to employ churn drills to bore large holes slowly which need little springing but which can be loaded more safely and quickly.

Transportation lay-out depends principally on the size and shape of the ore body. Where the deposit covers a considerable area and is more or less horizontal and regular in outline, a spiral arrangement may be devised. If the ore body is long and narrow, irregular in shape, or inclined, the switchback is generally adopted. Wherever possible, spirals are employed in preference to switchbacks. In the switchback system the movement of traffic is slower because of the necessity for the train to stop and reverse, whereas in the spiral system traffic may move continuously in one direction. Switchbacks, however, have one important advantage over the spirals: they permit easier curves and more rapid gains in elevation.

Whether spirals or switchbacks are used, the objective is to facilitate the movement of ore and waste from the pit. To avoid delays in spotting trains for loading, convenient passing tracks and spurs are placed as near to the shovels as possible. Double tracks are provided where the traffic on the main lines is heavy. Train movements are controlled by dispatchers and switch tenders stationed at advantageous points in the pit and on the dumps. Coaling and watering stations are installed at some convenient point where locomotives can take on water while they are coaling.

The disposal of the overburden is carefully planned. This waste is disposed of as cheaply as possible and dumped where it will not have to be rehandled. Dumps are therefore as close to the stripping shovels as practicable without occupying any areas that may possibly contain ore. The trackage for waste haulage, like that for ore transportation, is planned with the view of coordinating the haulage system with loading operations. Schedules that route and allocate the waste from the various benches to the various dumps are periodically prepared by the management.

The disposition of ore and waste is closely controlled by the management. A trained sampler is placed at each shovel to obtain a sample from each car of ore as loaded and to send it at definite intervals during each shift to the laboratory where it is immediately analyzed and recorded. The sampler also directs the haulage crew to transport the loaded cars to the mill or waste dump in cases where the shovel is loading material that may be either ore or waste.

Plans for operating an open-pit mine, no matter how carefully conceived, must undergo modification and improvement from time to time as experience indicates the need for changing the original method of procedure. Often only considerable experimentation, largely by trial and error, will permit determination of whether the plan is workable in all its details or whether the objectives for which it was originally designed can be achieved as anticipated. Moreover, as mining conditions and technology are changing constantly, planning must always be in a state of flux. Revisions of earlier plans may become necessary as new conditions present themselves. Inasmuch as mining functions are closely interrelated, a change in the plans in one respect may call for a chain of revisions in the entire original scheme. For example, if, with the beginning of a new level or for some other reason the physical character of the deposit is changed, the plans for the bench work, drilling and blasting operations, and the haulage system may have to be revamped in the light of the new development. If the ground becomes softer, blastholes may be drilled farther apart, less powder may be used, the height of the bench may be increased, and loading and haulage practices may have to be modified. If the rock becomes more difficult to break, other related changes may be necessary. Improvements in equipment may call for some modifications of the earlier plans. For example, with the introduction of the large full-revolving shovels, higher and wider benches may be resorted to where the topography and the nature of the rock permit. Again, the adoption of improved dumping mechanisms in ore and waste cars, which allow bigger fragments to be dumped, and the installation of larger primary crushers at the mills, together with the use of shovels having larger dippers, have made it less necessary to shatter the rock or ore in fragments as small as when older and less efficient equipment was in use.

Managerial changes often represent gradual perfection of details rather than complete revision of the general outlines of the original plans. Each change may not in itself be especially significant, but collectively and cumulated over a period of time they have contributed materially to increasing the output per worker and to counterbalancing the growth in mining difficulties.

INCREASING DIFFICULTIES OF OPEN-CUT MINING

Physical handicaps manifest themselves mainly in an increase in the depth of excavation, a rise in the ratio of overburden to ore, and a decline in the grade of ore. With the exception of two mines - Sacramento Hill, where the porphyry ore body has been exhausted, and United Verde, where open-cut operations probably will be discontinued in the next 3 or 4 years - natural difficulties have not been a highly important problem at open-cut mines.

Although natural handicaps have not been a serious problem in open-cut mining as a whole, this does not mean that mining conditions in every open-cut mine are equally favorable. As a matter of fact, they differ rather widely at each of the six mines. For this reason, physical conditions at each of these mines are briefly reviewed.

Changes in Physical Conditions

Mining conditions at the Utah Copper mine are exceptionally favorable. Excavation is carried on at the side of the hill with an average stripping ratio of about 1 ton of overburden to 1 ton of ore. The rock is relatively soft, breaking by primary blasting into fragments small enough to be loaded by power shovels with a minimum of secondary blasting. On levels above the main haulage level where the assembly yard is situated only the empty cars need be hauled upgrade. On levels below the assembly yard loaded cars must be hauled upgrade. The pit has been increasing in depth, and as operations continue to descend to lower levels, outlet tunnels will have to be driven into the pit to permit loads to be hauled downgrade, as is now the practice in the levels above the assembly yard. With increasing depth, the ratio of overburden to ore probably will rise.

At the New Cornelia mine the ore body is lying near the surface and is relatively flat. The ore, which is hard and difficult to drill, tends to break coarsely. As but very little barren rock covers the ore deposits, the stripping ratio has been extremely small. However, as the extension of the ore body dips under a heavier cover, a larger proportion of stripping will have to be handled, although this increase is expected to be moderate and to have only a minor influence on output per man.

The ore body at the Chino mine lies in a small basin. Here, too, the ore is hard and tough and tends to break in large fragments; it consequently requires twice as much secondary as primary drilling. The stripping ratio is relatively high because the ore alternates with waste, which must be removed as the ore is mined. Part of the ore body at the edge of the pit eventually will be mined by some underground method, which will tend to depress the output per man. As far as open-cut operations are concerned, no great additional physical difficulties are anticipated in the near future, as the present plan is to expand the pit operations in the horizontal rather than the vertical direction.

At the Copper Flat mine in Ely, Nevada, mining conditions are becoming progressively more difficult because of increasing depth of the pit, which necessitates the removal of a greater volume of waste per ton of ore mined and a long, costly haulage of material. Increasing physical difficulties, therefore, tend to lower output per worker from open-cut operations.

At Sacramento Hill mining was carried on at the side of the hill and in the pit at the base of the hill. By 1929 the pit had reached a considerable depth, leaving high, steep banks on two sides. The hazard from slides increased and the haulage of the ore up the spiral tracks from the lower levels became very expensive. Furthermore, the ore body is covered by a capping, averaging 280 feet in thickness, which makes shovel operations rather costly. The stripping ratio was extremely high, reaching an average of 2.75 compared with an average of 1.12 at the Utah Copper mine. For these reasons open-cut mining was discontinued in 1929, and the remaining porphyry ore is being recovered by the glory-hole method.[68]

At the United Verde mine open-cut operations have been conducted at the top of an ore body that is mined principally by the cut-and-fill method. Mining conditions have become progressively more burdensome. With increasing depth and high haulage costs, open-cut mining became uneconomical and was discontinued in 1930. However, power-shovel operations were resumed in 1935 and, according to the amended plans, about 3,000,000 more tons of ore will be mined by the open-cut method.

Changes in the Grade of Ore

As the depth of an open-pit copper mine increases, the grade of ore usually declines. This is explained by the fact that, as a result of secondary enrichment, the upper layers of a porphyry ore body have a larger amount of copper in them than the lower.[69] As mining operations

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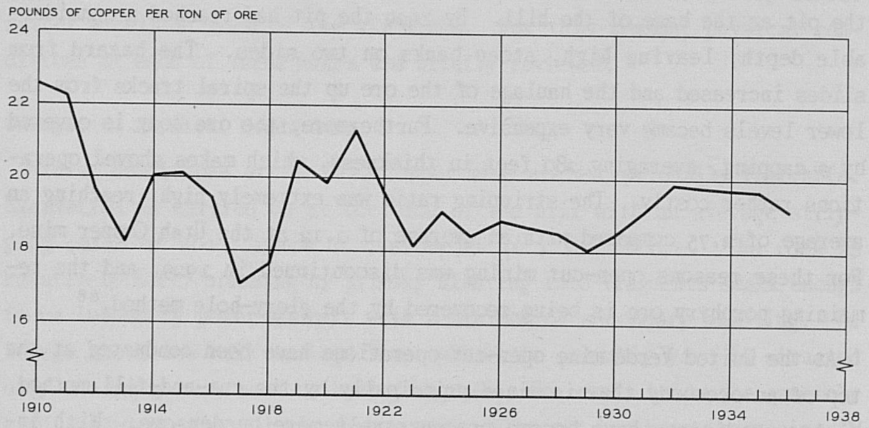
[68] In the glory-hole method the operations are conducted in a pit, the ore being loosened by drilling and blasting and conveyed underground largely by gravity through a funnel-like opening. It is then loaded into cars, transported to the shaft, and hoisted to the surface.

[69] According to geologic theory, when the rocks overlying a copper deposit are eroded, surface waters containing oxidizing agents (CO_2, O, etc.) act on the ore, oxidizing copper minerals which, together with water, form a solution of copper salts. The solution trickles down through a mass of rocks until it comes into contact with some reducing agent like pyrite and chalcopyrite, and is then precipitated in the form of chalcocite or some other sulphide of copper. The chalcocite is redissolved and reprecipitated again and again as the copper migrates persistently downward, forming in the course of geologic ages an enriched zone containing a great concentration of copper. This process is termed "secondary [*Con.*]

proceed downward, the ore usually becomes progressively poorer and may shade finally into barren rock.

One series of data that may reflect changes in the quality of ore mined is the yield of ore by the open-pit mines. The yield of ore, in terms of recoverable copper content per ton of ore excavated at open-pit mines, is illustrated graphically in figure 18. It may be seen that the trend is generally downward. It should be noted, however, that changes in the yield of ore are indicative not only of depletion and growing physical difficulties, but also of other factors.

Figure 18.- YIELD OF ORE AT OPEN-CUT COPPER MINES, 1910-36



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Selective mining, which usually accompanies low copper prices, is one of these. By mining the richer ore and neglecting the poorer grade, the yield will be higher than it otherwise would be; thus the actual changes in the quality of the ore will be obscured to some extent. Improvement in metallurgical technology is another important factor that tends to increase the yield of ore and to conceal the effects of increasing physical difficulties. Before the introduction of flotation and improved ore-dressing equipment about 1912-14, less than 70 percent of the valuable minerals in the ore was recovered even by the most efficient mill. With the advances in milling equipment and techniques, however, recoveries

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69 [*Con.*] enrichment." The concentration of copper is greatest near the permanent water level and diminishes gradually with depth. The presence of oxygen above the water level tends to prevent the precipitation of copper in that zone. The zones just below the water level are enriched to a greater extent than those at a greater depth, as copper is deposted in the upper layers as long as there is an adequate supply of reducing agents. Only with the gradual exhaustion of these in the upper zones does the deposition of copper proceed downward.

have been improved continuously. Today well over 90 percent of the metallic content is recovered.[70]

This gain in metallurgical efficiency, in lowering milling costs, has on the other hand made possible the exploitation of poorer ore. Similarly, the reduction in mining costs resulting from improvements in mining technology has permitted the mining of low-grade material hitherto considered as submarginal. The exploitation of low-grade ore has an effect on the yield of ore opposite to that of increasing efficiency in recovery. Whereas the former diminishes the yield per ton of ore mined, the latter raises it.[71]

In the light of these influences it is not surprising that the rate of change in the yield should vary from year to year. To arrive at an average rate of decline it is advisable to compare the yield during two comparable periods spaced far enough apart to reflect the trend. The comparability of two such periods may be determined by the extent to which the levels of copper prices and production are similar. In the 5-year period 1910-14, during which the fluctuations in the price of copper were relatively minor,[72] the average yield was 20.2 pounds per ton of ore. The other 5-year period chosen for comparison is that of 1923-27, during which the average high was 18.6 pounds per ton of ore. It should be noted that this latter period did not include the peak year of prosperity, that the level of copper production was closely similar to that of 1910-14,[73] and that the fluctuations in copper prices were negligible.[74] The decline between the two periods was 7.9 percent.

The yield of metal in the years during which production experienced a sharp decline was somewhat higher than that in the immediately preceding years. The years 1919 and 1921 bring out this point rather clearly (see figure 18). During the depression years 1931-34 the yield of metal was again consistently higher than that in any of the eight preceding years. This seems to indicate that selective mining is resorted to by the open-cut operators in periods of low production.[75] This conclusion

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[70] See ch. V.

[71] A more detailed discussion of the effect of the exploitation of low-grade porphyry ore and the improvements in metallurgical technology on the yield of ore may be found in A. V. Corry and O. E. Kiessling, *Mineral Technology and Output per Man Studies: Grade of Ore* (WPA National Research Project in cooperation with U. S. Department of the Interior, Bureau of Mines, Report No. E-6, Aug. 1938), pp. 46-56.

[72] The average price of copper for the period 1910-14 was 14.1 cents per pound with a maximum deviation of 2.4 cents from the average.

[73] See fig. 18.

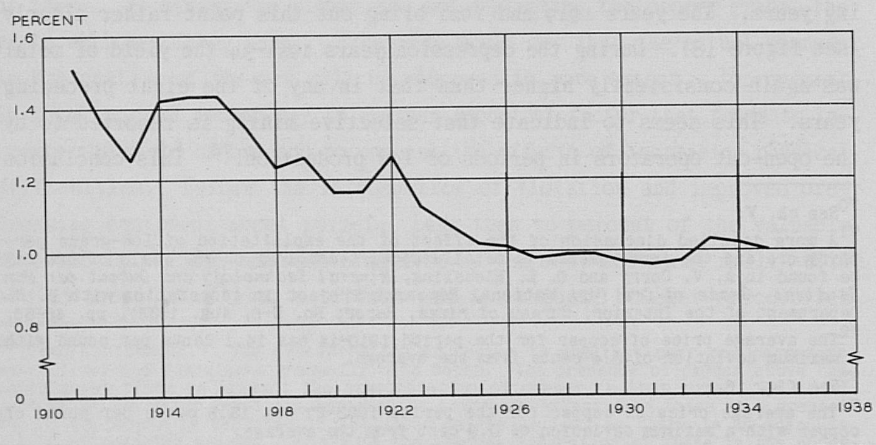
[74] The average price of copper for the period 1923-27 was 13.8 cents per pound of copper with a maximum deviation of 0.9 cent from the average.

[75] Except in 1933, when the New Cornelia mine was idle, the figures on yield during the depression years 1931-34 include the operation of all four mines. Also, no new surface operations were undertaken in this period.

is interesting in view of the widespread notion that selective mining is not practiced by the operators of open-cut porphyry mines.

Gradual depletion of richer ore would be better reflected in a statistical series registering the tenor or grade of ore than in one measuring the yield of ore. Although both the tenor of ore and the yield of ore indicate the effect of selective mining and the extraction of progressively lower grades of ore, the former is free from one of the main disturbing influences - the improvement in the efficiency of recovery. Unfortunately figures for the grade of ore are not available for all open-pit mines. However, the series available for one of the mines, Utah Copper, perhaps typifies the trend of the tenor of ore of the other open-cut mines. The annual average copper content of the ore that was concentrated at the mills of Utah Copper mine for the period 1911-35 is plotted in figure 19. It will be observed that the trend was declining. In 1911 the copper content of the ore concentrated was 1.51 percent, while in 1931 it was only 0.96 percent. The departure from this downward tendency during the period 1932-35 may be attributed mainly to selective mining. In part, at least, this declining trend is to be ascribed to the depletion of higher grades or ore and the growth in mining difficulties with the passage of time. In part, of course, it finds explanation in the fact that improvements in mining and metallurgical techniques and the resultant reduction in mining and milling costs permitted the treatment of material which had been regarded as waste but which in later years was classified as ore.

Figure 19.- COPPER CONTENT OF ORE CONCENTRATED AT MILLS OF UTAH COPPER MINE, 1911-35



U. S. BUREAU OF MINES
BASED ON TABLE A-15

MINERAL TECHNOLOGY AND OUTPUT PER MAN STUDIES
WPA-NATIONAL RESEARCH PROJECT E-199

It may be concluded that, in general, mining difficulties have not been a crucial problem at open-cut mines. To be sure, the pits have become deeper, the ratio of overburden to ore has become higher, and the richer ores have been gradually exhausted. For the most part, however, this increase in natural handicaps has been counterbalanced by improvements in mining and metallurgical technology and practices which in turn have permitted the mining and treatment of lower grade ores and have resulted in an increase in the amount of available reserves.

CHAPTER IV

UNDERGROUND MINES[1]

Despite the rapid growth of open-cut mining and the decline in underground mining, underground mines still produce almost 60 percent of the copper output and provide most of the employment in the industry. In the period 1917-22 they employed an average of 93 percent of the total number of men employed at copper mines. With the growth of open-cut mining the proportion was somewhat reduced; during the period 1923-31 it was 88 percent and in 1932-36 it was 83 percent.

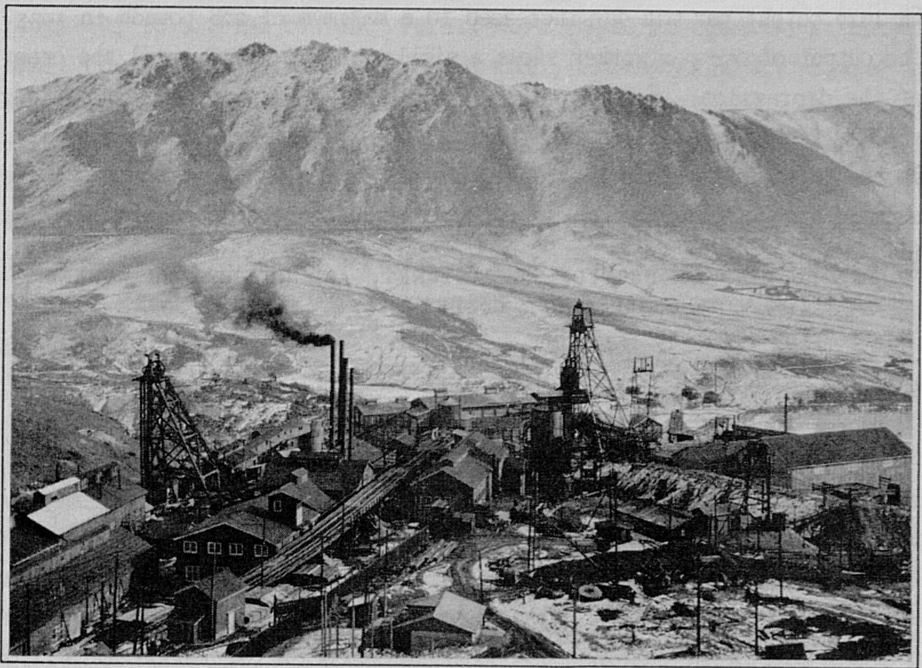
PRODUCTION, EMPLOYMENT, AND OUTPUT PER MAN

Production from underground mines fluctuates widely. The small and high-cost producers, who comprise a considerable portion of the total, are affected by fluctuations in business conditions to a far greater extent than the large open-cut mines. When prices fall a large number of these mines close down completely. The number of active mines during the 1932-34 period, for instance, when the price of copper fluctuated between 6 and 8 cents per pound, was about 50 percent lower than in the years prior to the depression when the price stood at about 14 cents or over. The peak output from underground mines was attained in 1916, when it exceeded 1,600,000,000 pounds, or roughly 83 percent of the total production (figure 21). Owing to the rapid growth of open-cut operations, the proportion from underground mines was steadily declining. In 1929 underground mines produced about 1,280,000,000 pounds of copper, or only about 65 percent of the total, even though the total output was practically the same as in 1916. In 1936 the share from underground mines reached a minimum, representing slightly over 56 percent of the total.

Although underground mines still account for most of the jobs in the industry, the number of men employed during 1917-36, the period covered by most of the accompanying statistics, showed a declining trend. From a peak of 57,600 men in 1916, or 94 percent of all workers employed in that year, employment in underground mines declined to 16,900 in 1921. It rose gradually to a high of 32,000 in 1929 but thereafter decreased rapidly to a trough in 1933 with only 5,700 men, or 82 percent of the total number employed in the industry.

The output of copper per worker in the underground mines was rising throughout the entire period covered by the study. From about 101 pounds

[1] By J. C. Burritt, Y. S. Leong, Emil Erdreich, C. E. Nighman, and George C. Heikes.



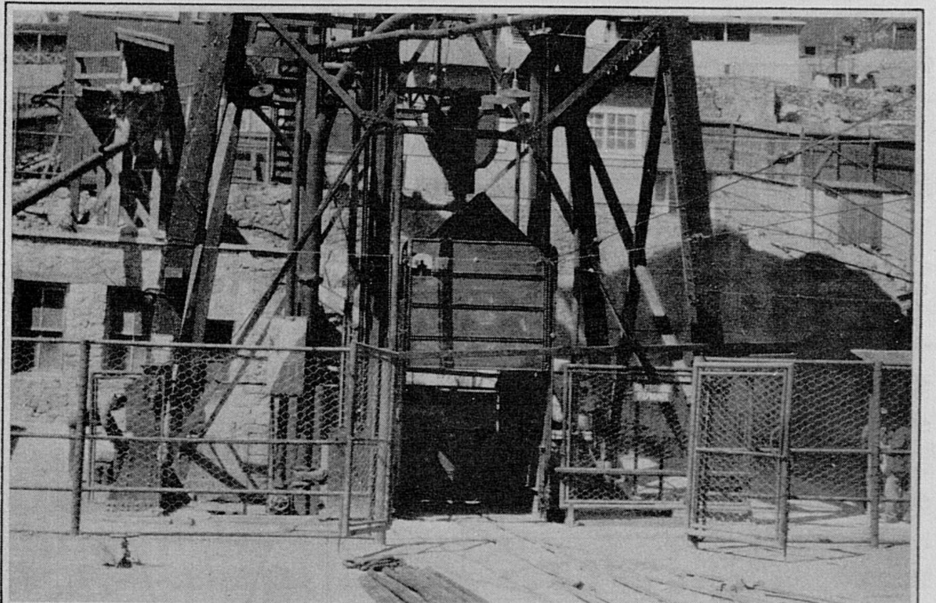


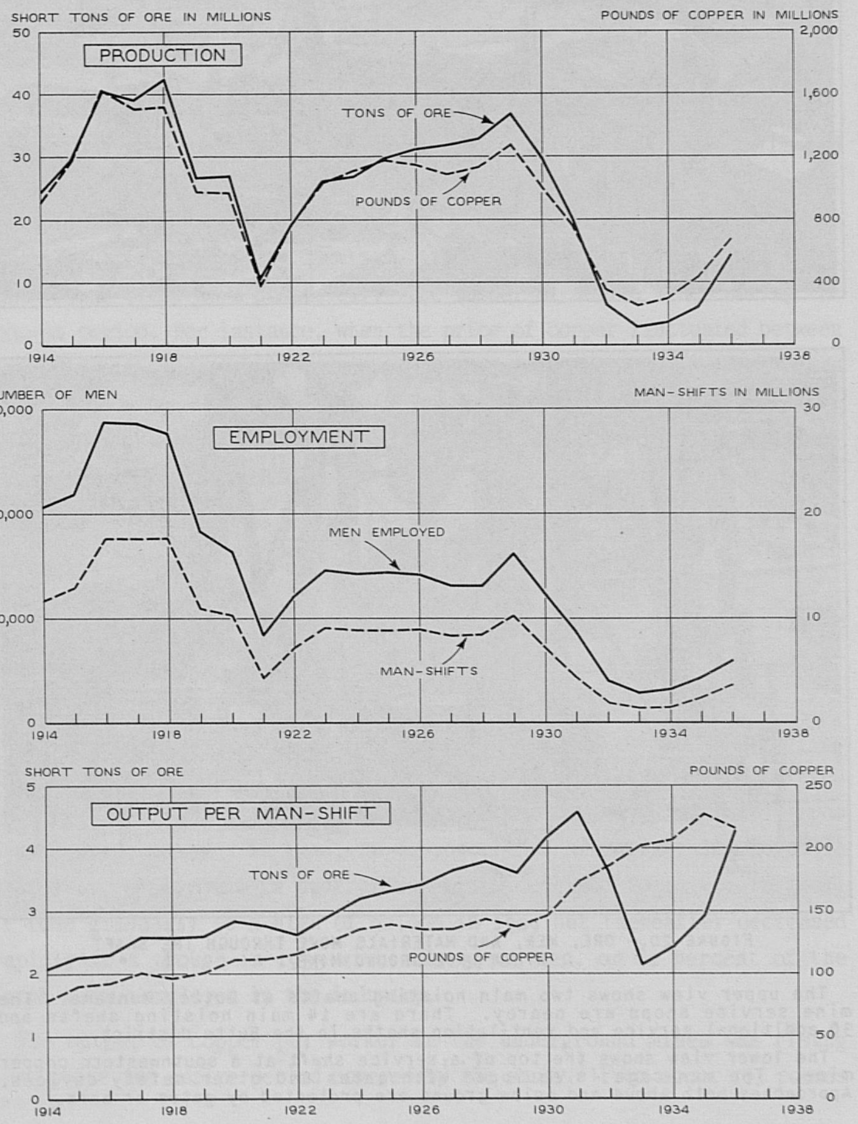
FIGURE 20.- ORE, MEN, AND MATERIALS MOVE THROUGH THE SHAFT AT AN UNDERGROUND MINE

The upper view shows two main hoisting shafts at Butte, Montana. The mine service shops are nearby. There are 14 main hoisting shafts and 30 additional service and ventilation shafts in the Butte district.

The lower view shows the top of a service shaft at a southwestern copper mine. The man-cage is equipped with gates and other safety devices. Approaches both above and below ground are protected by gates or bars.

in 1917 output per man-day increased to a maximum of 228 pounds in 1935. The output of ore per worker shows a similar rising trend until the onset of the depression. From about 2.6 tons per day in 1917 the output per man was increasing rather steadily up to 1931, reaching 4.6 tons per day. In the following 2 years it dropped sharply to slightly over 2 tons, with a rapid recovery in 1935-36. The decrease during the depression was due

Figure 21.- PRODUCTION, EMPLOYMENT, AND OUTPUT PER WORKER AT UNDERGROUND COPPER MINES, 1914-36



U. S. BUREAU OF MINES
BASED ON TABLES A-10, A-7

MINERAL TECHNOLOGY AND OUTPUT PER MAN STUDIES
WPA-NATIONAL RESEARCH PROJECT E-200

primarily to several causes. First, when the mines operated at only a small fraction of their productive capacity, the number of men engaged in maintenance was reduced but slightly, with the result that this relatively large overhead labor had a depressing effect on the output per worker during the periods of low production. Secondly, selective mining which was practiced intensively during the depression also tended to curtail the output of ore per worker, for it required a considerable amount of time to sort the broken ore underground by hand. Moreover, as the amount of broken material rejected and left in the mines was relatively larger than when selective mining was not done or done less intensively, the output of ore per worker was necessarily smaller than it otherwise would have been. Finally, some of the block-caving mines that had a high ore output per man reduced their production more than proportionally, thus causing a sharp decline in the over-all trend. As copper output per man at these mines was only moderately higher than that at the rest of the mines, this decrease in production had virtually no effect on trend of output of copper per worker. The divergence of the two trends shown in the bottom section of figure 21 is explained by these factors, particularly by the high degree of selective mining practiced during the depression. The high metal content in the ore mined selectively helped to maintain the output of metal per man at a high level even though selective mining required a greater amount of labor per ton of ore.

Among many factors that explain deviations of the output per man from the general trend, the most important are the volume of waste handled underground, the entrance and exit of marginal producers during periods of sharp price changes, development work, and maintenance and other overhead labor. The available data for volume of waste handled are fragmentary and do not permit an accurate analysis of the effect of this factor on output per man. It is obvious, of course, that a reduction in the amount handled increases output of ore per man and that an increase tends to have the opposite effect. Changes in the other factors mentioned are associated with fluctuations in the percent of mine capacity utilized and in general business conditions. Although the precise influence of each of these factors cannot be isolated, the over-all effect has been to raise the output of copper per worker during the depression years.[2]

The curves for production, employment, and output per man presented in figure 21 represent aggregate figures for underground mines employing a variety of mining methods. A better understanding of the effect of the various factors affecting these trends may be had by presenting separate

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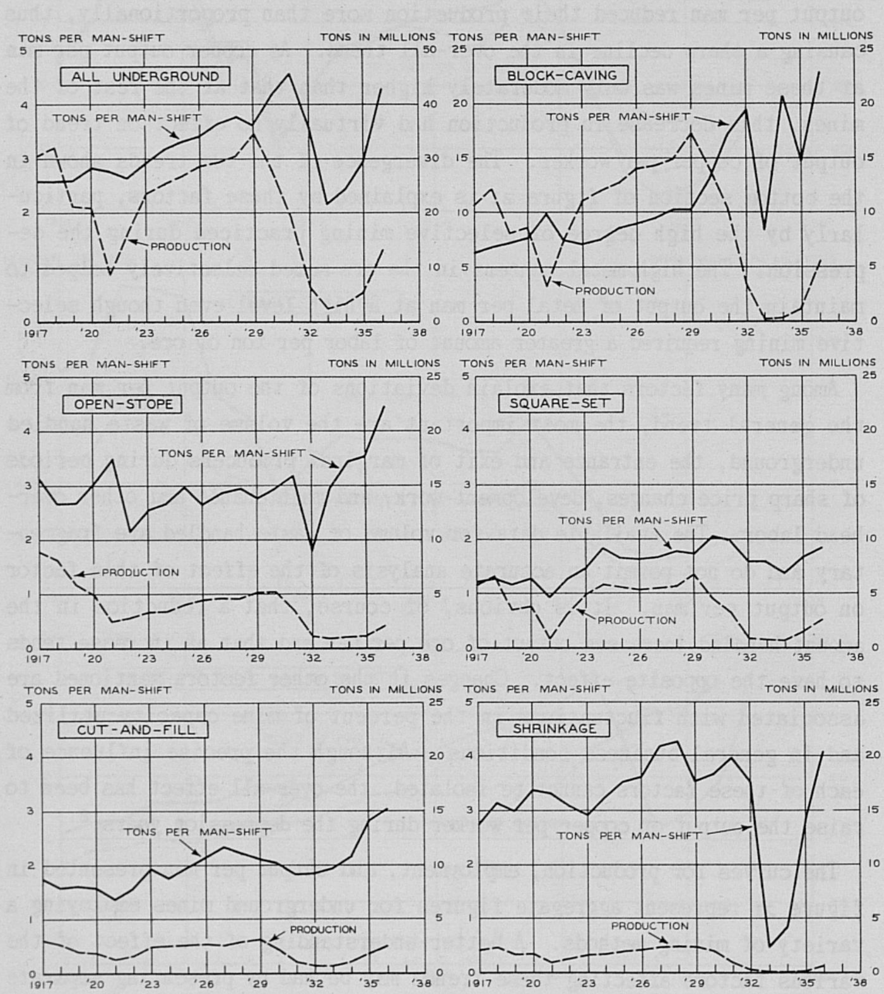
[2]A more detailed analysis of the effects of these factors is presented in chapter III.

series for individual groups of mines classified by the mining method employed. With few exceptions, however, nearly all underground mines used more than one mining method, either simultaneously or during the period covered, and some of them employed as many as three or four. Any grouping of mines on the basis of mining methods must therefore be more or less arbitrary. In the classification adopted for this analysis, the mines are segregated by the predominant methods employed during the period 1917-36. The period 1917-36 is selected for analysis because beginning in 1917 fairly reliable and detailed statistics of production and

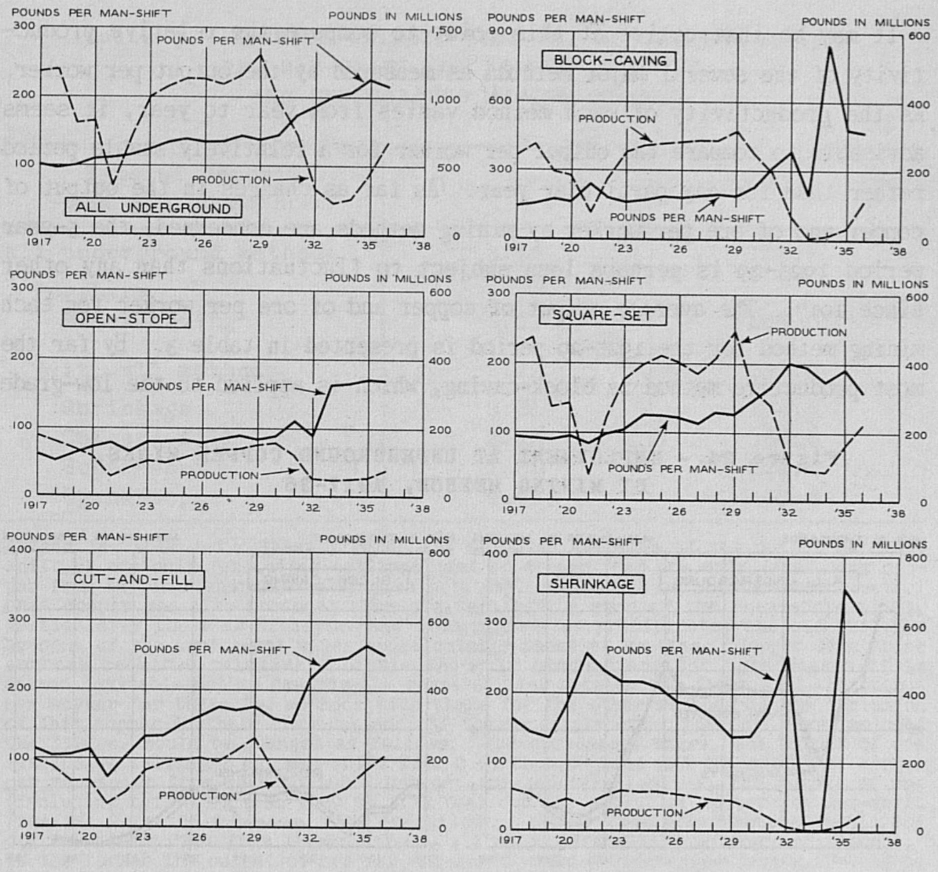
Figure 22.- PRODUCTION OF ORE AND OUTPUT PER WORKER AT UNDERGROUND COPPER MINES, BY MINING METHOD, 1917-36



U. S. BUREAU OF MINES
BASED ON TABLE A-7

MINERAL TECHNOLOGY AND OUTPUT PER MAN STUDIES
WPA-NATIONAL RESEARCH PROJECT E-201

Figure 23.- PRODUCTION OF COPPER AND OUTPUT PER WORKER AT UNDERGROUND COPPER MINES, BY MINING METHOD, 1917-36



U. S. BUREAU OF MINES
BASED ON TABLE A-7

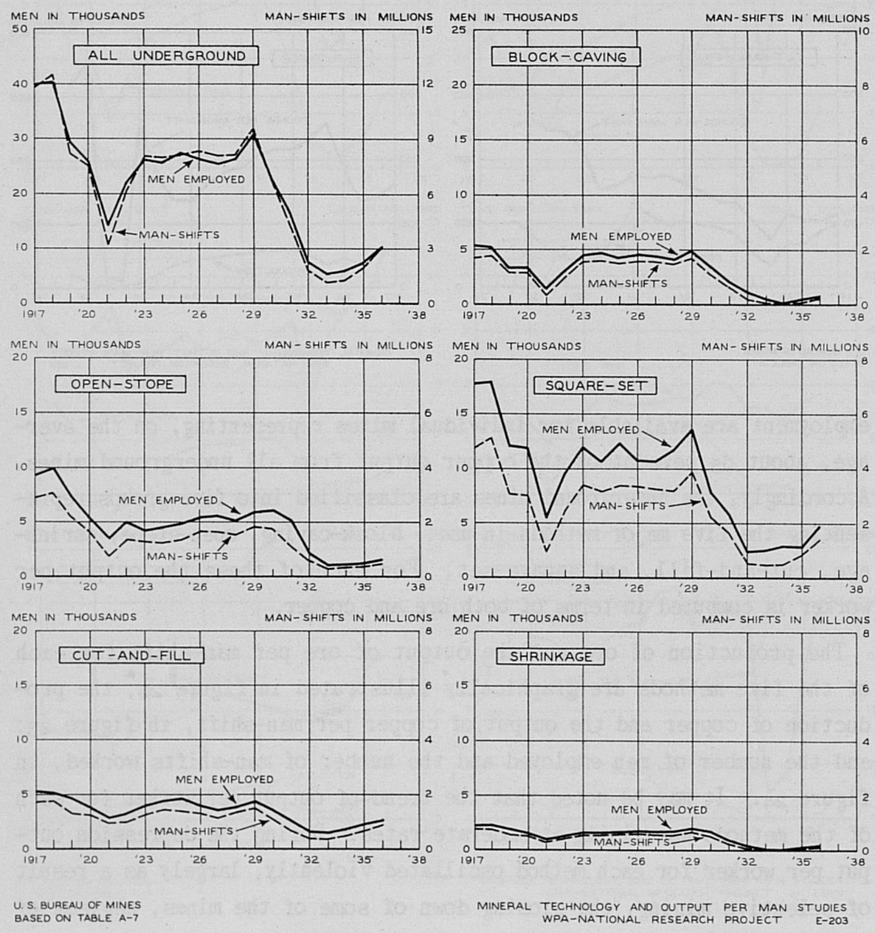
MINERAL TECHNOLOGY AND OUTPUT PER MAN STUDIES
WPA-NATIONAL RESEARCH PROJECT E-202

employment are available for individual mines representing, on the average, about 95 percent of the copper output from all underground mines. Accordingly, the underground mines are classified into five groups representing the five major methods in use: block-caving, open-stope, shrinkage, cut-and-fill, and square-set. For each of these the output per worker is computed in terms of both ore and copper.

The production of ore and the output of ore per man-shift for each of the five methods are graphically illustrated in figure 22; the production of copper and the output of copper per man-shift, in figure 23; and the number of men employed and the number of man-shifts worked, in figure 24. It may be noted that the trend of output per worker for each of the methods was rising at moderate rates. During the depression output per worker for each method oscillated violently, largely as a result of selective mining, the closing down of some of the mines, neglect of development work, and restricted maintenance and repairs. Even during the period 1917-29 the output per man for each of the methods fluctuated to some extent.

It may be instructive at this point to compare the relative productivity of the several major methods as measured by the output per worker. As the productivity of each method varies from year to year, it seems advisable to compare the output per worker for a relatively stable period rather than for any particular year. As far as changes in the output of copper and of ore per worker by mining methods are concerned, the 5-year period 1925-29 is perhaps less subject to fluctuations than any other since 1917. The average output of copper and of ore per worker for each mining method for the 1925-29 period is presented in table 3. By far the most productive method is block-caving, which is applied to the low-grade

Figure 24.- EMPLOYMENT AT UNDERGROUND COPPER MINES, BY MINING METHOD, 1917-36



disseminated deposits.[3] Among the methods utilized to exploit vein deposits, the most productive is shrinkage, which is followed by cut-and-fill, square-set, and open-stope.[4]

Table 3.- AVERAGE OUTPUT, 1925-29, OF ORE AND COPPER PER MAN-SHIFT, BY TYPE OF DEPOSIT AND UNDERGROUND MINING METHOD[a]

Type of deposit and mining method	Tons of ore	Pounds of copper
All underground methods	3.56	138.2
Porphyry		
Block-caving	9.51	188.5
Vein, all methods	2.23	127.0
Shrinkage	3.81	174.5
Cut-and-fill	2.15	166.6
Square-set	1.69	124.2
Open-stope	2.85	83.8

[a]Based on table A-7. These figures are weighted averages of the output per man-shift of ore only (excluding tailings) and of copper from ore only (excluding copper from tailings and mine-water precipitates). It should be pointed out, however, that copper was also produced from old tailings by some of the open-stope mines, particularly those at Michigan, and from mine-water precipitates and old tailings by some of the square-set mines, particularly those at Montana. Copper from these sources required relatively little labor to produce and for this reason it is deemed advisable not to destroy the comparability between the figures of the output per worker for these two methods with those for the other methods by the inclusion of this copper in their computation. If the materials other than ore were included the figures would be changed as follows: For open-stope mines, the output of ore (including tailings) per man-shift from 2.85 to 4.53 tons and the output of copper per man-shift from 83.8 to 102.5 pounds; for square-set mines, the output of ore (including tailings) from 1.69 to 1.72 tons and the output of copper per man-shift from 124.2 to 126.2 pounds. The inclusion of copper from other sources would, of course, affect two other items: "Vein, all methods" and "All underground methods." In the former the output of ore per man-shift would be increased from 2.23 tons to 2.64 tons and the output of copper per man-shift from 127.0 pounds to 132.4; in the latter, the output of ore per man-shift would be increased from 3.56 to 3.89 tons and the output of copper per man-shift from 138.2 to 142.6 pounds.

It should also be noted that byproduct metals in the form of lead, zinc, gold, and silver were also produced by copper mines employing the various methods. The effect of these byproducts on labor productivity may be observed from the following tabulation which presents for comparison the average output per man-shift, 1925-29, of copper and of copper plus the copper equivalent of the four accessory metals:

Type of deposit and mining method	Pounds of copper	Pounds of copper plus copper equivalent
All underground methods	142.6	154.0
Porphyry		
Block-caving	188.5	190.8
Vein, all methods	132.4	145.7
Shrinkage	174.5	187.1
Cut-and-fill	166.6	191.5
Square-set	126.2	141.4
Open-stope	102.5	103.1

As accessory metals were also derived from tailings and mine-water precipitates, the copper from these sources is included in the above tabulation. It will be seen that these byproducts had a substantial effect on the output per worker of three of the methods - cut-and-fill, square-set, and shrinkage.

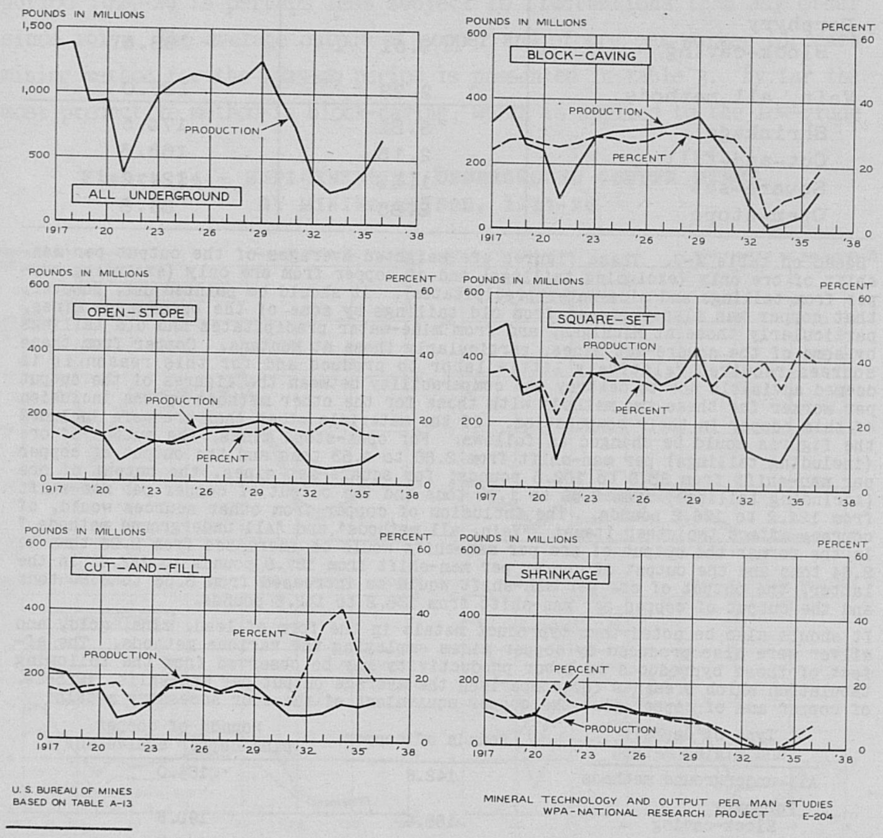
[3]The open-cut method, which is also applied to porphyry deposits, is even more productive: for the period 1925-29 the average output per worker per 8-hour shift is 16.9 tons of ore, 318.7 pounds of copper, and 338.7 pounds of copper plus the copper equivalent of the accessory metals (see table A-7 for detailed figures).

Footnote 4 appears on the following page.

RELATIVE IMPORTANCE OF MINING METHODS

Only part of the increase in the over-all output per worker for underground mines (as shown in the top left section of figures 22 and 23) is ascribable to the rise in labor productivity of the several methods. The other part of the gain in over-all output per worker is accounted for by the growing proportion of the production coming from mines employing the more productive methods.[5] This is revealed in figure 25, which shows

Figure 25.- DISTRIBUTION OF COPPER PRODUCED AT UNDERGROUND COPPER MINES, BY MINING METHOD, 1917-36



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[4]It is interesting to note that although the output of ore per worker for the open-stope method was higher than that for the cut-and-fill or for the square-set method, the output of copper per worker was the lowest. This is largely explained by the low yield of ore from mines that employed the open-stope method. Thus, in the period 1925-29 the average yield from the mines using the open-stope method was 29 pounds to the ton compared with 73 pounds for mines employing the square-set method.

[5]It should not be interpreted that because one method is more productive (that is, has a larger output per worker) than another it is more efficient. As a matter of fact, a method that is less productive than another may be highly efficient if it is properly applied. As will be shown later, the choice of a method is governed largely by the structure of the deposit and the nature of the ore. The open-stope method, for example, which is the least productive, may be the most efficient if it is applied to the appropriate type of ore body, the mining of which by some other method would have been more costly or could not have been done at all.

graphically the underground production of copper and the percentage contributed by each method for the period 1917-36. It will be observed that block-caving, the most productive of the underground methods, gained in importance before the depression; the square-set, cut-and-fill, and open-stope methods were barely able to hold their own and the shrinkage method declined rapidly in relative importance after 1921.[6] The chart does not cover the period in which the most marked shifts in relative importance occurred. Had data been available before 1917, they would have disclosed more striking changes than those shown here.[7] During the depression period production by all the methods declined, but not evenly.

Figure 26 illustrates the percentage distribution of copper production by surface as well as underground methods. The top section indicates that before 1929 the output from porphyry ores, which were exploited by the more productive open-cut and block-caving methods, was increasing in relative importance at the expense of the output from vein ores, which were mined by the less productive shrinkage, cut-and-fill, square-set, and open-stope methods and their variations. The middle section tells much the same story as figure 25, namely, that block-caving was increasing its relative share and that shrinkage was losing ground. The relative importance of the several methods employed in mining vein deposits is shown in the lower section, which discloses that except for the decline in the share of the output by the shrinkage method the changes in the relative positions of the other major methods used in vein mines before the depression were not particularly significant.

If reference is made to the top right section of figure 25 and the middle section of figure 26, it will be seen that the block-caving mines, which are situated principally in Arizona and Nevada, increased their share of the output of copper from 24 percent of the total production from all underground mines in 1917 to about 29 percent in 1930. Beginning in 1932, when the demand for copper dwindled to a low level, mining companies found it necessary to close some of their mines. It was more economical to cease operating their block-caving mines and to concentrate

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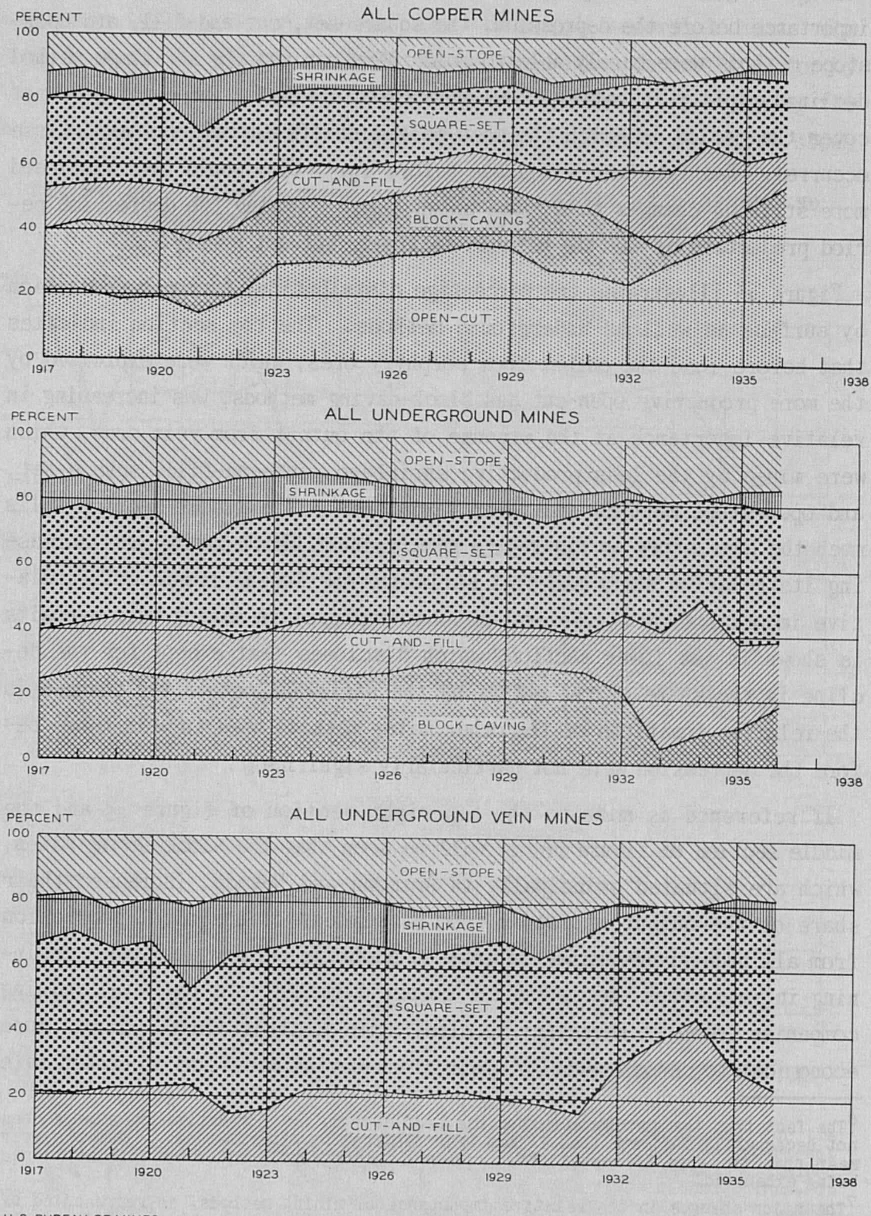
[6]The fact that the shrinkage method has been decreasing in relative importance does not necessarily indicate that it has been replaced by other methods. It may simply mean that the type of deposit that can be advantageously mined by this method is being exhausted.

[7]The major changes in the relative importance of mining methods, as represented by the relative volume of production, occurred at the end of the first decade of the present century, when the open-cut and block-caving methods came into general use in the exploitation of porphyry ores. During the next 10 years the production of copper by these methods increased steadily, reaching about 40 percent of the total output at the beginning of the World War. These changes are indicated in chapter II where, in the absence of actual production statistics by mining methods prior to 1917, the analysis was based largely on the estimated output of copper produced by mines employing the more important methods.

Figure 26.- PERCENTAGE DISTRIBUTION OF COPPER PRODUCTION, BY MINING METHOD, 1917-36



U. S. BUREAU OF MINES
BASED ON TABLE A-13

MINERAL TECHNOLOGY AND OUTPUT PER MAN STUDIES
WPA-NATIONAL RESEARCH PROJECT E-205

their operations at other mines under their control.[8] Consequently the relatively small volume of production during the period since 1931 does not indicate waning importance of this method.

The square-set mines are situated mainly in Montana and Arizona. They retained their relative position practically throughout the period covered. However, increasing depth, which has been a handicap at most of these mines, will in the near future result in a decreasing share of production by mines utilizing this method.

Open-stope mines are situated largely in Michigan and Tennessee. Before the depression the proportion of the output from open-stope mines was practically unchanged. In the depression period it increased to some extent largely because of a greater decline in the output by some of the other methods. In connection with this group of mines, two points should be mentioned. First, in Tennessee copper is only a byproduct, for the sulphide ores are mined principally for the manufacture of sulphuric acid. Consequently fluctuations in the copper output of these mines were not attributable to the same factors that affected the copper production of other mines. Second, the mines in Michigan could maintain their relative position in the industry largely because of the availability of the large volume of tailings[9] accumulated in the past from exploitation of the richer and more available ores of these mines. The cost of producing copper from this source was only a fraction of that from the underground mines where physical handicaps in the forms of increasing depth and declining quality of ore were consistently accumulating. The amount of labor required to produce copper from tailings, too, was relatively small. The significance of this source of copper is indicated in figure 27, which shows the production of ore and copper and the output of ore and copper, with and without tailings, per worker for Michigan. Without tailings the level of production and the output per worker both in terms of ore and of copper are appreciably lower. Since this source of copper is likely to be exhausted in the next 10 years or so, and since the cost of producing copper from underground mines is rising with increasing physical difficulties, the relative volume of output from open-stope mines will probably decline in the not-distant future.

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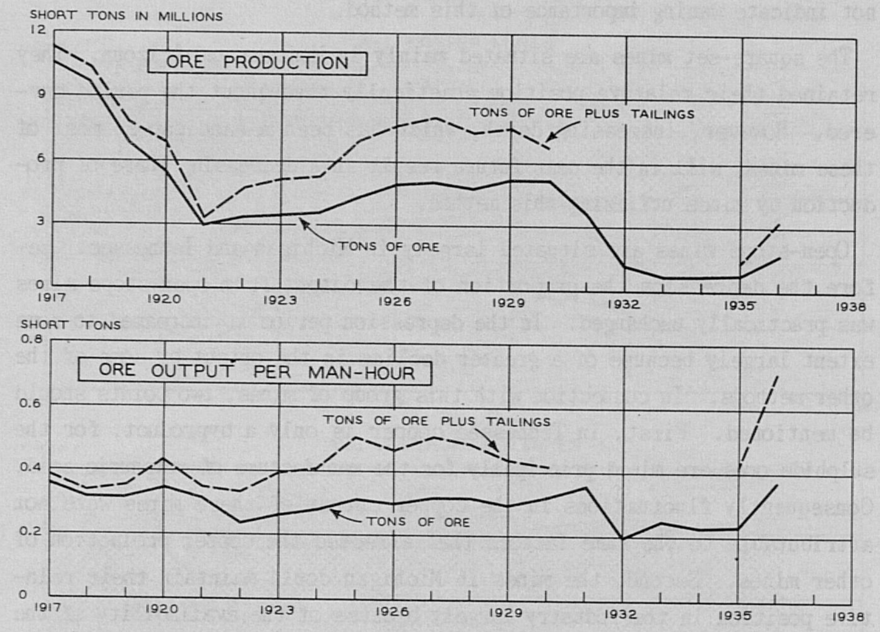
[8] If a lengthy shut-down is contemplated, all blocks which have been developed and from which ore is being drawn must be worked until all the ore is extracted before the block-caving mine can be closed down. Otherwise the costly development work, together with the ore in the developed blocks, may be a total loss.

[9] "Tailings" may be defined as a waste product from a mill or concentrator, or that part of the ore which remains after a portion of the valuable mineral is removed by some metallurgical process. It contains a small amount of valuable mineral which could not be economically recovered at the time the ore was concentrated but which might be subsequently recovered with improved separation techniques or as a result of other changes in the cost-price relationship.

Figure 27.- PRODUCTION AND OUTPUT PER MAN AT MICHIGAN COPPER MINES, 1917-36

(With and without tailings)



U.S. BUREAU OF MINES
BASED ON TABLE A-12

MINERAL TECHNOLOGY AND OUTPUT PER MAN STUDIES
WPA-NATIONAL RESEARCH PROJECT E-206

Nearly all the cut-and-fill mines are to be found in Arizona and Michigan. The proportion of the output of copper from this group of mines was more or less constant during the period 1917-31. The sharp rise in 1932 and subsequent years was due principally to the fact that the output from mines employing this method declined less than that from mines using other methods (see bottom left section of figure 25). This group of mines would prevent its production from dropping as sharply and to such a low level as that of other methods largely because selective mining was practiced (confining exploitation to the richest portion of the ore) and because of the consolidation of the properties of two mining companies, Calumet and Arizona, and Copper Queen. The former, which had been using cut-and-fill as the principal method, continued to do so; the latter, which had previously been employing mainly the open-cut and square-set methods, shifted to cut-and-fill as the predominant method. Under more normal conditions the proportion of the output contributed by the mines using the cut-and-fill method probably will return to the predepression level, and in the near future no great change in the relative importance of this method need be expected.

Most of the mines employing the shrinkage method are in Alaska, California, and Nevada. Since 1917 the volume of output from these mines declined almost continuously. Of the several major methods of mining, shrinkage has long since become the least important. In view of the scarcity of deposits that can be mined efficiently by this method, it will continue to be relatively unimportant in the near future.

INCREASING MINING DIFFICULTIES

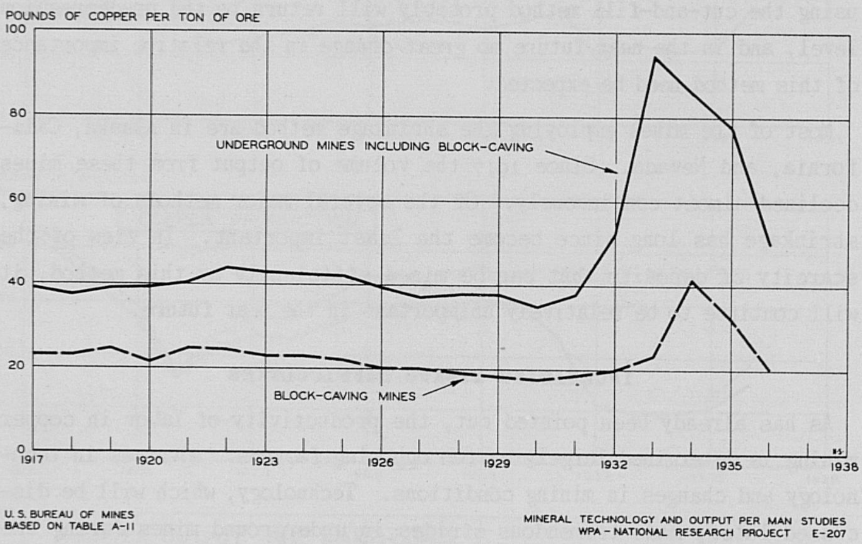
As has already been pointed out, the productivity of labor in copper mining is determined largely by two opposing factors: advances in technology and changes in mining conditions. Technology, which will be discussed later, made tremendous strides in underground mines during the period under review. At the same time, however, mining conditions were becoming increasingly adverse. Thus, in spite of vast improvements in technology, the output per worker in underground mines increased in the period since 1917 at only a moderate rate, indicating that cumulative mining difficulties were counteracting the influence of technology. In analyzing the effect of technologic developments on the output per worker, therefore, the increasing difficulties of mining must be given careful consideration.

Growing physical difficulties may manifest themselves statistically in a decline in the yield of metal and an increase in the depth of the mines. As the richer and more accessible deposits are exhausted, mining enterprises must resort to leaner ores or to deposits lying at greater depths or abandon the mines and migrate to a new area where mining conditions are more favorable.

Yield of Ore[10]

The changes in the average yield of ore from underground mines for the period 1917-35 are indicated in figure 28. In the period 1917-30 the trend of yield was slightly downward, declining at an average annual rate of about 0.3 pound per ton. During the depression the yield rose sharply from 35 pounds in 1930 to 95 pounds in 1933, largely as a result of intensive differential mining and the closing of block-caving mines when the price of copper was falling rapidly.

Figure 28.- YIELD OF ORE AT UNDERGROUND COPPER MINES, 1917-36



Figures representing the average yield for a large group of mines, however, may not reflect the full extent of the change in the quality of the ore over a period of time. In the first place, the change in the quality of ore may be obscured by the fact that, as a result of technologic improvements in ore dressing and metallurgy, the recovery of copper has become increasingly more efficient. Whereas in the 1910's only about

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[10] The discussion here is confined to underground mines during the period 1917-35. For an analysis of the yield of ore for all copper mines and for principal copper-mining districts over a much longer period than that covered in this report see A. V. Corry and O. E. Kiessling, *Mineral Technology and Output per Man Studies: Grade of Ore* (WPA National Research Project in cooperation with U. S. Department of Interior, Bureau of Mines, Report No. E-6, Aug. 1938), pp. 36-56.

75 percent of the valuable minerals was recovered, today a recovery of over 90 percent is not uncommon. Thus the decline in the quality of the ore mined may be concealed by the gain in the efficiency of recovery. In the second place, the change in the quality of ore may be hidden by the factor of selective mining. In mining a body of ore the miner must distinguish ore from waste. Ore and waste are rocks, the difference between the two being that the former contains more valuable mineral than the latter. Thus mining is always selective to some extent. If a high degree of selectivity is resorted to, the yield will be high, but the mine is to that extent depleted of its richest ore, leaving the poorer grades for later extraction. It should be borne in mind, therefore, that because of this element of selective mining the degree of depletion of ore in a group of mines may not be completely revealed by figures for average yield of ore.[11] However, since mining enterprises usually begin with the richest ore[12] and turn to lower grades as the former is exhausted, it may be assumed that the rich ores are gradually depleted and that over a long period a declining trend in the yield of ore such as that for underground mines during the period 1917-30 is an indication of this phenomenon.[13]

Figure 28 shows the average yield of ore mined by the block-caving method. As this ore was largely of the porphyry type, the yield was considerably lower than that for all underground mines. Generally it ranged from 18 to 25 pounds, although in 1932 it was as low as 11 pounds per ton of ore at one mine and as high as 40 at another. The trend in the period 1917-30 showed a slight decline. The rise during the depression was due not so much to selective mining as to the closing down of particular mines in this group.

Increasing Depth

Increasing depth, so characteristic of mining with the passage of time, is one of the major difficulties that tend to neutralize the economies

[11] The increasing amount of hand sorting at the face in the same mines tends to disguise the change in the quality of the ore. As only such broken rocks having the requisite amount of valuable mineral are selected as ore, this practice would naturally hide the evidence of depletion which would otherwise have been disclosed in the figures for the average yield of ore. The application of closer engineering control over mining operations has also resulted in obscuring an actual decline in the grade of ore. In some of the modern mines practically all mining operations are under the supervision of a staff of engineers who direct the workers where to mine and what to regard as ore. Samples of ore are constantly taken and analyzed at the laboratory to determine and control the grade. In the case of caving mines, the drawing of ore is discontinued as soon as the samples indicate that the grade has fallen below a certain minimum. In vein mines, similarly, the broken rocks which do not have the required metal content are left underground as waste.

[12] Since there has been secondary enrichment in many of the copper deposits, the miners who begin extraction at the top would obviously reach the richer part of the ore first.

[13] Although a decline in yield may reflect a progressive depletion of high-grade ore, it may also indicate that technologic advances have enabled mining companies to extract low-grade ore formerly regarded as waste, thus enlarging rather than exhausting the recoverable ore reserves of the Nation.

resulting from improvements in mining methods and equipment. Together with depletion, it has forced many mining enterprises to abandon their workings and to migrate to new fields. Problems arising from increasing depths have been particularly acute in deep-vein mines of Michigan, Montana, and Arizona. Although no accurate statistics are available on the depth of mines, the necessity of going progressively deeper underground to extract copper deposits is indicated by the gradual increase in the depth of shafts sunk. Changes in the average and maximum depth of shafts in vein and in block-caving mines are shown in table 4. In the 30-year period covered the average depth of shafts increased by 50 to 100 percent in the underground mines of these States.

Table 4.- DEPTH OF SHAFTS IN UNDERGROUND MINES IN MICHIGAN, MONTANA, AND ARIZONA, 1905-35[a]

Type of mine and State	Depth of shaft (feet)						
	1905	1910	1915	1920	1925	1930	1935
Vein mines							
Michigan, maximum depth (inclined shaft)	5,289	7,995	8,132	8,360	9,065	9,300	9,300
Montana							
Average depth	2,005	1,986[b]	2,456	2,621	2,763	2,799	3,155
Maximum depth	2,450	2,956	3,400	3,800	3,800	3,800	4,100
Arizona							
Average depth	892	984	1,238	1,445	1,719	1,931	1,954
Maximum depth	1,700	1,700	2,000	2,300	2,630	2,915	3,150
Porphyry (block-caving) mines							
Arizona							
Average depth	450	640	513[b]	564	596	629	642
Maximum depth	500	720	720	930	1,000	1,000	1,120

[a]Data are from an unpublished study, "Depth of Mines," by O. E. Kiessling, J. C. Burritt, C. N. Gerry, and T. H. Miller, prepared for WPA National Research Project "Mineral Technology and Output per Man Studies."
[b]The decrease in the depth of shafts since the preceding specified year is explainable by the advent of new mines.

Increasing depth has created a host of difficulties, all of which have a strong tendency to retard the rise in the output per worker and to increase mining costs. Heavy ground in some mines calls for extensive artificial supports which are costly to construct. With greater depth the weight of the hoisting cable increases, thereby necessitating a reduction in the load to be hoisted. The choice of mining methods and machinery, too, is restricted. Rock bursts and air blasts resulting from excessive strains imposed by the weight of the overlying rocks add to the hazards of mining. Moreover, as the mine descends to lower levels, the distance that water must be lifted increases, requiring more powerful and a larger number of pumping units.

Other major difficulties confronting deep mines are high temperature, high humidity, and dust. The presence of fine dust has an adverse effect on the efficiency as well as the health of miners. The moisture content of the air, which generally increases with depth, not only makes it uncomfortable and fatiguing for the miners and thereby lowers their output but also accelerates timber decay and thus increases the volume of maintenance necessary to prevent cave-ins and rockfalls. By far the most formidable obstacle to greater labor productivity in deep underground mines, however, is the high temperature which, together with humidity, causes so much discomfort to the miners as to counteract part of the effect of technologic progress on output per worker.

As temperature and humidity rise with increasing depth, operators must resort to the costly method of mechanical ventilation.[14] The Anaconda Copper Company, for example, began mechanical ventilation of its Butte mines about 1910, when the average depth was 2,000 feet. By 1913 it was using 2,500 electric horsepower for driving ventilating fans;[15] by 1933, when the operating depth exceeded 3,000 feet, it was utilizing approximately 8,500 horsepower.[16] At present, with the depth at about 3,500 feet, it is employing about 10,000 horsepower and moving approximately 3,000,000 cubic feet of free air per minute.[17] The cost for power alone amounts to about $350,000 per year.

IMPROVEMENTS IN THE ART OF MINING

The rise in the output per worker, despite mounting physical difficulties, is principally attributable to the developments of mining technology - improvements in the art of mining and progress in mechanization. This section will be devoted to an examination of the changes in the art of mining that have influenced the output per worker.

Selection of Mining Methods

The choice of a mining method and the ability to adapt it to a given set of conditions may have as much influence on labor productivity as the improvements in the mining methods. In the earlier days the open-stope and square-set methods were mainly employed. As competition became increasingly more acute, greater stress was laid on the reduction of costs through the improvement of existing mining methods, the introduction of

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[14] The subject of ventilation is discussed later in this chapter.

[15] J. Gillie, "Use of Electricity in Mining in the Butte District," *Transactions of the American Institute of Mining Engineers*, Vol. XLVI (1914), p. 818.

[16] W. B. Daly and A. S. Richardson, "Experimental Air-Conditioning for the Butte Mines," *Transactions of the American Institute of Mining and Metallurgical Engineers*, Vol. 109 (1934), p. 232.

[17] W. B. Daly, "Reducing Health and Accident Hazards in Industry" (address before the 26th Annual Meeting of the Chamber of Commerce of the United States, Washington, D. C., May 3, 1938). The term "free air" is used here as distinguished from compressed air.

new methods, and the adoption of better equipment for drilling, blasting, mucking, hauling, hoisting, pumping, ventilating, and other functions connected with underground mining. Mining has become more and more an engineering problem. Instead of having a miner plan and execute the work, as he formerly did, the operator now has a corps of experts to investigate and lay out every detail and to supervise operation of the mine according to a well-conceived plan. Before a method is selected engineers make a thorough study of the ore body. Care is exercised in selecting the most economical method, for a wrong choice may mean the failure of the mining venture.

The factors that govern the selection of a mining method may be classified in two broad categories: physical and economic. The first involves the consideration of the geologic structure and position of the ore body and the nature of the ore. The second concerns primarily the comparative costs of production and capital requirements of alternative methods. A third group should perhaps be added - the factors that affect the safety, health, and welfare of the miners. Of late, these factors have become important considerations in the selection of a mining method.

Among important characteristics of an ore body which influence the choice of a mining method are strength of the ore and the wall rock; size, shape, and dip of the deposit; and depth of the ore below the surface and character of the capping or overburden. The strength of the ore and wall rock determines the size of excavation, the length of time the roof of the mine can be left unsupported, and the character of support. The size and shape of a deposit govern the method of excavation. A small, narrow ore body, for example, may be worked from wall to wall without support, whereas a large, wide one in a similar type of ground may require that pillars be left or artificial supports be used. If the dip of the deposit is rather gentle, additional support may be necessary. If the ore body lies at great depth, strong supports may be required to counteract the high pressure transmitted from the overlying rock.

The chief characteristics of the ore that influence the selection of a mining method are its grade, hardness, and friability; mineralogical character; and the distribution of valuable minerals within the ore body. If the ore is of relatively low grade, a considerable quantity may be left in the form of pillars to support the roof; on the other hand, if the ore is of high grade, artificial supports are employed to reduce to a minimum the quantity left in the mine. A hard ore may be mined by methods utilizing little or no support, natural or otherwise, whereas a soft ore must be excavated by methods utilizing artificial supports. Distribution of the valuable minerals in the ore determines the degree of selectivity to be applied. Irregular occurrence of minerals and the presence of more than one valuable mineral in sizable quantities usually calls for a method that permits a high degree of selectivity. The mineralogical character of the ore may necessitate a modification of the mining method that otherwise would have been employed. Some sulphides, if broken and exposed to the action of the atmosphere, tend to oxidize, and oxidation may lower the recovery of metal in the ore-dressing process or cause mine fires. In such cases a method that permits the immediate removal of the broken ore would be employed.

Where geologic conditions permit the application of more than one method, the choice is dictated largely by the mining cost of alternative methods and the amount of capital required for both preliminary and current development work. Where the cost of timber is high, the methods requiring the least artificial supports are preferred. If the ore deposit is of low grade, it is essential that it be mined by a method that permits mass production. In large-scale operations adequate preliminary development necessitates a large outlay of capital but may result in lowering mining cost after the mine begins to produce at capacity. The shortage of available capital, however, may force the application of more costly methods which may result in quicker though smaller returns to the operator.

A method flexible enough to meet demand situations is desirable, for it permits expansion and contraction of production or variation in the grade of ore mined without seriously affecting mining costs. In mines that use the cut-and-fill or square-set methods because of existing geologic conditions, expansion can be attained only by having a large number of stopes prepared for mining. Consequently a large amount of development work is required ahead of actual mining. Such advance development ties up capital in periods of low copper demand and, because it entails additional maintenance expense, tends to raise mining costs.

Apart from humanitarian considerations, operators have come to appreciate the fact that safety work is an essential element in good mining practice. More and more they have realized that accidents and occupational diseases are important cost items that may be curtailed by safety work. Moreover, it has been found that in mines where hazards have been reduced to a minimum, the workers, feeling more secure, have been able to accomplish more with less fatigue. Greater safety, together with better working conditions, also increases the productivity of labor by reducing the amount of time lost owing to accidents and illness. Unless the mining method adopted is the safest one to use in a given instance, accident and disease prevention in mining may not be effective.

Advances in Mining Methods

Once he has selected the mining method that seems most appropriate for a given ore body the operator must adapt, modify, and improve it in accordance with changes in mining and operating conditions. Not only is each mine a problem in itself, requiring special adaptation of a basic method to suit differences in the character of deposits, but the method adopted at each mine is modified as the factors that determine the choice of the original method change with the passage of time. Consequently many variations of each basic method are in use. To cover all variations would be beyond the scope of this report. In this section, therefore, are considered the principal advances made by the five basic methods - open-stope, shrinkage, cut-and-fill, square-set, and block-caving - together with some of the more important variations of these methods.[18]

Open Stoping.- The open-stope method is the simplest and the oldest form of underground mining. It is well suited to many types of deposits and with its many variations is used rather extensively in Tennessee and Michigan. Generally it is employed for the mining of hard ore bodies with sound walls in which the roof or walls of the workings are sustained by pillars of ore or by artificial supports spaced at various intervals. Narrow deposits may be mined from wall to wall without any support. Among the main advantages of the method are low capital requirements and early returns on investment made possible by the fact that it requires relatively little development work at the beginning and during the period of production, feasibility of mining the ore selectively and following the irregularities in the ore bodies, and the relatively low cost per unit of output. The chief disadvantages lie in the necessity, at least in some instances, of leaving ore as pillars; the difficulty of closely inspecting the walls and roof in thick deposits, with the result that the accident rate is higher than in some other methods; and the likelihood that the ground eventually will cave in, thus removing the possibility of re-entry should it later become desirable to mine some of the ore left behind.

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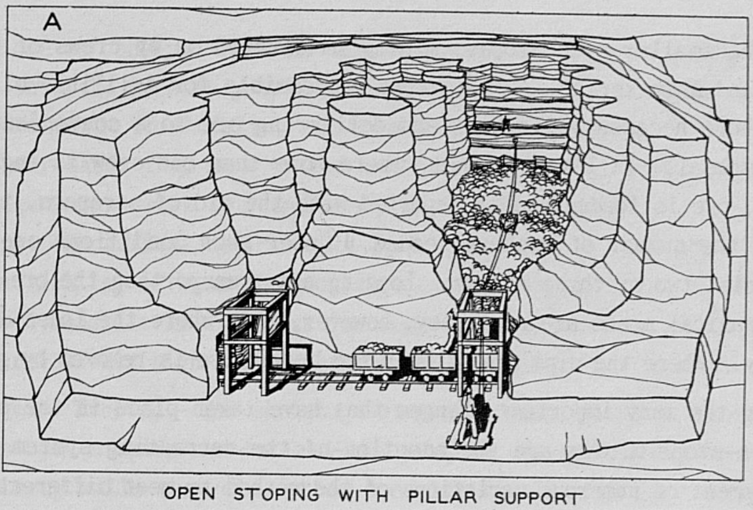
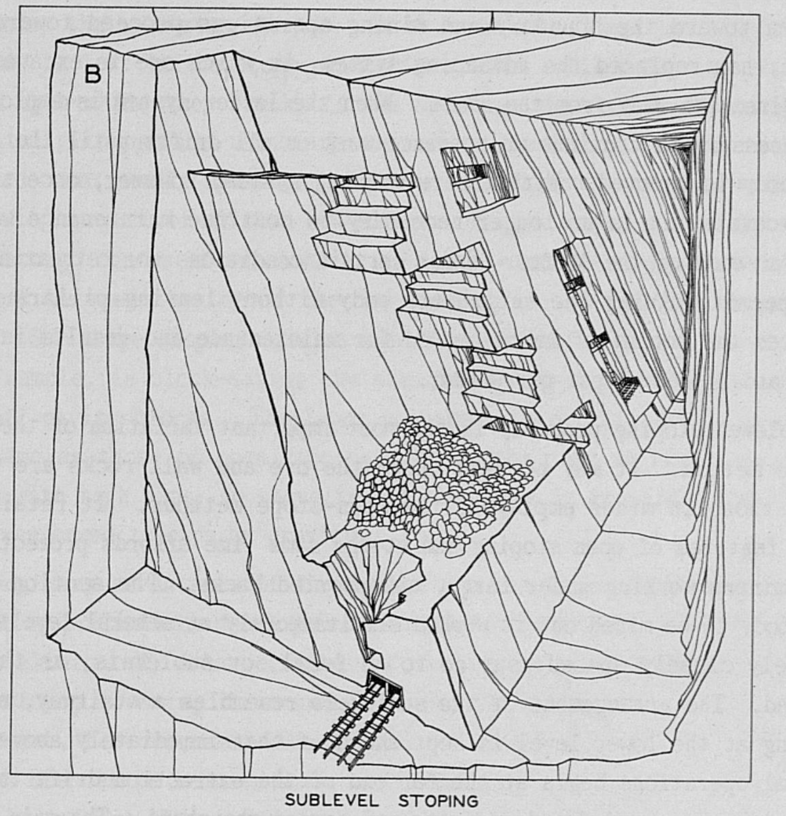
[18]Ensuing descriptions are intended to give the reader the more fundamental principles involved in the application of each of the five mining methods. For a more detailed description of these methods, the reader is referred to the following sources: C. F. Jackson and E. D. Gardner, *Stoping Methods and Costs* (U. S. Dept. Int., Bur. Mines, Bull. No. 390, 1936); E. D. Gardner, *Undercut Block-Caving Methods of Mining in Western Copper Mines* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6350, mimeo., Oct. 1930); C. F. Jackson, *Mining Ore in Open Stopes, Central and Eastern United States* (rev. ed.; U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6193, mimeo., Apr. 1931); C. H. Johnson and E. D. Gardner, *Cut-and-Fill Stoping* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6688, mimeo., Feb. 1933); E. D. Gardner and W. O. Vanderburg, *Square-Set System of Mining* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6691, mimeo., Apr. 1933); and C. F. Jackson, *Shrinkage Stoping* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6293, mimeo., June 1930).

Figures 29 to 33 are idealized illustrations of typical underground mining methods and are not necessarily exactly those practiced at any single mine.

Figure 29.- OPEN-STOPE METHODS OF COPPER MINING



OPEN STOPING WITH PILLAR SUPPORT



SUBLEVEL STOPING

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The upper diagram shows how pillars are left for natural support in the mining of wide, flat ore bodies. At the left of the pillars is a "worked out" stope. At the right are active mining operations showing the ore being drilled before being blasted from the benches (shown in the background) and transferred by scrapers to the loading chute (in the foreground).

The lower diagram shows how labor is saved by utilizing gravity for transferring the broken ore to the loading chutes. In this method the ore is broken from the benches and allowed to fall into mill holes, from which it is drawn through various chutes into cars located on the lower haulageway.

In the smaller open stopes, mining is carried on by crews of two - a miner and a mucker.[19] Each crew is responsible for drilling, blasting, and, where necessary, loading and delivering ore to a convenient point for disposal. In larger stopes, where more than one crew is engaged in mining, ore is loaded and transported with the aid of scrapers, thus reducing the number of muckers needed. Under such conditions one mucker may assist two or three miners. Loading and transporting the ore by hand or mechanical means are necessary, however, only where the lodes dip moderately. Where the dips are steeper the broken ore is removed by gravity.

Among the many important changes that have taken place in the practice of open-stope mining are the adoption of the retreating system and the development of numerous variations of the method to meet different mining conditions. The retreating system, in which a working tunnel is first driven toward the boundary and mining operations proceed towards the shaft, has replaced the advancing system, in which ore is excavated in the direction away from the shaft. When the latter system is employed it is necessary to continue maintenance work on all drifts until the entire ore body is removed. With the retreating system, however, once the ore is excavated it is no longer necessary to continue maintenance work in the far ends of the drifts. Under certain conditions the retreating system permits mining the entire ore body without leaving pillars. This reduces the amount of labor needed for maintenance and results in lower cost and higher output per worker.

Sublevel stoping probably is the most important variation of the open-stope method. It may be used where the ore and wall rocks are weaker than those in mines employing the open-stope methods. It retains the good features of open stoping and at the same time affords protection to the miners working under large, unsupported backs. The section of the ore body to be mined out is worked simultaneously at several levels (relatively closely spaced, say 20 to 40 feet), or sublevels, as they are called. The arrangement of the sublevels resembles a stairway, and the mining at the lower level is kept ahead of that immediately above. The actual operations begin at the far end of the extraction drift that has been driven on each level and retreats toward the shaft. The main advantage of the method lies in the fact that mining operations are conducted at a number of sublevels simultaneously; that the bench at each sublevel is wide enough or long enough to permit several men to drill at the same time; that, as a result of the concentration of ore, a better utilization of hoisting and transportation facilities is attained; that since the

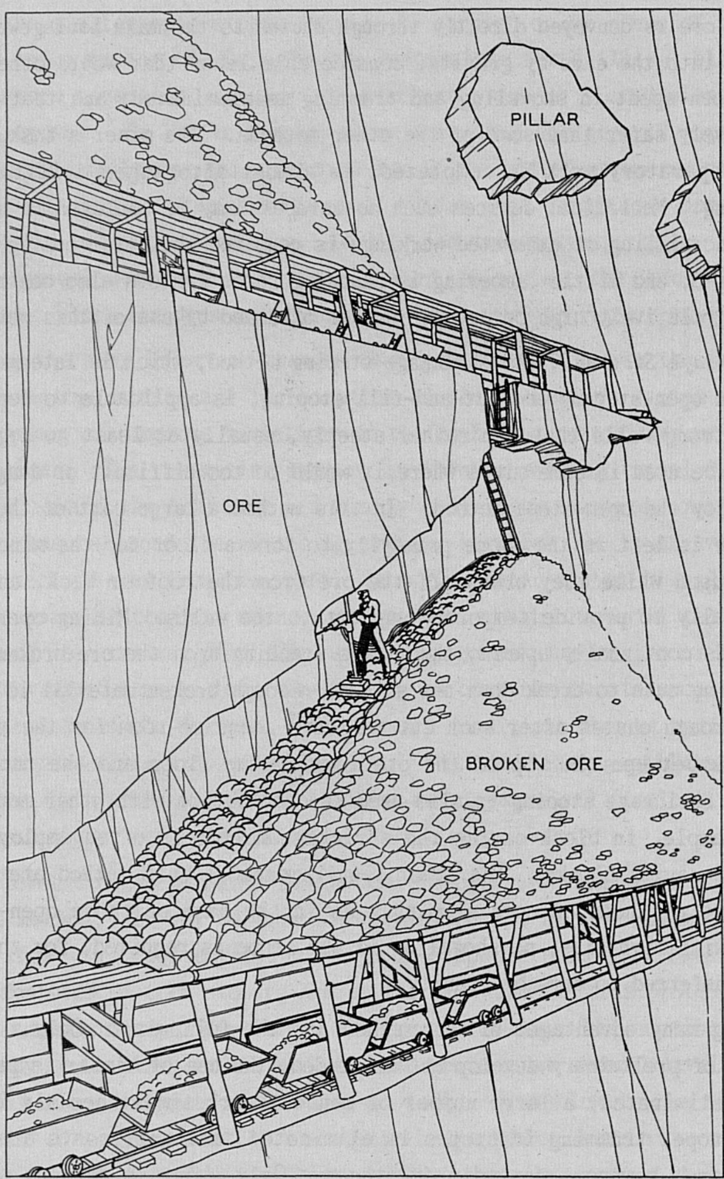
[19]The mucker assists the miner in erecting the drilling machinery, loads the cars and trams them to the nearest chute or main haulageway. He is also responsible for procuring needed materials and supplies.

broken ore is conveyed directly through chutes to the main haulageway and loaded into the cars by gravity, considerable labor that would otherwise have been spent in shoveling and tramming is eliminated; and that it is relatively safer than some of the other methods. The miner's task, once the preparatory work is completed, is almost altogether drilling and blasting. Mechanical devices such as scrapers may be employed if necessary, no filling of excavated workings is required while the ore is being extracted, and little timbering is needed. These factors also contribute to the relatively high output per worker attained by use of this method.

Shrinkage Stoping.- The shrinkage-stoping method, which is intermediate between open-stoping and cut-and-fill stoping, is applicable to deposits with strong walls that dip rather steeply, usually at least 50 degrees. It may be used in some mines where it would be too difficult or dangerous to employ the open-stope method. In this method a large part of the broken ore is left in the stope primarily to form a floor for the miners to stand upon while they break off the ore from the roof or back, and incidentally to provide temporary support to the walls. Mining operation proceeds continually upward, the miners standing upon the ore broken from preceding cuts to break down new slices. Enough broken material is drawn off through chutes after each cut to leave adequate room for the miners to work between the top of the ore forming the floor and the unbroken back. Shrinkage stoping is also used in combination with other methods. For example, in block-caving the shrinkage method is often employed to cut off caving blocks. The labor requirements of this method are about the same as those of open stoping. As in the case of those open-stope mines with high dips, no shoveling in the stope is required, for the ore is transferred to cars by gravity.

Among many advantages of the method are the following: Only a small amount of preliminary development is needed, the use of timber is practically eliminated, a large number of men can work simultaneously in the same stope, tramming in stopes is eliminated, and unit costs are low. There are, however, many disadvantages. Only about 40 percent of the broken ore can be removed from the stope while excavation is going on; dilution of ore with waste is likely to occur, particularly where the walls are not firm enough; sorting of the ore is not feasible; storage of the ore in the mine ties up capital; filling of empty stopes is often required; subsidence of broken rock hanging up in the stopes may cause serious accidents; the oxidation of sulphide ores that are left exposed to the atmosphere is likely to become a fire hazard and may tend to decrease the recovery of metal in the milling processes; and travel and

Figure 30.- SHRINKAGE METHOD OF COPPER MINING



INCLINED SHRINKAGE STOPING

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Cross-sectional view of inclined shrinkage stoping as applied in Michigan. During the breaking process the bulk of the broken ore remains in the stope to provide a working floor and support for walls. Only enough ore to provide working room between the top of the pile and the back of the stope is drawn off into cars on the lower level. Cars shown are the open-end type used in hand tramming. Access to the stope is by means of a raise driven to the upper level. Cross sections of horizontal pillars of barren rock left to support the walls are shown at the upper right of the diagram.

handling of supplies and equipment in shrinkage stopes are difficult, resulting in a considerable loss of time.

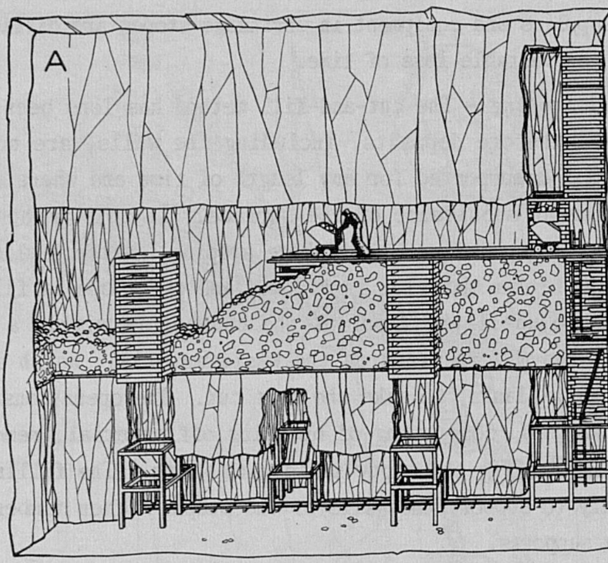
Cut-and-Fill Stoping.- The cut-and-fill method has long been employed in mines where the ore deposits, including the walls, are not strong enough to remain unsupported for any length of time and where it is desirable to prevent subsidence of the surface. In this method, mining operations proceed upward as in shrinkage stoping. After a slice of ore is cut off, the broken material is removed and the stope is filled with waste from above until the floor of the chamber is within a few feet of the roof. The miners stand on the waste material, which is generally covered with planks, to make the next cut. The operations are thus conducted in cycles, consisting of breaking off material, removing the broken ore, and filling the empty space with waste. The filling is introduced mainly to support the walls of the stope. Often timber is used for auxiliary supports.

Cut-and-fill stoping may utilize two variations: horizontal and the inclined or rill. In the horizontal cut-and-fill method the back and filling are maintained practically horizontal. In the inclined or rill method the back and filling are kept parallel to each other and the stope faces are inclined at about the angle of repose of the waste material; this permits the use of gravity to remove the ore and to fill the excavated workings with waste material. There is a variant of the inclined method known as the sublevel inclined cut-and-fill. In this method a block of ore, usually 100 feet in height, is first subdivided into three horizontal sections by driving drifts throughout the length of the block. Mining operations begin in the upper section and retreat in a horizontal direction toward the shaft. The excavation in the upper section is kept ahead of that in the middle section, which in turn is kept ahead of that in the lowest (third) section. After the ore is broken off and removed, each excavated section is filled up to the roof with waste material brought from above by gravity. Only several feet of space are left unfilled between the inclined working face of the unbroken rock and the inclined surface of waste formed by the angle of repose of the material.

Introduction of the inclined cut-and-fill method has permitted utilization of gravity in filling excavated stopes and in removing broken ore. Together with the application of the retreating system, these improvements have resulted in a substantial economy of labor.

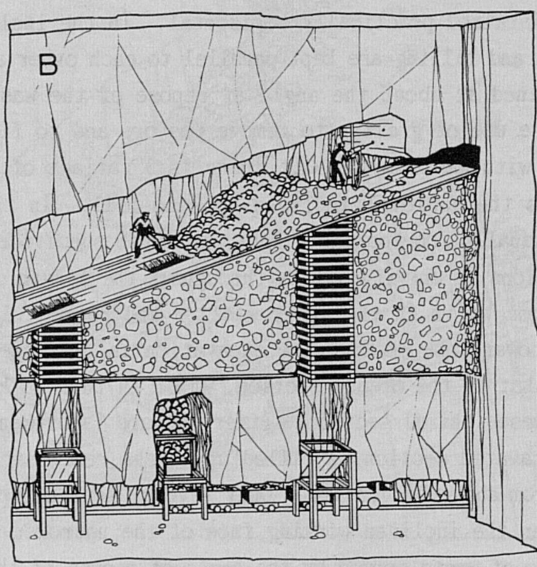
Other important advances that have also greatly reduced time and labor include application of scrapers for transporting ore to the chutes and spreading filling material and the use of sand tailings for filling material. Tailings are pumped to the stopes through large rubber-lined

Figure 31.- CUT-AND-FILL METHODS OF COPPER MINING



HORIZONTAL CUT-AND-FILL STOPING

DISTRIBUTION OF WASTE FILLING BY HAND TRAMMING



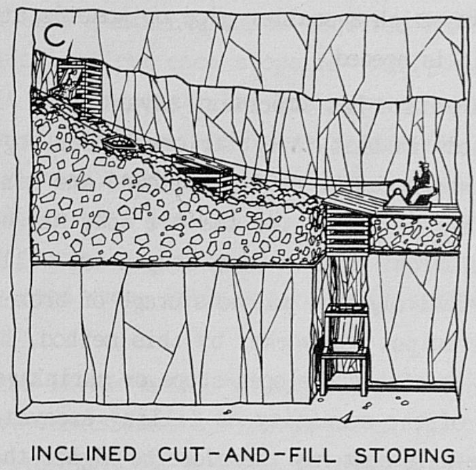
INCLINED CUT-AND-FILL STOPING

BREAKING AND REMOVAL OF ORE

Diagram A shows the preparation of a horizontal stope for the next cutting operation. After the ore has been removed, ore passes (criblike structures at the left) are extended to the mining floor, and the empty space is filled with waste. Waste is passed through the waste raise (upper right), loaded into cars, trammed by hand (center), and dumped at the end of the track. Filling proceeds until the empty space (left) is filled level with the track.

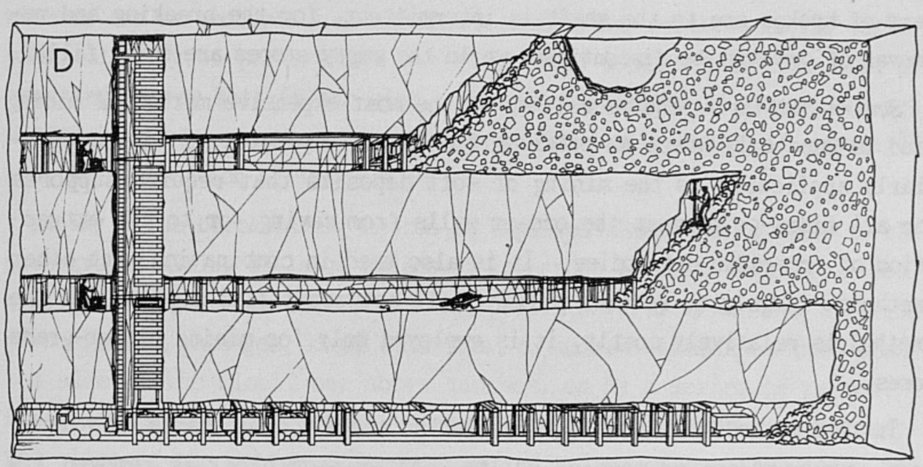
Diagram B shows how gravity is utilized to help transport the ore to the chutes. After the ore is drilled (by man at the right), it is blasted down onto a planked floor where it slides down to the ore chute; here the mucker (shoveler in the center) may or may not sort out the waste before shoveling the ore into the chute.

Figure 31.- CUT-AND-FILL METHODS OF COPPER MINING - *Continued*



INCLINED CUT-AND-FILL STOPING

DISTRIBUTION OF WASTE FILLING BY SCRAPERS



SUBLEVEL INCLINED CUT-AND-FILL STOPING

RETREATING SYSTEM

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Diagram C shows the application of scrapers for distributing waste in an inclined cut-and-fill stope. Waste is introduced from above through a waste raise (left) and falls into the stope where it is dragged into place by a scraper.

In sublevel inclined cut-and-fill stoping (D) considerable time and labor is saved, for the waste material is moved by gravity into the excavation immediately after the small supporting pillar is blasted down. The ore is removed in a series of inclined slices, leaving a small horizontal pillar of rock 2 to 4 feet thick to support the fill in the worked-out level above. The broken ore from these slices is then dragged to the ore chute (left) by scrapers. The pillar is blasted, and the waste runs in from above, completely filling the excavated section. Most of the ore from the pillar runs to the bottom of the pile where it can be picked out. The cycle is then repeated until the entire block is mined out. When stoping operations have progressed about 45 feet toward the shaft, mining is started on the next lower sublevel. Barren ground (upper block, right) is left as a horizontal pillar.

pipes, at the end of which a smaller pipe or hose is attached to direct the sand wherever it is needed.

The cut-and-fill method has important advantages. It is one of the most flexible methods in that irregular ore bodies together with their offshoots and stringers can be easily followed and mined. Part of the development costs may be saved, for the ore body may be explored while the mine is in operation. The working stopes are well ventilated, the accident rate is relatively low, and no storage of broken ore in the mine is necessary. The output per worker by this method, however, is lower than that attained by either the open-stope or shrinkage-stoping method, primarily because of the necessity of filling excavated workings with waste material; the unit cost may therefore be higher than at mines where other methods are used. Another disadvantage is the fact that the delivery of broken ore to the shaft is intermittent, for the breaking and removal of ore must be discontinued while the empty stopes are being filled.

Square-Setting.- Square-setting is the most expensive method of mining and is used only where no other method can be employed. It is particularly applicable to the mining of soft deposits that require supports on all sides to prevent the ore or walls from caving, or to the extraction of irregular ore bodies. It is also used in combination with other methods, largely to recover pillars left between filled stopes. As the method is relatively costly, it is employed only for mining higher-grade ores.

In this method small rectangular blocks, about 8 cubic yards in volume, are excavated one or more at a time. After each block is removed its place is taken by framed timber sets that serve as temporary supports until the cavity is filled with waste material. The operation may proceed upward, laterally, or (less often) downward, depending principally on the intensity and direction of the rock pressure, the size and shape of the ore body, and the amount of sorting required.

The labor requirements are much greater in the square-set than in any other method, and they are of a more specialized character. Small workings do not permit the use of scrapers and loaders and the greater part of the operations must be done by hand labor. Furthermore, much labor is needed in framing timber sets, transporting them to the mine, and assembling them.

Square-setting is the most flexible of all methods. It permits a high degree of selective mining and consequently close control of the grade of ore mined. It also enables the miners later to work the low-grade ore that might have been left in the first mining. It has an additional advantage in that only a small space is required for mining operations. Because the production from each stope is small, many stopes must be worked to obtain a large output. The handicaps of the method lie in the relatively high mining cost, intermittent production, low productivity, slow rate of extraction, and relatively high accident rates. The accident rate is higher than that of some other methods, mainly because square-setting is generally employed in bad ground where no other method can properly serve and where it is necessary to handle heavy timber and to use sharp tools.

The major changes in the method during the past two or three decades have been the improvements and standardization of the timber designs that have resulted in transferring the task of cutting timber and constructing sets from the mine to surface shops where more efficient mechanical methods can be employed. Standardization also has resulted in the interchangeability of parts which has simplified considerably the task of erecting timber sets in the mine. Several variations of the square-set method have been developed to permit a fuller utilization of gravity[20] and the application of scrapers.[21]

Caving.- The block-caving method is employed almost exclusively in the underground mining of large bodies of low-grade porphyry deposits. Generally, a large block of ore is first partly severed by driving a series of horizontal passages, known as drifts, or narrow shrinkage stopes along the side of the block, one above another, or by a series of raises.[22] The block is then undercut by removing, partly or completely, a horizontal slice at the bottom. The unsupported column of ore then caves and breaks under its own weight, and the broken ore is drawn off gradually from below. The development work is conducted on three levels - the level at which the ore is undercut, the "grizzly" level at which the ore is drawn, and the haulage level on which the ore is transported to the shaft. After caving has begun, however, the work is confined largely to the grizzly and haulage levels.

In block-caving operations considerable labor is needed for preliminary development, but labor requirements become relatively small when the

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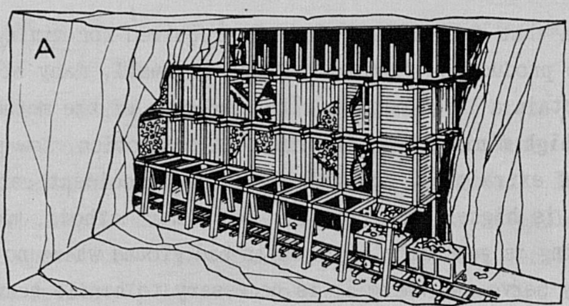
[20] At the United Verde Extension mine, for example, gravity is utilized to convey the ore to the chutes by spacing the chutes closely and placing slides with grizzlies in adjoining sets (see R. L. D'Arcy, *Mining Practice and Methods at the United Verde Extension Mining Company, Jerome, Ariz.* [U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6250, mimeo., Feb. 1930], p. 7).

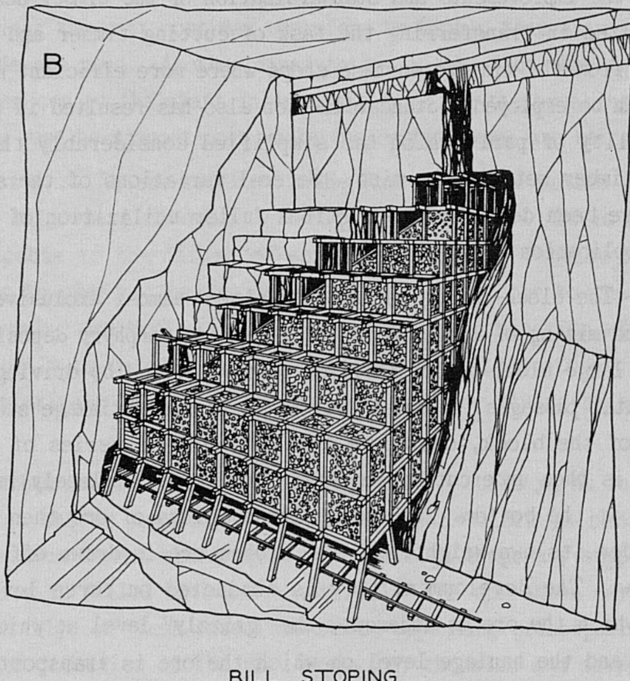
[21] Scrapers have been successfully used in square-set sections of the Ruth mine, Nevada Consolidated Copper Company (see C. F. Steinbach, "The Use of Scrapers in Underground Loading at the Ruth Mine," *The Mining Congress Journal*, Vol. 14, No. 10 [Oct. 1928], pp. 767-70).

[22] A raise is a vertical or inclined passage connecting two or more working levels underground.

Figure 32.- SQUARE-SET METHODS OF COPPER MINING



HORIZONTAL SECTIONS

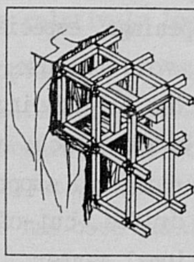


RILL STOPING

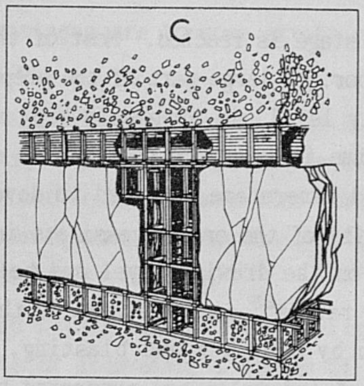
In square-setting, timber sets (boxlike structures) must be inserted to support the ground upon the removal of small ore blocks. The most common variation is using horizontal sections (A). Excavation proceeds horizontally from wall to wall and progresses upward level by level. After mining has advanced two sets high, the ore is broken on the upper or mining floor (top) and falls to the next lower level (shoveling floor) where the waste is sorted out and the ore shoveled into the chute.

Timbered rill stoping (B) utilizes gravity to transport the broken ore to the haulageway and to distribute waste filling. Ore is removed in small blocks as in ordinary square-setting, but the excavation proceeds by removing the blocks on the incline - step fashion. Waste filling is introduced at the top of the stope through the waste drift (top).

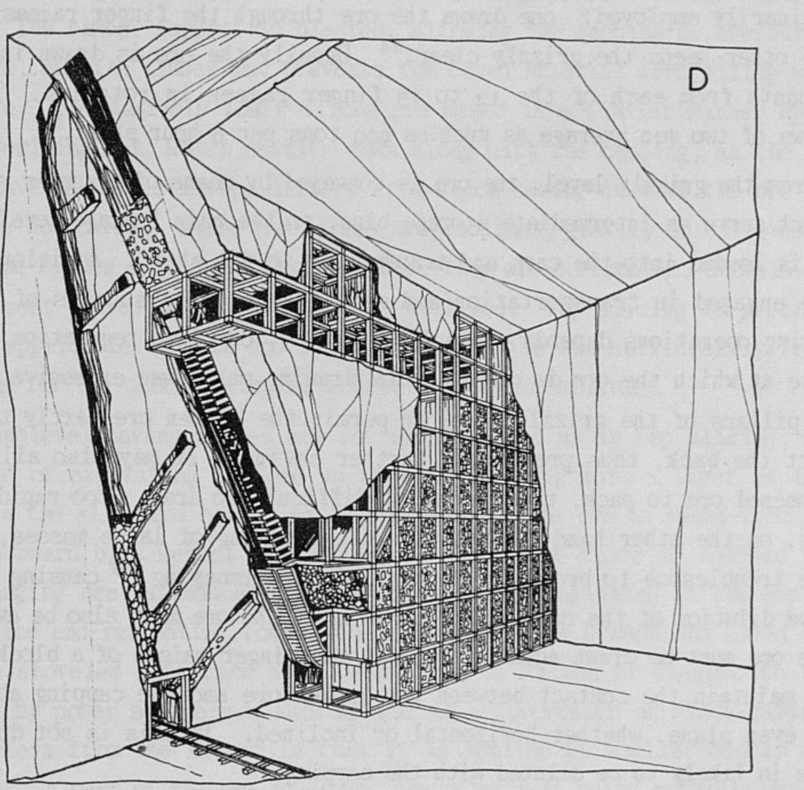
Figure 32.- SQUARE-SET METHODS OF COPPER MINING - *Continued*



DETAILS OF SUSPENDED SETS



DOWNWARD VERTICAL SECTIONS FOR RECOVERING PILLARS AND CROWNS



VERTICAL SECTIONS

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Vertical sections (D) are advantageously used for mining high-grade ore and in deposits where the hanging wall is heavy. In this method squares are removed vertically, beginning the operation on the hanging-wall side (back). The ore is broken by drilling and blasting, falls to the next lower level, is hand-sorted if necessary, and the mineralized pieces are shoveled into ore chutes. Waste is brought through waste raises (upper left), trammed to the working stope by hand, and dumped.

In downward vertical sections (C) the small blocks are mined vertically and downward. Hanging suspended sets (detailed drawing to the left) are inserted in the excavated space. Ore is raised to the haulage level by small hoists.

production stage is reached. Most of the development work has to be done by hand labor. A large part of the preparatory openings, especially above the grizzly level, is purposely kept small to avoid the use of timber support; thus it is seldom possible to employ mechanical loading methods. Experienced miners are required to develop ore blocks which, because of the fragility of the ore, necessitate immediate artificial supports, particularly on the drawing level and below. The drifts, cut-off stopes, and finger raises[23] are developed in the conventional manner. The rocks are broken by drilling and blasting, the broken material is removed, and the roof and walls are supported by timber. The finger raises are equipped with chute gates at the grizzly level in order that the rate of drawing ore may be controlled. In the drawing of the ore two men are ordinarily employed: one draws the ore through the finger raises while the other keeps the grizzly clear.[24] Usually the ore is drawn in equal amounts from each of the 12 to 15 finger raises in rotation. Tapper crews of two men average as much as 400 tons per 8-hour shift.[25]

From the grizzly level, the ore is conveyed by means of transfer raises, which serve as intermediate storage bins, to the main haulage level where it is loaded into the cars and transported to the shaft. Additional men are engaged in transportation and maintenance. The success of block-caving operations depends, to a large extent, on close regulation of the rate at which the ore is drawn. Slow drawing may cause excessive weight on pillars of the grizzly level or permit the broken ore partly to support the back, thus preventing further caving. It may also allow the loosened ore to pack, thus making it difficult to draw. Too rapid drawing, on the other hand, may result in the caving of large masses, which are troublesome to break up, or may induce chimneying,[26] causing excessive dilution of the ore. Uneven drawing of the ore must also be avoided. The ore must be drawn equally from all the finger raises of a block so as to maintain the contact between the broken ore and the capping above on an even plane, whether horizontal or inclined. If this is not done the ore is likely to be diluted with the capping.

The caving method is not without defects. It precludes selective mining and may involve a loss of ore amounting to as much as 20 percent of

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[23]Finger raises are inclined passages through which broken ore is drawn from the caved block to the grizzly level.

[24]A grizzly is a grating placed below the chute gate to prevent large fragments of ore from falling into the chute and blocking the passage. It may consist of a group of parallel wooden bars, steel bars, or rails spaced 10 to 12 inches apart.

[25]E. D. Gardner, C. H. Johnson, and B. S. Butler, *Copper Mining in North America* (U. S. Dept. Int., Bur. Mines, Bull. No. 405, 1938), p. 164.

[26]When ore is drawn too rapidly through a small chute, the ground immediately above moves downward, forming a relatively small channel (chimneylike in shape) into which the capping flows. This dilutes the ore by forming pipes of waste in the main ore body.

the total.[27] However, its disadvantages are to some extent compensated for by the relatively low mining cost.

The adoption of block-caving has been one of the most significant events in the copper-mining industry. It has permitted the exploitation of large bodies of low-grade ores that could not have been worked profitably by any other method. As a consequence, a considerable quantity of copper ore has been added to the Nation's reserves.

Two other caving methods are used rather widely: top slicing and sublevel caving. Top slicing consists of excavating a series of horizontal or inclined slices, beginning at the top of the ore deposit and working successively downward. While the slice is being removed timber supports are placed, and usually, when the slice is completed, a timber floor is laid. The timber support is blasted, allowing the capping or overburden to cave, and the timber mat prevents the caved material from mixing with the ore. Successively lower slices are mined in a similar manner up to the overlying mat, which settles down along with the capping, as the ore is excavated. This method differs from block-caving in that the ore itself is not caved. Where the slices are inclined, gravity is enlisted to move the broken ore to the chutes; where they are horizontal, shovels and (in recent years) power scrapers are utilized. Top slicing is particularly applicable to soft and sticky ore which is not sufficiently free-running for the block-caving method to operate efficiently.

In sublevel caving slices are cut and timbered as in top slicing; but instead of excavating the ore up to the overlying mat, a layer is left between the slice and back. As in top slicing, the ore is mined from the top downward by a series of sublevels. After the slice is removed the overhanging ore is induced to cave by blasting, beginning at the end of the slice and retreating toward the entrances. The broken and caved ore may be shoveled into cars and trammed to the raises or dragged to the raises by power scrapers. This method is a forerunner of block-caving; it differs from the latter in that it is applied to smaller blocks and that only a part of the ore is caved, the other part being actually mined. The vertical distance of a sublevel may range from 18 to 30 feet, whereas the height of a block-caving block may range from 40 to over 300 feet. Sublevel caving is applicable where it would be dangerous to use the top-slicing method because of the tendency of the capping and mat to arch and hang up.

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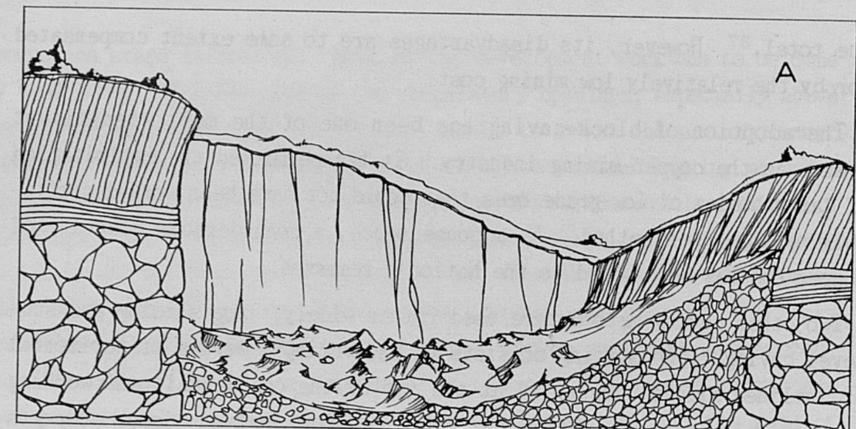
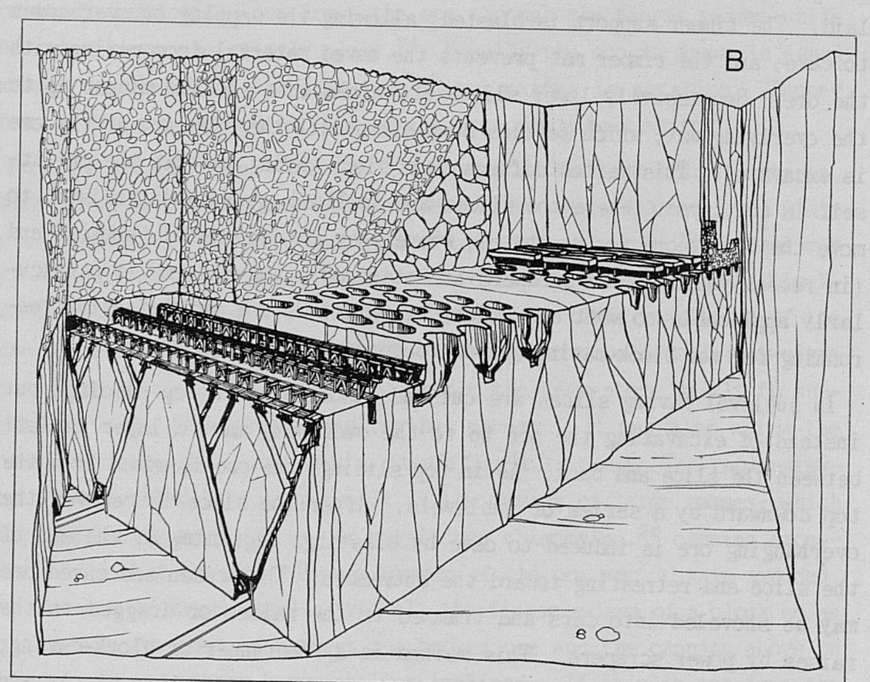
[27] A part of the ore left in the mine may be recovered later by leaching after underground operations have been completed. Several of the block-caving mines are planning to do so.

Figure 33.- CAVING METHODS OF COPPER MINING



SUBSIDENCE OVER CAVED BLOCK

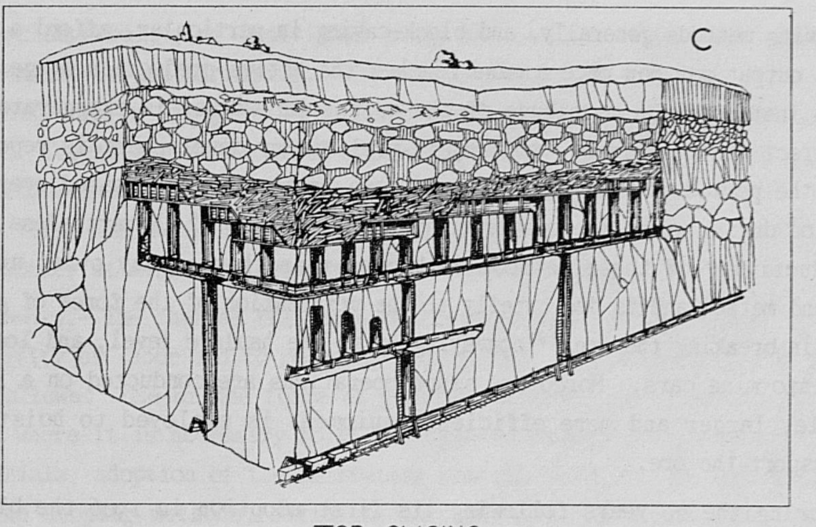


BLOCK—CAVING

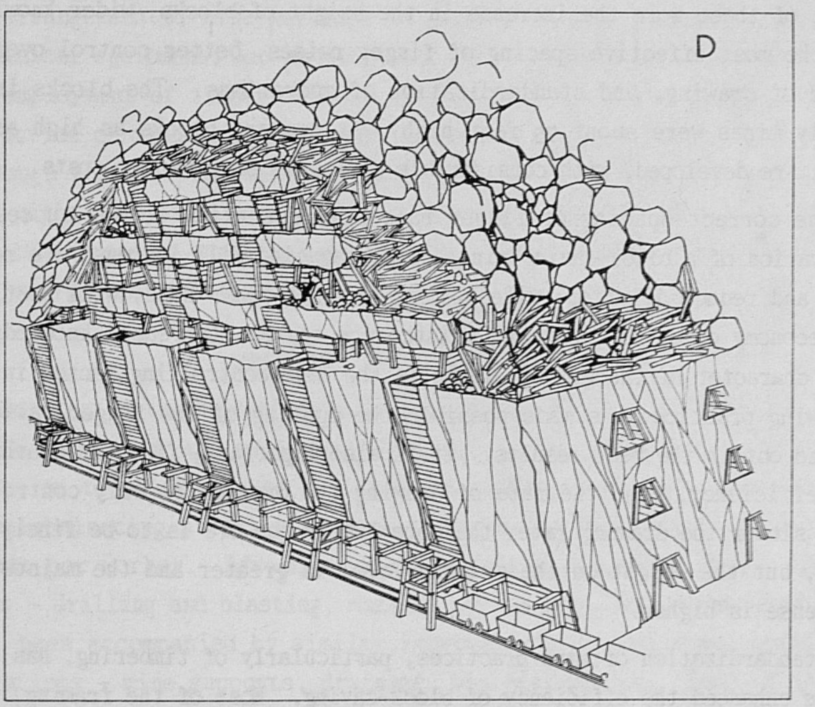
In block-caving (B) gravity is utilized to break and transport the ore to the loading chutes. The ore block is isolated on the sides and ends by boundary shrinkage stopes (extreme right) and undercut on the bottom by removing a horizontal slice (left of boundary shrinkage stope). Ore is drawn through finger raises (craterlike holes in the center) onto grizzly level (match-like maze in left middle) and diverted into transfer raises (lower left) through which the ore falls to the haulageway (bottom left).

The upper diagram (A) shows how the surface is damaged by the removal of ore underground.

Figure 33.- CAVING METHODS OF COPPER MINING - *Continued*



TOP SLICING



SUBLEVEL CAVING

Top slicing (C) is differentiated from other mining methods in that the excavation starts at the top of the ore body and advances progressively downward. The ore is broken by drilling and blasting and is excavated in slices, each successive slice being mined out right up to the overlying timber mat (section below the boulders) which prevents the mixture of ore with waste and furnishes a protective roof for the miners.

Sublevel caving (D) has features common to top slicing and block-caving. As in top slicing, a timber mat prevents ore from mixing with the waste. Relatively small blocks of ore are undercut by excavating a small slice of ore from beneath the block; after its removal, the block is allowed to cave. Ore is transported to the chutes by hand or scraper.

Caving methods generally, and block-caving in particular, afford a very high output per man once a mine reaches the active-production stage. In some instances the magnitude of the output per worker is exaggerated by the fact that the labor expended in preliminary development is reported for the period in which the work is done, rather than prorated over the period during which the ore or copper was produced.[28] Nevertheless, the productivity of these methods is higher than that of any other underground method and is due chiefly to the utilization of the force of gravity in breaking the ore,[29] conveying it to the haulage level, and loading it into mine cars. Moreover, since operations are conducted on a large scale, larger and more efficient equipment is employed to hoist and transport the ore.

During the 20 years following its first adoption in 1906 the block-caving method underwent a series of improvements. Among the more important of these were the increase in the height of blocks, wider knowledge of the most effective spacing of finger raises, better control over the rate of drawing, and standardization of operations. The blocks in the early mines were about 75 feet high. At present blocks as high as 300 feet are developed, with considerable saving in preparatory costs.

The correct spacing of finger raises is essential to the successful operation of a block-caving mine. Close spacing will increase ore recovery and reduce dilution; wide spacing, on the other hand, will result in an economy of development and maintenance costs. Increased knowledge of the character of the ground, which is the main controlling factor in ore-drawing practice, has made possible the spacing of the finger raises so as to obtain the best results. It is also important, from the standpoint of efficiency, that the rate of drawing the ore is properly controlled. The slower the drawing rate, the more likely the ore is to be finely broken, but the weight on the grizzly level is greater and the maintenance expense is higher.

Standardization of mine practices, particularly of timbering, has likewise enhanced the efficiency of block-caving. Most of the framing, which formerly had been done underground, can now be done in surface shops; the

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[28] Even after the preparatory work is complete the output per worker may tend to fluctuate because the current development work may be done intermittently and the labor time so spent is reported at the time such work is done and not distributed over the period in which actual production occurred. Normally, development work proceeds at a fairly even pace with production, with the result that the ratio of preparatory work to output is kept more or less constant. However, with falling copper prices development work is likely to be abandoned; conversely, with rising prices it becomes necessary to prepare enough of the ore body not only to take care of the increased demand but also to make up for the pause during the period of low demand. Thus output per worker may rise sharply when development work is neglected and decline when the amount of preparatory work increases.

[29] Only 10 to 20 percent of the ore is drilled and blasted.

interchangeability of individual parts has reduced delays and labor requirements in actual mining. The use of mechanical loading devices for driving drifts has greatly reduced hand labor in primary development work. Together with parallel improvements in drilling and blasting and in transportation and hoisting practices, discussed later in this chapter, these advances have resulted in a material increase in the output per worker at mines employing the block-caving method.

Summary.- The changes that have contributed most to increasing the productivity of labor and to reducing mining costs may be summarized briefly as follows: Use of the force of gravity to break and transport the ore and, where it is necessary to fill excavated stopes, to transfer filling materials; adoption of the retreating system, which may be used with any of the stoping methods, obviating the necessity of maintenance work at the far end of the drift where the ore has been excavated; improvement in the arrangement of mine workings to permit the greatest utilization of mechanical equipment; and economy and, wherever possible, elimination of the employment of timber. These changes are the result of the gradual perfection of details rather than the invention of entirely new methods. Although these improvements in mining methods are of great importance, they might have had little effect in counteracting the depressing influence of increasing handicaps on the output per worker had it not been for the progress in mechanization.

PROGRESS IN MECHANIZATION

Concurrent with the evolution in mining methods and the advent of scientific planning and management has been a phenomenal growth in mechanization to which must be assigned the greater share of the credit for counterbalancing steadily increasing natural difficulties and raising output per worker. Advances in mechanization in major mining operations - drilling and blasting, mucking and loading, and transportation - have been accompanied by similar improvements in important auxiliary operations - mine supports, drainage, and ventilation. Each of these, while part of a whole, nevertheless is a field in itself; each presents problems calling for separate treatment. For this reason and also for convenience in presentation, these significant functions of underground mining are treated separately in the following discussion.

Drilling and Blasting

Advances in the technology of drilling and in the development of more powerful explosives are among the more significant factors that have affected output per man in underground copper mining. The introduction of mechanical drills has increased the speed of drilling operations substantially and raised the tonnage per drilling unit materially. By the development of various types of drills to meet specific ground and operating conditions, drilling efficiency was improved still further. The importance of these advances, however, has not always been fully appreciated; their effects are spread over such a long period that oftentimes they are obscured by the improvements in related fields. In many mines, however, it was possible to use mechanical loading devices only because the power drills could break enough ore so that loading machines could be used effectively.

Ore must be broken before it can be loaded and transported to the mill. To break copper ore it is ordinarily necessary to drill and blast it; owing to its comparative hardness the labor involved in breaking it generally constitutes a substantial part of the total underground labor force. When hand methods were used two-thirds of the underground men probably were engaged in shattering rock. The introduction of power drills gradually reduced the relative number of men engaged in drilling and blasting until at present these operations occupy 20 to 30 percent of the total underground labor. In 1919 the "miners and drillmen, including their helpers", constituted 28 percent of the total underground labor in Lake Superior copper mines;[30] in the mines of the Tennessee Copper Company, drilling and blasting labor comprised almost 27 percent of all persons employed underground in 1929.[31]

Drilling[32].- The exact date when power drills were introduced in the copper mines of this country is not known. They were in use in the Lake Superior district in 1876.[33] They were probably introduced at about this time; before then hand drilling was the only method available for breaking the rock. Hand drilling was done by the "hammer and drill" method, either by one man holding and turning the drill and striking it ("single jacking"), or by one man holding and turning and one or more men striking the drill ("double-jacking"). Although the operation seems simple, it required considerable skill. Not only was it necessary for the expert hand driller to be able to manipulate the hammer and drill efficiently under any condition but it was also most important that he be skilled

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[30] *Fourteenth Census of the United States: 1920*, Vol. XI, "Mines and Quarries: 1919" (U. S. Dept. Com., Bur. Census, 1922), p. 371.

[31] C. H. McNaughton, *Mining Methods of the Tennessee Copper Company, Ducktown, Tenn.* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6149, mimeo., June 1929), p. 16.

[32] Technological progress of rock drilling, including drilling at copper mines, is the subject of a report by C. E. Nighman and O. E. Kiessling, *Mineral Technology and Output per Man Studies: Rock Drilling* (WPA National Research Project in cooperation with U. S. Department of Interior, Bureau of Mines, Report No. E-11, Feb. 1940).

[33] T. Egleston, "Copper Mining on Lake Superior," *Transactions of the American Institute of Mining Engineers*, Vol. VI (1879), pp. 290-1.

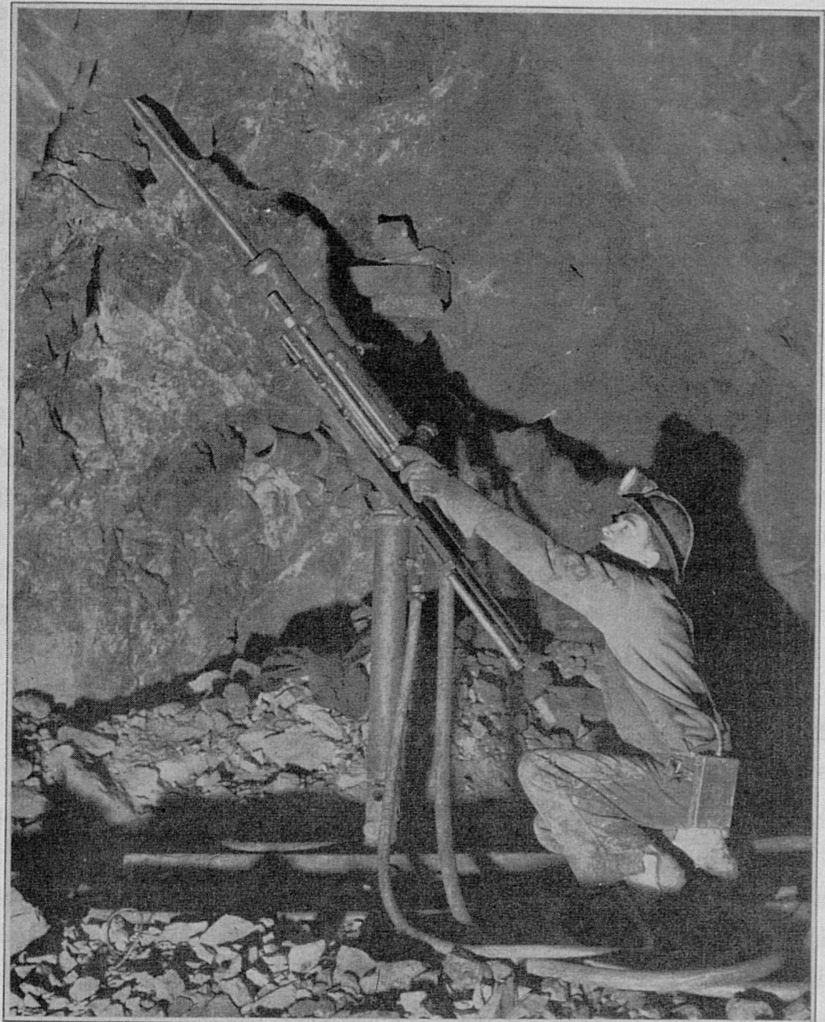

FIGURE 34.- AN IMPORTANT STEP IN METAL PRODUCTION IS THE BREAKING OF ORE BY DRILLING AND BLASTING

This picture shows a modern, one-man, column-mounted drifter drill boring an inclined hole at a western copper mine. Such drills are more commonly used, however, for drilling holes in a relatively horizontal direction. The "hard hat" worn by the miner affords protection against falling rock fragments, and the electric cap lamp provides illumination.

enough to point[34] his holes so that they would break the maximum volume of rock for every inch of hole drilled.

The first type of power drill introduced was known as the "piston" drill. It consisted essentially of a cylinder in which a piston was actuated backward and forward by compressed air. The piston was attached to a piston rod that was clamped rigidly to the drill steel. The piston delivered a succession of heavy blows, at the same time automatically rotating the steel. The early models were heavy and cumbersome. This

[34] To "point" a hole is to give direction to the hole.

was the principal type of power-drilling machine that was used for mining underground until about 1906. It had been considerably improved since introduction, having been made lighter, stronger, and faster.

The design of the early models was poor and two or more men were required to operate each drill; the initial cost of the drill, the air compressor, and the air-distribution systems was high; and the net result was that not infrequently the cost of a ton of ore obtained by power drills would be as much as, or even more than, that by hand drills. Another more elusive reason for the low effectiveness of the first machines, particularly during the earlier period of the development of the drill, might have been associated with the inability of the hand driller to adapt himself to the new machines, or perhaps to his reluctance to accept them. Moreover the piston machine, the only type of power drill then available, was ill-adapted to drilling "up"[35] holes. In the first place, it was difficult to set up the machine in a cramped space; in the second place, the effectiveness of the drill was dependent almost entirely on the delivery of a heavy blow. In "down" holes part of the strength of the blow was obtained from the gravity effect of the dropping mass of the drill, piston rod, and piston, whereas in "up" holes the effect of this mass was to lessen the force of the blow.

The stoper,[36] the next type of power drill to be generally adopted, was particularly well suited to drilling "up" holes. It probably was used in some copper mines as early as 1906-8. It is a hammer-type drill mounted upon a pneumatic-feed cylinder that is provided with a piston and a piston rod. The extension of the picton rod feeds the drill upward as the bits cut and the air pressure in the feed cylinder holds the drill bit against the rock face. The drill cannot be used satisfactorily for drilling holes at an angle of less than 20 degrees above the horizontal without special mountings or support.

At about the same time the "drifter" type, so called because it was designed primarily for use in drifts, came into use. It is especially effective in drilling "flat" holes. Like the stoper, it consists of a cylinder containing a floating piston that acts as a hammer and delivers a rapid succession of light blows against the end of the drill steel. The steel, which is nonreciprocating, is held against the bottom or end of the hole by the forward movement of the machine in a guide shell and

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[35] "Up" holes refer to holes pointed above a horizontal plane, whereas "down" holes refer to holes pointed below a horizontal plane. The terms are generally limited to steep holes 60 degrees or more above or below the horizontal plane. "Flat" holes are intermediate.

[36] The name of "stoper" was applied to this type of drill probably because it was developed primarily for use in stopes.

is automatically rotated. Hollow drill steel is used that allows a mixture of compressed air and water to be played on the extremity of the hole, thus keeping the hole clean and reducing dust. This hammer-type drill is superior to the piston type, for it is better designed and lighter and can drill faster in hard rock. As a result it has almost completely displaced the piston drill.

About 1909 still another type of hammer drill, the automatically rotating "plugger" or jackhammer was developed. It is best adapted for down holes and secondary drilling. It is mobile and light enough to be operated by one man. Its construction is based essentially on the same principle as that of the drifter, the principal differences being that it is lighter in weight and is equipped with handlebars for the driller to hold on to instead of a cradle that supports the larger type of drills. Like the drifter, it uses hollow steel through which compressed air is blown, with or without water. Although its rate of drilling per unit of drilling time is generally lower than that of the drifter, its extreme mobility makes it possible to operate this machine a greater proportion of the shift; it therefore usually produces a greater tonnage per driller-shift and often a greater tonnage per machine-shift. It has been generally adopted by the industry and its use has expanded rapidly. However, where the rock or ore is extremely hard the blow delivered by this type of drill is too light to be effective. The hand-held jackhammer is also least effective when horizontal or up holes are to be drilled, mainly because it is difficult for a driller to hold it against the end of a hole. It has therefore by no means entirely displaced either the drifter or the stoper types.

The hand drill and the piston type of power drill are both practically obsolete in underground copper mines at present. They have been supplanted by the drifter, stoper, and jackhammer types which can satisfactorily meet the needs of any drilling problem that may be encountered in underground development or mining.

Although it is obvious that substitution of power drills for hand drills has reduced the amount of labor required for drilling, it is difficult to determine the magnitude of these savings because of the lack of adequate statistical data; where data do exist for isolated examples it is often impossible to segregate the effects of power drilling from the effects of other factors.

In the Lake Superior country, virtually the only copper district for which statistics are available, the rock was hard and tough, and it is probable that the average footage drilled by hand per man-shift in this district was somewhat less than that for the copper industry as a whole. In 1878 three men could double-jack three holes per shift of 8 hours, each hole being 2-2½ feet deep.[37] If this were the average performance by hand drills in this district, it would represent about 1 ton of ore broken per driller-shift. Extensive tests showed that about 1913 the piston-type drill broke 12 tons per man-shift, the stoper broke 13 tons, and the jackhammer, 38 tons.[38]

The shift to power drilling affected not only the quantity of labor needed but also the quality. Hand drilling was an art that took years to learn. First it was necessary to learn to strike the drill in all directions and from various postures. An experienced driller was expected to manipulate the hammer and drill with the same facility in "up" and "angle" holes as in "down" holes. To turn the drill between blows properly so that the hole would remain round and the drill would not stick required a special skill. A successful hand driller must not only have strength and stamina but also a knowledge of how to conserve his energy. Because hand drilling was a comparatively slow process, it was essential that the holes be pointed so as to permit the maximum volume of rock or ore to be broken when the hole is blasted. The inexperienced driller was apt to let the easiest drilling conditions dictate the pointing of the hole, whereas the experienced one was governed solely by the possibility of breaking the greatest volume with the least possible amount of drilling. Although some strength was required to move the piston, drifter, and stoper types of drill from hole to hole and a certain skill was needed to set them up and point the holes properly, power drilling generally does not demand the skill or the stamina of the hand driller, for the heavy work is intermittent. Introduction of the automatically rotating jackhammer, easy to move and run, simplified drilling considerably. Almost any workman can be trained readily to operate such a drill.

Although there has been no revolutionary development in power drilling since the one-man jackhammer was introduced about 1909, constant efforts have been made to develop special models of existing types to meet particular conditions, to improve the designs of the drills, and to use more suitable materials in their construction. Most of these refinements have resulted in lengthening the effective life of the machines, improving their mobility by mounting them on trucks, decreasing their weight, increasing their speed, and reducing the consumption of air. These advances generally have been translated into reductions in the cost of

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[37] Egleston, *op. cit.*, p. 290.

[38] W. L. Saunders, "Rock-Drilling Economics," *Transactions of the American Institute of Mining Engineers*, Vol. XLVII (1914), p. 166.

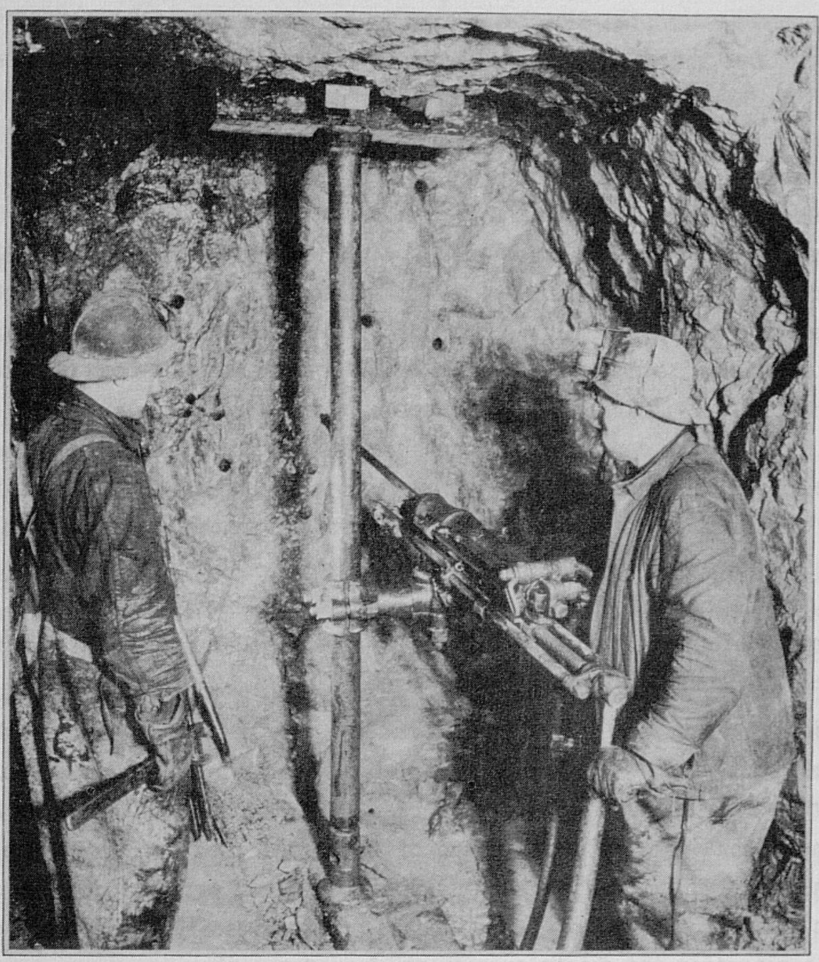

FIGURE 35.- DEVELOPMENT WORK UNDERGROUND

Before an ore body can be extracted in underground mining, it is usually developed by a series of horizontal and vertical or inclined openings of relatively small cross section. This development work consists largely of drilling, blasting, and removing material so as to provide the desired openings. The illustration shows a modern high-speed drifter drill at the face of a development heading. Part of the drilling round has been completed.

mining the ore and, indirectly, into savings of drill labor. The early drills frequently broke down and had to be sent to the shops for repairs. Every break-down meant a substantial loss of the drillers' time. Modern drills are more durable and remain in service longer.

Much attention has been paid to drills which have been improved by the use of special steels in their manufacture and by making the form of the bit or cutting edge suitable for various rock conditions. This has reduced breakage and wear and consequently has increased the tonnage of ore produced per pound of steel used.

Considerable effort has been devoted to developing detachable bits. The Anaconda Copper Mining Company has been one of the pioneers in this field and has adopted detachable bits in all its mines at Butte. At both the Calumet and Hecla and the Champion mines in the Lake Superior district and at some of the mines in other districts detachable bits have been used experimentally. Their use has increased appreciably during recent years but they have not as yet entirely supplanted the common drill steel. It is probably safe to say that the trend is toward their wider adoption by underground copper mines. The main advantages of detachable bits include an appreciable reduction in labor and cost involved in distributing sharpened drills and in collecting dull ones, for the miners are supplied with enough detachable bits for a day's work from the storeroom at the beginning of each shift; an increase in the speed of cutting and in the volume drilled by each bit and machine; and a decrease in steel loss, in the investment in drill steel, and in the unit cost of drilling. An incidental but beneficial effect of the use of detachable bits has been the lowering of the accident rate. When nondetachable bits are used, accidents often occur from handling the heavy drill steel. For example, electric shocks may result if the steel carried by the miners comes into contact with the trolley line.

Blasting.- Progress in drilling methods has been accompanied by advances in blasting techniques and in blasting agents. These changes not only have raised the output per worker by permitting the breaking of a higher tonnage of rock per pound of explosive used but also have decreased the dangers of mining. Such improvements as have occurred have been in the direction of selecting the proper blasting agent for a specific situation and obtaining more complete detonation.

In the early days blasting in the Lake Superior copper district was done largely with black powder. Nitroglycerin appeared in this district about 1874 but because of the difficulty of handling it safely, it did not entirely displace gunpowder. In the seventies dynamite became available and shortly afterward was generally adopted by the industry.

The original dynamite consisted of nitroglycerin and some absorbent, inert material such as diatomaceous earth or wood pulp. This type, known as "straight" dynamite, is still used to some extent. The so-called "ammonia" and "gelatin" dynamites were developed in the early eighties. In the former, part of the nitroglycerin is replaced by ammonium nitrate; in the latter, the explosive base is a waterproof jelly made by dissolving a special grade of guncotton in nitroglycerin. Several radically different kinds of blasting agents have been tried, such as liquid oxygen and water under pressure, but so far none has shown much promise for underground copper mining.

This historical development in the Lake Superior district was roughly paralleled in other copper districts. The introduction of dynamite coincided roughly in time with the introduction of the piston power drill, so that there might have been some relationship between the rates with which the piston drill and dynamite were adopted in each district. Since dynamite is denser and more powerful than gunpowder,[39] it permits the use of deeper and more widely spaced holes and heavier charges. It is therefore probable that some part of the increased tonnage efficiency of the piston drill over the hand drill is due to the use of a more powerful explosive in connection with the former.

There have been marked advances in the application of blasting agents coincident with the improvements of these agents. In the past 20 years the tendency has been to select the explosive best applicable to specific conditions. Explosives are now available that can give almost any desired action; a slow, heaving effect can be produced by black powder whereas any degree of shattering may be obtained by the use of the various grades of dynamite. Moreover, special consideration is given to charging the drill holes so as to obtain complete detonation. At present explosives are firmly tamped to exclude air pockets and carefully stemmed to confine the gases of combustion. Such practices result in an increase in the blasting efficiency of the explosive of 20 to 40 percent.

Detonators and fuses have been improved to such an extent that costly misfires are now rare. Ordinary fuse and caps have been improved to assure effective firing. The introduction of electrically ignited blasting caps has marked another important forward step in this field. The best possible location of the primer[40] to secure instantaneous detonation is determined by experiments at individual mines. Groups of fuses that formerly were ignited by hand are now fired by bunch blasting, which insures ignition of each fuse and promotes safety. In breaking hard rock delayed blasting,[41] which tends to increase the efficiency of blasting agents, can now be controlled by electric blasting caps.

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[39]The following tabulation showing the total energy of various blasting explosives, is compiled from R. Peele, *Mining Engineers' Handbook* (2d ed.; New York: John Wiley & Sons, Inc., 1927), p. 205:

Explosive	Foot-ton per pound of explosive
Blasting gelatin (U. S. Bur. Mines, Brunswig, Heise)	996-1,149
Nitroglycerin (Brunswig, Heise)	1,030-1,157
Dynamite, 75 percent (Brunswig)	819,904
Black powder (U. S. Bur. Mines, Brunswig)	402-553

See also C. Hall and S. P. Howell, *The Selection of Explosives Used in Engineering and Mining Operations* (U. S. Dept. Int., Bur. Mines, Bull. No. 48, 1913), p. 48.

[40]The primer is the dynamite cartridge in which the detonator is placed.

[41]Delayed blasting is usually accomplished by the use of varying lengths of fuse that fire the charges in a predetermined order.

As far as can now be visualized there is not likely to be any great change in explosives, fuses, or detonators that will significantly affect output per man in the near future.

Mention should be made of the increase in labor productivity attributable to better planning and supervision of drilling and blasting operations. In well-managed mines little is now left to the judgment of the miners. The rocks to be broken are examined carefully by a staff of experts and the work to be done is carefully planned. The miners are instructed how to drill and blast. The number, placing, spacing, size, and depth of holes are predetermined and the kind and quantity of powder to be used are prescribed.

Mucking and Loading

The function of mucking and loading is to move the broken rock from the working face to the main haulage system. It is one of the most time- and energy-consuming functions of all underground work in mining. Mechanization and improvement of mucking and loading operations have obviated a substantial part of the back-breaking, time-consuming work. In addition to decreasing the cost of handling material, this has attracted a different type of worker, reduced labor turn-over, and increased output per unit of labor time.

The earlier improvements were confined largely to methods of shoveling. These improvements consisted of instructing workmen how to shovel properly, of using shovels of proper size and proportion to facilitate shoveling, of designing lower cars so as to decrease the shoveling lift, of utilizing wood and steel plats to enable the men to sink their shovels into the broken ores more easily, and of pairing left-handed and right-handed workers in narrow drifts. The speed of loading was also increased by the introduction of a bonus system to induce the muckers to work faster.

Precise data showing productivity of labor before and after the improvements were introduced are scarce, but such fragmentary records as are available indicate that the output per man increased substantially after the methods of shoveling were improved. For instance, time studies conducted during 1917 at the Burro Mountain group of the Phelps Dodge Corporation at Tyrone, New Mexico, showed that after the improvements were made the tonnage in at least one of the large stopes could be increased from 8.5 to 22.9 tons per man-shift and the cost of shoveling could be reduced from 38 to 24 cents per ton.[42]

Footnote 42 appears on following page.

It was not until toward the end of the World War and in the period immediately following, when operators were confronted with a rising demand for copper, a scarcity of labor, and an increase in wages, that serious attention was given to mechanical mucking. The initial attempt at mechanization was directed toward the elimination of hand shoveling by substituting mechanical shovels. These shovels were of three general types.[43] The first included machines in which a dipper or scoop was crowded into a pile of broken ore and was then swung around to one side or to the rear and dumped. These were essentially miniature steam shovels which were used mostly in large open stopes and large tunnels and which required an operating width of 20 feet and a height of 15 to 20 feet. The second type consisted of machines in which the dipper or scoop was forced into a pile of ore and then raised and overturned backward, discharging its contents upon a conveyor or directly into a car. These shovels are used today in drifts and tunnels 8 to 14 feet in width and 7 or more feet in height. The third type operated on the principles of the other two; it was suitable for drifts 6 by 7 feet to 9 by 10 feet in cross section.

A considerable number of these mechanical shovels achieved temporary success, but only a few of the better-designed ones have survived. These machines lessened the amount of labor in mucking, but the delays necessitated by the expenditure of time to move the loaders, set them up, and repair them when break-downs occurred, together with the machine's inability to cope with large boulders and its limited flexibility, more than offset the labor saving incident to their adoption.[44] Weakness in design was the fundamental cause of the failure of these loaders. Most of them were designed to operate under standard conditions, without consideration of the fact that in mining unusual conditions have to be met frequently.

After considerable experimentation some of the more suitable loaders were redesigned. The steam-shovel type, with dipper and boom, was eliminated because of space limitations in underground workings. The important resulting improvements included lighter weight without sacrifice of strength, more flexible and mobile units, elimination of intricate

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[42]G. T. Harley, "A Study of Shoveling as Applied to Mining," *Transactions of the American Institute of Mining and Metallurgical Engineers*, Vol. LXI (1920), p. 186. It should be pointed out that low-wage Mexican labor was employed and that the spacing of the raises was rather poor. These factors probably explain the low tonnage obtained per shoveler at the beginning of the experiment.

[43]For a detailed description of the more important of these mechanical shovels see L. Eaton, "Underground Loading Machines," *Engineering and Mining Journal*, Vol. 134, No. 5 (May 1933), pp. 195-6.

[44]For a discussion of the experiments conducted with some of these mechanical loaders see W. V. DeCamp, "Underground Mechanical Loading at United Verde," *The Mining Congress Journal*, Vol. 12, No. 3 (Mar. 1926), pp. 197-9 and C. E. van Barneveld, *Mechanical Underground Loading in Metal Mines* (Rolla, Mo.: University of Missouri, School of Mines and Metallurgy, May 1924), pp. 535-41.

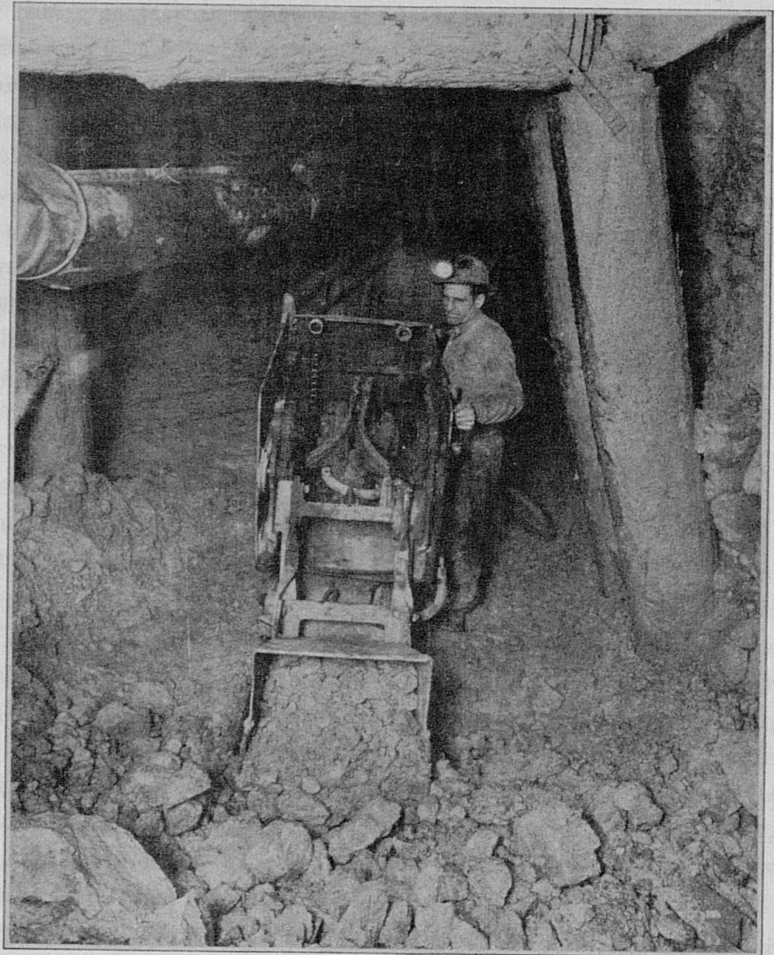

FIGURE 36.- THE USE OF MECHANICAL LOADING MACHINES IN UNDERGROUND MINES HAS INCREASED

The small, compact mechanical loader shown above operates in drifts that are small in cross section; it greatly reduces the amount of hand loading. The dipper or scoop is forced into the pile of ore, raised, and overturned backward, and its contents are discharged directly into the car behind.

mechanisms that are not absolutely essential, substitution of steel castings for riveted and bolted members, replacement of cast iron by steel, improved bearings and axles to reduce friction, positive lubricating points, fewer gears and chains, and sturdier conveyor belts.

At the same time drilling and blasting methods were altered to meet the requirements of mechanical shovels. Rocks were fractured to smaller pieces to facilitate handling by these loaders. Because they were not equipped to clean up "fly dirt" satisfactorily, the blasting practice was altered to concentrate the muck pile. Moreover, efficiency engineers were employed to coordinate auxiliary operations, and accessory equipment was used with the shovels to assure efficient operation.

Even with these improvements the loaders have seen little service in the stopes of the underground copper mines. In the first place some of the more important underground mines, which employ mining methods utilizing gravity, do not require shovels. In the second place many mines, especially those which use artificial methods of support, do not permit these loading machines to operate efficiently because of the limitation of space and congestion. Consequently the machines have been confined largely to development work where they increased output per man and the speed of drifting.[45]

Modern mechanical mucking machines must be small in size and rugged in construction, able to operate in drifts as small as about 5 by 7 feet, and capable of cleaning up a working face in a relatively short time so as to allow a complete cycle of drilling, blasting, and mucking operations to be completed in an 8-hour shift. Moreover, the size of the cage is limited and loaders had to be made small to facilitate their transfer from level to level. Such small, flexible loaders were introduced by various manufacturers beginning in 1931.[46] They are compact and efficient, able to operate in drifts as small as 4 by $6\frac{1}{2}$ feet, can be transported in any section of the mine, have a low operating cost, and require only one man for their operation. It has been found that with the introduction of these machines the output per man in development work has increased considerably, the cost per foot of advance has declined, and the speed of driving drifts has been accelerated. These machines are most effective in development work, although they have been found equally satisfactory for stoping operations where mining conditions permit their use. They have proved to be the salvation of some of the deep, hot mines where men find physical exertion difficult because of the great heat and humidity. It would have been extremely difficult to conduct development work in some of these mines but for the introduction of these loaders. That these machines have been efficacious in increasing the output per man is

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[45] See F. W. Snow, *Mining Methods and Costs at the Magma Mine, Superior, Ariz.* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6168, mimeo., Sept. 1929) and M. Mosier and G. Sherman, *Mining Practice at Morenci Branch, Phelps Dodge Corporation, Morenci, Ariz.* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6107, mimeo., Mar. 1929).

[46] For a discussion of these loading machines see J. S. Finlay, "Mucking at North Lily Mining Company," *The Mining Congress Journal*, Vol. 21, No. 4 (Apr. 1935), pp. 33-4; J. S. Finlay, "Eimco-Finlay Loader and Its Applications," *Mining and Metallurgy*, Vol. 16, No. 347 (Nov. 1935), pp. 457-8; B. P. Spann, "Mechanical Muckers Answer Demand for Speedier Loading," *The Mining Congress Journal*, Vol. 24, No. 7 (July 1938), pp. 13-4; and M. Mosier and J. H. Steinmesch, *Mechanical Shoveling in Underground Metal Mines* (U. S. Dept. Int., Bur. Mines, Bull. No. 423, 1939, in press).

attested to by the fact that they are rapidly replacing hand shoveling wherever it is advantageous to do so.[47]

Meanwhile scrapers[48] were adopted widely for mucking in underground mines.[49] By 1921 the operators had come to the realization that even with the most efficient loaders then available hand shoveling could not be eliminated. Loaders then in existence required much headroom and could be used effectively only on a level floor. This meant that they could be employed mainly for development work. Moreover, they were expensive to construct and maintain. Scrapers, on the other hand, could work in mines where the workings are restricted, where the deposits lie at an angle that would preclude the use of mechanical shovels, or where the stopes are unsafe for hand shovelers to work. They were not only efficient in the stopes but also in the drifts to which the shovels were largely if not wholly confined; they were not only effective as loading devices but also as transporting mechanisms with a wide action radius. Finally, they had the advantage of comparatively low initial and maintenance costs.

Experiments in the design, construction, and operation of scrapers and auxiliary equipment have been intensified since the close of the last World War. They have been made larger and heavier; alloy steel has been substituted for ordinary steel in the construction of wearing parts; hoisting units have been made more portable and efficient, with double and triple drums replacing single drums; and electricity has been supplanting compressed air as a source of power. Moreover, the scraper and its auxiliary equipment have been constructed to suit particular mining conditions or mining methods employed and, in some instances, mining methods have been modified to utilize scrapers more advantageously.

As a result of these improvements scrapers have been displacing hand shovels and mechanical loaders in a number of mines, particularly those

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[47] At the Butte mines of the Anaconda Mining Company 75 mechanical shovels are now in use. They not only have been replacing hand shoveling but also have almost entirely displaced scraper slides for loading mine cars. (See H. M. Courtney, "Mechanical Loading in the Butte Mines," *Engineering and Mining Journal*, Vol. 139, No. 12 [Dec. 1938], pp. 31-7, 56.) These shovels are also utilized in all the drifts at the Magma mine in Arizona, where, because of high temperatures and humidities, it is difficult to muck manually. It is estimated that the saving in labor cost resulting from the use of one of these loaders is sufficient to pay for the machine in less than a year. Apart from the cost, there is a saving in physical effort for the workers themselves. (See D. Gardner, "Mucking Machines at the Magma Mine," *The Mining Congress Journal*, Vol. 25, No. 3 [Mar. 1939], pp. 13-5.)

[48] A scraper is a mucking device shaped like a box or hoe and is somewhat analogous to the familiar horse-drawn scraper frequently used in surface grading. The box type is used for loading fine and medium-coarse mucks; the hoe type, for coarser materials. The scraper is not a self-contained loader, for it must be dragged by a main and a tail rope attached to a hoist which is powered by compressed air or electricity. The load may be scraped along the floor of a working and dumped into an ore chute, or it may be dragged along the floor and up an inclined slide to a platform where it is discharged into a car beneath.

[49] For a detailed discussion of the use of scrapers in underground metal mines see C. F. Jackson, *Underground Scraping Practice in Metal Mines* (U. S. Dept. Int., Bur. Mines, Manuscript Report No. 1 [reprinted by Sullivan Machinery Co., 1933]).

that do not require much hand sorting.[50] Instead of being shoveled by hand or with loaders, the broken material is dragged from the working face to the chutes which convey it to the haulage level where it is loaded into mine cars, or it is dragged up an inclined platform and dumped directly into cars below. The use of scrapers is especially appropriate in stopes where, because of squeezes, rock bursts, or unsafe roofs, it would be dangerous for hand muckers and loaders to enter. Moreover, scrapers require no tracks upon which to operate, and can work at any angle to the horizontal. Thus the adoption of scrapers has not only dispensed with shoveling and loading wherever conditions permit but has also eliminated tramming or shortened tramming distances considerably and increased the safety of mining. Where it is necessary to fill excavated stopes, scrapers may be employed to transport and spread the filling material.

The effect of the utilization of scrapers in copper mining, like that of the introduction of mechanical loaders, has been to increase appreciably the output per man in mucking. As mining conditions and methods differ from mine to mine, however, the gains in output per unit of labor attributable to the use of scrapers may, of course, be expected to vary. Thus at the Conglomerate mine of Calumet and Hecla Consolidated Copper Company in Calumet, Michigan, where the deposits are of the tabular type dipping at less than the angle of repose of the broken ore, the amount of material mucked per man-day ranged from 25 to 78 tons when scrapers were used and from 14 to 15 tons when mucking was done by hand shoveling. Under practically similar conditions the amount of material loaded with scrapers at the Ahmeek mine was 50 to 75 tons per man-day.[51] Not all the increase, of course, is attributable to the use of scrapers, for in some instances mining methods have been altered to permit utilization of these machines. In the Champion mine, for instance, the method of mining was modified drastically before scrapers could be introduced. The combined result of the change in mining method and the introduction of scrapers was an increase of 43 percent in the output per man in mucking.[52] Where conditions and methods favored the utilization of scrapers, as in the case of mines employing top-slicing and sublevel-caving methods, the increase in the output per man was considerably larger. Here application of scrapers to mucking has not only increased output per man by 250 to

[50] Hand sorting of the ore (that is, the separation of the valuable mineral-bearing pieces from the barren waste rock) is practiced to reduce the haulage, hoisting, and milling charges.

[51] Van Barneveld, *op. cit.*, pp. 332-4, 323, 340.

[52] Jackson, *Underground Scraping Practice in Metal Mines*, pp. 76-8. This increase occurred between 1921 and 1931, years in which labor conditions were similar. Mining conditions were, however, much more favorable in 1921 than in 1931. In the earlier year the stope backs were solid and secure; in 1931 they were squeezing and crushing, and rock bursts were frequent. Hand sorting was practiced in both years. But for the change in mining conditions and for hand sorting, the increase would have been much greater.

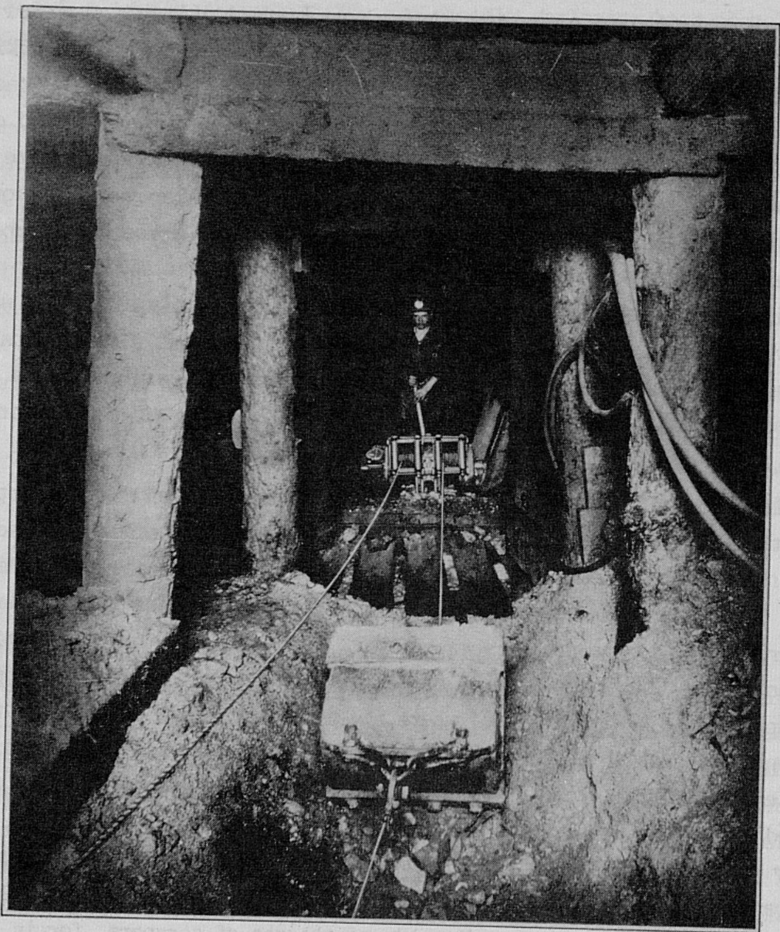

FIGURE 37.– A SCRAPER LOADER IN A WESTERN COPPER MINE

Scraper loaders are used to advantage where the workings are restricted or where the deposits lie at an angle and preclude the use of other loading arrangements such as mechanical shovels or gravity methods. They are also used to remove ore from stopes where continuous work by miners would be too hazardous. The scraper shown above, operated by a double-drum hoist, is moving ore to a grizzly-covered ore chute (the parallel bars shown between the scraper and the miner) through which the ore falls into a loading bin.

300 percent[53] but has also eliminated track work, reduced the number of ore chutes, and permitted ore to be removed from areas where, because of crushed timber or the presence of other unsafe conditions, it would be impossible for hand shovelers or mechanical shovels to operate.

Scrapers may be utilized advantageously not only for transporting ore and filling material in the stopes but also for removing the muck from inclined shaft-sinking operations and drifts. In 1931 the Copper Range Company of Michigan, by applying scrapers to mucking in shaft sinking,

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[53] W. L. G. Muir, "Applications of Scraper Loading to Mining," *Mechanical Handling and Works Equipment*, Vol. XXI, No. 7 (July 1934), p. 195.

attained 19 to 21 tons per man-shift with the bucket dump 330 feet above the bottom compared with 9 to 10 tons obtained by hand mucking.[54] The case of the Champion mine may be cited as an example illustrating the effect of the use of scrapers for driving drifts.[55] Before 1929, when hand mucking was employed, the drift advance at this mine was 0.7 foot per man-shift. Since then, with the adoption of the scrapers, the drift advance increased to 1.2 feet per man-shift.

It may be stated in summary that the introduction of mechanical loaders and scrapers and the constant improvements of these devices have raised the productivity of labor; because these advances have occurred simultaneously with other technologic changes, however, it is not possible to state numerically what portion of the over-all gain in the output per worker is attributable to the progress in mucking and loading practices. They have also served to lessen the drudgery and increase the safety of mining and thus have made mining a more attractive occupation.

Transportation

Underground transportation - that is, the hauling and hoisting of ore, waste, supplies, and workmen between the surface and the underground workings - is another important mining function that has undergone improvements which have profoundly influenced the productivity of labor. These advances have proceeded along several general lines, the more important of which being an increase in the substitution of mechanical power for manual labor, an expansion in the capacity of equipment without a corresponding increase in labor, development of new labor-saving equipment and methods, better adaptation of new and improved equipment to fit the requirements of the mines, and improved coordination of transportation facilities to eliminate delays and prevent the loss of time. In view of the fact that the equipment and problems involved in hauling and hoisting differ, improvements in the two forms of transportation and their effect on the output per worker are discussed separately.

Haulage.- The function of underground haulage is to move ore from stope to shaft and to bring supplies and, where necessary, filling material to the stope. The ideal haulage system is one that performs this function at the lowest possible cost and with the least possible delay. Copper mining has made great progress in increasing the efficiency of its underground haulage system by improving equipment and coordinating transportation facilities.

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54 A. H. Hubbell, "Economies and Efficiencies in Mine and Plant," *Engineering and Mining Journal*, Vol. 133, No. 1 (Jan. 1932), pp. 7-8.
55 Muir, *loc. cit.*

In the early days, when the mines were shallow and shafts were numerous, most of the ore was trammed by hand in small $\frac{3}{4}$-, 1-, and 2-ton cars. These cars, which were of end-dump design, were heavy and crude. They were loaded by hand, pushed through the drifts at low speed by two or three men over 18-inch track, and dumped by hand. The average day's work per trammer man at 17 domestic copper mines ranged from 0.55 to 2.33 ore ton-miles.[56] Mules and horses were employed underground as the distances and tonnages increased. Animal tramming was much more efficient than hand tramming. Steeper grades could be overcome more readily and larger tonnages transported per trip. The amount of ore hauled per trammer-shift ranged from 16 to 30 ton-miles.[57]

Hand tramming and animal hauling were adequate when production was small and the distances involved were short. As the mines became larger, however, the area from which the ore was to be hauled underground increased considerably. The underground haulage distances could, of course, be shortened by sinking more shafts, but shaft-sinking and its attendant hoisting equipment were so costly, particularly when the depth was increasing, that it was more economical to haul from longer distances than to hoist from more shafts. There was increased pressure to replace hand and animal tramming by a more efficient system of hauling. Introduction of the locomotive and of the scraper was the result. The scraper, discussed in a previous section of this chapter, will be considered in another connection later in this section.

Electric-trolley locomotives were first used in underground copper mines in the nineties.[58] Although they were a great improvement over animal tramming,[59] they were nevertheless much less efficient than their modern successors. Their motors were small in relation to their heavy cast-iron frames and they broke down frequently. This resulted in costly delays and required high expenditures for maintenance. The usual power specification was 10 horsepower per ton of locomotive weight. Substitution of steel for cast iron, which permits a decrease in the weight of the frame and an increase in that of the motor, has gradually raised the ratio until at present it is not uncommon to find locomotives having

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[56] Computed averages from M. J. Elsing, "Cost of Mine Transportation," *Engineering and Mining Journal*, Vol. 134, No. 3 (Mar. 1933), p. 103.

[57] Peele, *op. cit.* (1st ed.; 1918), p. 865.

[58] See the following articles in *The Mining Congress Journal*, Vol. 21, No. 7 (July 1935): G. H. Shapter, "Modern Main Line Haulage Locomotive for Coal and Metal Mines," pp. 50-1 and J. S. Beltz, "The Economy Slant on Mine Locomotives," pp. 22-3.

[59] Writing in 1922 Woodward pointed out that each trolley locomotive replaced five animals at the Butte mines. Obviously the number of drivers and helpers also was reduced in the same proportion. See C. D. Woodward, "Electric Haulage System in Butte Mines," *Transactions of the American Institute of Mining and Metallurgical Engineers*, Vol. LXVIII (1923), p. 101.

15 horsepower per ton of weight. Not only have the motors been enlarged but they also have been made sturdier, speedier, and more powerful per unit of weight. The early, open types of motors, in which the mechanisms were exposed to the hazards of dripping water and dust, have been supplanted by totally enclosed types. Instead of the former oil- and waste-packed axle bearings they have dust- and dirt-proof axle bearings; this alone has reduced the replacement cycle from a few months to many years. The cast-iron, sectionalized, and other types of gears formerly used in motors have been replaced by those constructed of the best-quality forged and heat-treated steel. The locomotives, in addition to their motors, have been improved materially by superior control apparatus, better wiring, the substitution of antifriction journal bearings for those formerly employed, and the installation of more efficient and more powerful brakes. The net result has been a considerable reduction in the cost of maintenance, a practically complete elimination of delays due to break-downs, and a substantial increase in hauling capacity.

Electric-trolley locomotives have been confined largely to the main haulageways where they may be used to the best advantage for long-haul operations. For this purpose they have been widely employed in both vein and porphyry mines. In vein mines the smaller locomotives, weighing 6 tons or less, are generally used. They can haul twenty to thirty 4- to 5-ton cars over 30-inch tracks. For use in caving mines the larger types, weighing 8 to 15 tons and able to pull 20 or more 6- to 10-ton cars over 36-inch tracks, are commonly employed. Where conditions do not permit use of the heavier types several smaller locomotives are operated in tandem.

Although electric-trolley locomotives are unequaled for long hauling, they are not without drawbacks for other purposes. In drifts, where the clearance is small and the ground wet and where the shape and alignment are constantly changing, trolley wires are a source of danger: they may injure workmen or cause mine fires. In new workings, too, where the track is laid in advance of the trolley line, locomotives cannot enter. To enable the locomotives to operate beyond the end of the trolley line, several devices have been utilized.[60] The crab reel, consisting of a rope-reel hoisting mechanism from which a cable is paid out, is used to pull cars from the drifts to the main haulageway. The electric-cable reel, which connects the motor to the source of power, permits the locomotives to enter working places where tracks have been laid but where trolley lines have not been installed.

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[60] Van Barneveld, *op. cit.*, pp. 47-9.

Several types of self-contained locomotives, which carry their own source of power, have been experimented with and developed. One of the earliest of these is the compressed-air locomotive.[61] In mines where conditions prohibited the installation of trolley wires compressed-air locomotives were used to good advantage. In other mines, however, they are poor substitutes for the electric type. Their initial cost, covering the locomotives and the auxiliary compressors, pipe lines, and storage tanks, is extremely high. They are expensive to operate, both in the matter of power and maintenance; they are bulky in relation to their weight; their center of gravity is high above the rail, causing frequent derailment; their cruising distance for one charge of compressed air is short, necessitating the location of charging stations at frequent intervals; and they are often stalled between stations, because of either lack of power or derailment, thus blocking the entire haulage system. The trolley locomotives, in comparison with the compressed-air locomotives, are much more economical, both as to first cost[62] and operation.[63] They are rugged, compact, small for their weight, and have a low center of gravity so that derailments are infrequent.

The gasoline locomotive, another self-contained type, was tested in some mines. Although it is economical to operate, it is dangerous to use underground; it discharges gases containing a large proportion of carbon monoxide and sometimes emits sparks from its exhaust that may start a mine fire.

Not until the copper-mining industry adopted the storage-battery locomotive did it begin to solve one of the difficult problems of underground haulage. Like the compressed-air engine, the storage-battery unit is self-contained, independent of trolley wires, and capable of traveling on its own power wherever rails are laid. The earlier models were large and cumbersome, were equipped with relatively small batteries, and had restricted cruising distances and hauling capacity. By 1916, however, there was developed a reliable locomotive having a battery whose capacity was sufficient for fairly heavy duty and long hauls.[64] Since then many more improvements in the design and construction of the locomotive have

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[61] W. Jurden, "Development and Application of Energy in the Mining and Metallurgical Treatment of Copper Ore," *Transactions of the World Power Conference. Sectional Meeting, Scandinavia, 1933* (Stockholm, Sweden: Kungl. Boktryckeriet. P. A. Norstedt & Soner, 1934), p. 153.

[62] It is estimated by Jurden that in a mine hauling 20,000 tons of ore per day the compressed-air system would cost approximately $350,000 compared with $250,000 for the electric-trolley locomotive installation (*ibid.*, p. 154).

[63] The power costs of compressed-air locomotives usually average about four times those for electric locomotives. See C. F. Jackson, *Some Notes on Underground Transportation* (rev. ed.; U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6326, mimeo., June 1931), p. 15.

[64] Van Barneveld, *op. cit.*, p. 40.

been added. The batteries have also been improved, having a longer life and a greater ampere-hour capacity per unit of weight than their predecessors. These advances have increased the efficiency of the locomotive to such an extent that it is now widely used in copper mining. Its initial and operating costs are relatively low. It is simple and compact in construction and flexible in operation; it can travel through narrow, crooked openings and can enter newly made drifts and other workings where it is difficult or dangerous to maintain a trolley wire.

The battery locomotive is well adapted to vein mines where loads are light and haulageways narrow. It is used to good advantage not only in the main haulageways but also in small drifts. The introduction of the $1\frac{1}{2}$-ton gathering locomotives into these mines has been particularly significant, for these small units have been displacing a large part of the manual labor and animal power, which were mainly relied upon until about 1924 to tram ore and material in drifts and other workings that could not be serviced by the electric-trolley locomotives.[65] The battery locomotives have been rapidly replacing the trolley locomotives in some of the important vein mines.[66] In caving mines, however, the usefulness of the battery locomotive is rather limited. Because the duty here is heavy and continuous and the haul long the locomotive must have an oversize battery with a capacity to store sufficient energy for a day's work. Since the drifts must be kept as small as possible, a locomotive large enough to contain such a battery would be too bulky for efficient operation underground. The battery locomotive is therefore largely used to deliver timber, drill steel, and other supplies to the working face and to haul ore from workings where it is not possible to install a trolley wire to the main-line haulage level, where the ore is transported by electric-trolley locomotive to a central point near the hoisting shaft and dumped into an underground ore bin.

Although the electric-trolley and storage-battery locomotives have been coming into general use since their introduction, tramming by hand and animal power is still resorted to in many mines, especially the smaller ones, in prospect work and in the intermediate levels. The main factors that determine whether one kind of motive power or another is to be employed are the length of the haul and the volume of traffic. Where the distance is short and the tonnage is small, tramming by manual labor or by animal power may be most economical. Where the haul is long, the traffic

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[65] Three small storage locomotives installed at the United Verde Extension mine in Jerome, Arizona, paid for themselves in 2 years and eliminated six to eight men per locomotive. (See Hubbell, *loc. cit.*)

[66] In the Butte mines, for example, there has been a complete shift from trolley to battery locomotives.

heavy, and the service continuous mechanical methods of haulage usually are employed. With respect to the choice of locomotives, the general practice is to use small battery units to gather ore from the working face and transport it to the main haulageway, where it is hauled to the shaft by 4- to 6-ton storage-battery locomotives (if the daily tonnage is relatively small and the tramming distance not over 5,000 feet) or by electric-trolley locomotives (if the volume of traffic is heavier and the haulage distance longer).[67]

Equally significant advances have been made in the construction of mine cars. One of these is enlargement of capacity by decreasing deadweight per unit of pay load and by increasing the size of cars. Reduction of the deadweight of the cars in relation to their pay load has been achieved by altering designs, by substituting high-tensile-strength steel for the ordinary steel and wood as construction materials,[68] and by welding instead of riveting the steel plates together.[69] The increase in the size of cars has been made possible largely by the adoption and improvement of mechanical haulage. Before the introduction of the locomotive the size of the car was conditioned by the load that could be trammed by hand or hauled by animals. Since then, with the increase in the track gage and the radius of curves, the size of cars has been enlarged to the extent permitted by the drifts. The adoption of scrapers and mechanical shovels also has contributed to the growth in the size of cars. When loading was done by hand the height of the cars was limited by the most effective lift of the shovelers; with the advent of mechanical devices it could be extended to the height of the chutes.

The substitution of antifriction for plain bearings,[70] the use of improved wheels with manganese-steel tires instead of cast-iron or steel wheels, and the development of new methods of attaching the wheels to axles have reduced by 40 to 50 percent the tractive effort required to move a loaded car. These advances have permitted the same locomotive to pull a heavier load. In hand tramming they have enabled the miner to tram cars with a larger capacity and at the same time lightened his burdensome task considerably.

Not only has the capacity of the cars been increased and tractive effort reduced but the design of the cars has also been radically altered

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[67] L. Eaton, "Changing Standards in Underground Transport," *The Mining Congress Journal*, Vol. 23, No. 9 (Sept. 1937), pp. 50-1.

[68] Steel with high tensile strength has about the same weight as ordinary steel, but because less of it is required for the same strength, a saving in the weight of the car is attained.

[69] C. G. Sensenich, "Electric Welding and Alloys in Mine Cars," *The Mining Congress Journal*, Vol. 21, No. 7 (July 1935), p. 47.

[70] E. C. Reither, "Roller Bearings Cut Production Costs," *The Mining Congress Journal*, Vol. 21, No. 7 (July 1935), p. 46.

to facilitate dumping operations and thus raise further the efficiency of the underground haulage system. In the early days of hand and animal tramming the cars were usually of the end-dump type.[71] They required much labor and time to unload, for they had to be uncoupled before they could be emptied. To obviate this shortcoming bottom-dump cars were employed. They, too, were by no means perfect, for they were not suited for hauling coarse or wet ore and were costly to maintain. With the advent of the locomotive it became necessary to construct cars that would facilitate dumping. To this end several types of cars, each with a different method of discharging ore, have been designed and developed.

The earliest of these is the gable-bottom type which dumps on both sides through swinging doors hinged at the top. Its chief merit lies in the fact that the load may be dumped while the car is in motion. But it has many disadvantages. The deadweight is heavy in relation to the pay load, the cost of maintenance is high, and the amount of leakage is large and requires the employment of workmen to keep the tracks clean.

The Granby or the "tipping-side dump" type, which dumps on one side only, is an improvement over the gable-bottom. Like the latter it can discharge its load while it is in motion. As it has only one door, which is closely fitted to the body, it reduces leakage. It is capable of handling coarse or fine, wet or dry ore. It has a low center of gravity and is intended for use in mines where a car of large capacity and low height is desired.

The rocker-dump, another side-dumping car, is especially useful for handling wet ore. It has a V-shaped or U-shaped body which rolls sideways on "rockers" or arch-shaped tracks and dumps its contents to one side. However, its construction is such that its center of gravity is high, and consequently its capacity is relatively small. This limitation is compensated to some extent by the fact that the car has no doors, thus eliminating the problem of leakage along the haulageway and the necessity of cleaning the track. It also has the advantage of low maintenance cost.

The rotary-dump or box-type car is perhaps the most efficient, being without doors or levers of any kind and rigid in construction. The deadweight per unit of pay load is small and the cost of maintenance is relatively low. Dumping is done by means of a rotary tipple or cylinder

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[71]For a discussion of the various types of cars mentioned here see the following series of articles by A. H. Hubbell, appearing in the *Engineering and Mining Journal*: "Which Mine Car, and Why?" Vol. 128, No. 10 (Sept. 7, 1929), pp. 400-2; "Mine Cars: A Study in Design, Performance, First Cost, and Maintenance Expense," Vol. 130, No. 5 (Sept. 8, 1930), pp. 225-8; "Mine Cars: Design, Performance, and Maintenance Expense Contrasted," Vol. 130, No. 7 (Oct. 9, 1930), pp. 322-5; "Mine Cars: The Rigid-Body Type Requiring a Rotary Dumper," Vol. 130, No. 11 (Dec. 8, 1930), pp. 560-2; and "Mine Cars: Bottom-Dump and End-Dump Types," Vol. 131, No. 4 (Feb. 23, 1931), pp. 162-5.

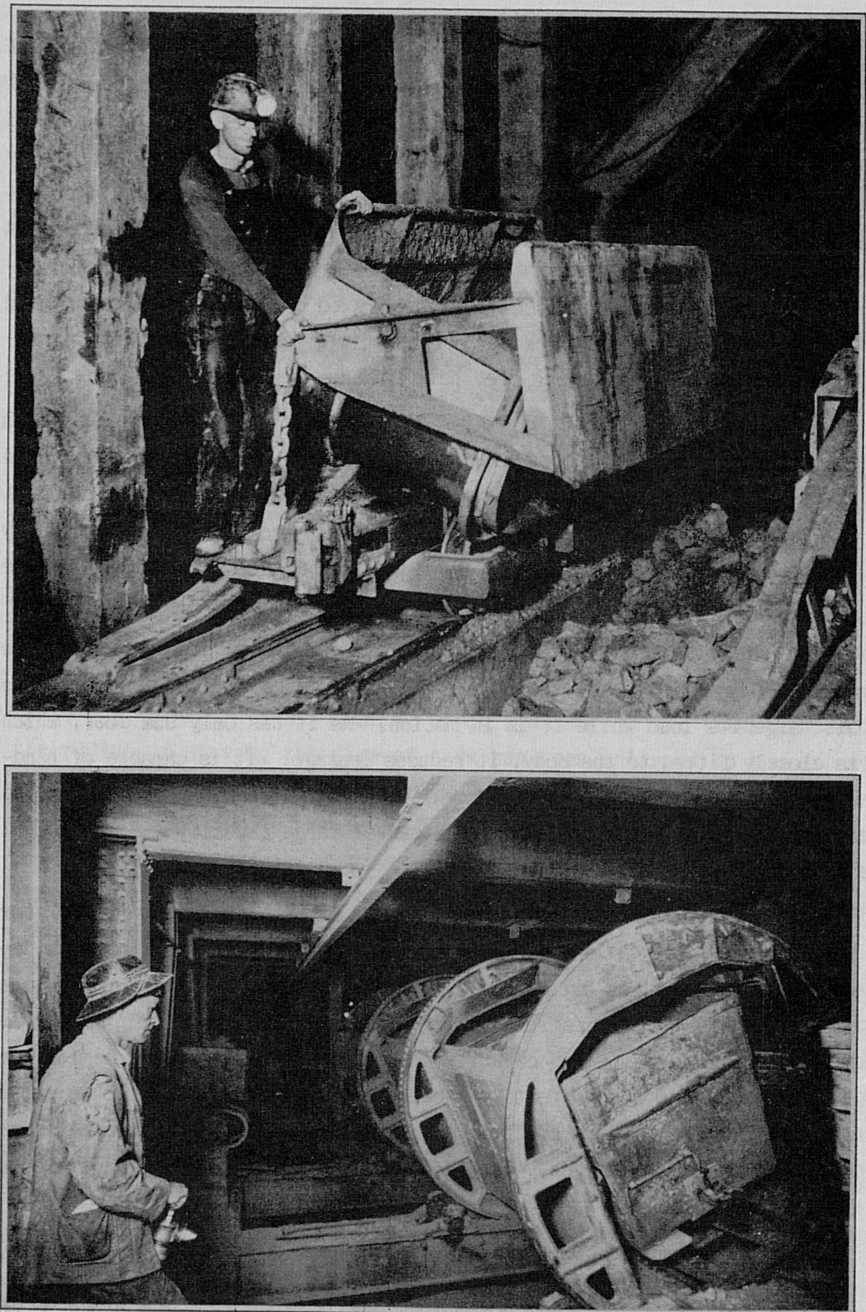

FIGURE 38.- EFFICIENT UNDERGROUND TRANSPORTATION REQUIRES THAT CARS BE UNLOADED QUICKLY

The side-dump car in the upper view is automatically dumped while a train of cars is in motion by elevation of one side of the car with a dump rail. The lower view shows three box-type cars being dumped by a rotary tipple. In some instances swivel couplings are installed on the cars so that uncoupling of the cars is unnecessary.

that turns over several cars or even an entire trainload of cars at one time, discharging the ore into a skip pocket below. This type of car can handle coarse or fine and wet or dry ore; it is especially suitable for operations in porphyry mines where the tonnage is large and the ore is handled on one level. Its main drawback is that it requires extra equipment in the form of a rotary tipple. This disadvantage, however, is offset to some extent by the fact that a number of cars may be dumped at one time.

Each type of car has been constantly improved and modified to suit conditions in various mines. One type may be efficient under one set of conditions and not under another. Some important factors determining whether one or another type is to be adopted are the kind of ore - wet or dry, coarse or fine; the frequency of the service - continuous or intermittent; and the volume of traffic - large, as in porphyry mines, or small, as in some of the vein mines. Perhaps the selection and adaptation of the appropriate type of cars to fit conditions are just as important in promoting efficiency in underground haulage as are improvements in the cars themselves.

One other important advance in the construction of cars should be mentioned - the change in the method of coupling. Formerly the cars were joined by the hook-and-chain or link-and-pin methods that required much time for connecting and disconnecting cars and caused many accidents. The present practice is to install partly or fully automatic couplers on all cars. On rotary-dump cars pivot couplings usually are employed to allow the cars to be dumped without being disconnected from the locomotive or sections not in the tipple. Another advance is the use of drawbars that reduce shock and damage to the car frame.

The advent of new and larger types of equipment has necessitated many changes in the haulageway. The drifts have been enlarged and straightened and the curves made less abrupt. The tracks have been improved: gages have been widened, heavier rails are used, switches and turn-outs have been made more foolproof, roadbeds have been better ballasted, untreated small ties that deteriorated rapidly have been replaced by larger chemically treated ones, and the lay-outs of the track systems have been better planned. Block-signal systems and automatic-switching devices have been introduced in some mines to facilitate the loading and movement of trains. In brief, the trend in underground haulage has been to adopt the advanced practices of surface-railroad engineering.

But for the parallel advances made in the loading as well as in the hauling of cars, the growing efficiency of the underground transportation system would have been greatly impeded. The amount of time and labor required to fill a train of cars frequently is several times as much as that consumed in hauling;[72] unless the rate of loading is increased, the improved haulage system cannot operate efficiently. The progress achieved in loading cars by hand shovels, scrapers, and mechanical loaders already has been discussed. The part played by chutes in loading operations and in promoting the efficiency of underground haulage remains to be reviewed here.

In mining an ore body of considerable vertical dimension, gravity generally is utilized to remove the broken ore from the mining face either directly from the stope or through chutes to points on the haulage level where it is loaded into cars. Before locomotives were introduced and haulage systems improved, little attention was paid to the spacing, construction, or design of the chutes. Chutes were apt to be poorly spaced, requiring a considerable amount of labor for shoveling the broken ore in the stope into the chute, and poorly constructed, involving excessive wear and leakage of fine ore. Since they permitted the "free fall" of rock from the mining floor to the haulageway, chutes frequently collapsed when the "hung up" ore was dislodged. To overcome these defects, many changes have been introduced. The chutes have been so spaced as to avoid a considerable amount of shoveling. Their linings have been improved to prevent leakage and to reduce wear and tear. In some mines the use of linings has been eliminated when the openings through rock can be maintained satisfactorily without them. The height and inclination of the chutes have been modified to safeguard against collapse as well as to increase durability. Instead of using long, vertical chutes, operators tend at present to break the continuity of the chutes by means of a series of offsets or, in some instances, to use inclined chutes[73] with similar offsets.

Chute-gates or chute-doors are employed to control the flow from the chutes. The earlier chute-gates were rather crude, consisting of short pieces of 2-inch planks placed across the mouth of the chute and held in place by cleats or angle irons attached to the sides. A considerable amount of time and labor was consumed in opening and closing the gate and removing the large pieces of rock lodged in the narrow openings. Often the flow could not be shut off quickly, and this resulted in overflowing, filling the track with the spilt-over material which had to be removed

[72] C. F. Jackson and J. B. Knaebel, *Underground Chute Gates in Metal Mines* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6495, mimeo., Aug. 1931), p. 1.

[73] In an inclined chute the weight of the ore is distributed along the incline; in a vertical one it is concentrated at the bottom, which is likely to give way when the weight becomes excessive.

before the car could be trammed, and, not infrequently, serious injuries to workmen. More rapid loading, necessitated by an improved haulage system, has come to require more efficient chute-gates as well as chutes. Chute outlet openings have been enlarged, gates designed to open and close more rapidly have been constructed of steel instead of wood, and the method of control has been improved by employing a system of levers that may be operated by hand or compressed air. Grizzlies, or gratings of timber, iron, or steel bars, have been installed over the inlet openings of the chutes to screen out the large rock and to prevent workmen from falling in.

Today three major types of chute-gates are in use: the stopper-board, the arc-gate, and the finger-board. Some mines employ one type, with its many variations; others utilize all three, with their various modifications.[74] The choice is determined by a number of factors, the more important of which are the nature of the ore (that is, whether it is wet and sticky or dry and easy-flowing and whether the fragments are large or small), the rate of loading required, the total tonnage to be handled during the life of the chute, the size of mine cars, and the cost of installation. The present tendency is to select the type best suited for the particular mine and to modify and improve it as circumstances demand.

Generally, the improvements made in chutes and chute-gates have accelerated loading, with the result that the over-all speed of hauling also has been more rapid, the productivity of the labor engaged in chute-loading as well as that employed in hauling has been increased, and the cost of maintenance has been reduced.

In connection with the discussion of chutes as means for loading cars, mention should be made of the use of scrapers.[75] In mines where conditions are favorable scrapers are employed to drag the ore from the working face to the nearest chutes. Thus the use of scrapers, together with chutes in such instances, has virtually eliminated hand mucking and hand tramming.

The concentration of mining operations on fewer levels, permitting the use of larger mechanical haulage equipment, has also influenced the efficiency of the haulage system. Formerly the levels were spaced about 100 feet apart. It has been found that longer intervals are more economical for some mines. Where conditions permit, therefore, the interval

[74] For a discussion of various types of chutes used in underground mines see Jackson and Knaebel, *op. cit.*

[75] The scraper is generally regarded as a loading device, although when it is employed to convey ore from the working face to chutes or cars it is in reality also a haulage machine. However, its use is limited to secondary or auxiliary haulage, as the economic working range under most conditions seems to be about 200 feet.

has been increased to 150, 200, 250, and even 300 feet.[76] In some mines transfer chutes are used to connect several levels and to convey the ore from these levels to the main haulageway. Similar raises are employed in some mines to transfer filling material from the surface to the lower levels. Thus the use of interconnected chutes has made it feasible to have a single haulageway serve a block of ore as high as 500 feet.[77] As a result of the increase in the level intervals and of the gathering of ore and the distribution of waste by interconnected chutes, the amount of haulage equipment has been materially reduced. Even more important is the fact that larger and more efficient equipment has come into use. The physical conditions of certain mines may, however, prevent them from resorting to the practices of spacing their levels farther apart or joining several levels by a chute. The cost of driving the raises, the subsequent expense of their maintenance, and the added cost of hoisting must be balanced against the savings to be derived from a smaller outlay for haulage equipment and from a more efficient haulage system.[78]

Other factors, perhaps less important than those already mentioned, also have contributed to the increase in efficiency of underground haulage. One of these is standardization of the rolling stock and the track system. The tendency in recent years has been to adopt equipment that is best suited to a particular mine and to standardize it insofar as conditions permit. In consequence, there has been a reduction in the cost of maintenance and, to some extent, in the cost of equipment.

Another factor is better coordination not only of the haulage system but also of the entire underground transportation system - hoisting as well as haulage. It has already been pointed out that by improving track lay-out and installing block-signal systems the speed of trains has been greatly accelerated. As will be shown in the following section on hoisting, adoption of the skip pocket or underground ore bin, which has permitted trains to discharge their load of ore while they are in motion, and use of multiple-compartment shafts and of improved hoisting equipment, which have enabled the hoists to move the ore rapidly to the surface, have also facilitated the movement of trains and have eliminated the congestion of traffic on the main haulageways and in the shafts.

In the future haulage efficiency will continue to increase. Locomotives will be improved further, and more of the storage-battery type will

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76 Gardner, Johnson, and Butler, *op. cit.*, p. 111.

77 G. J. Young, *Elements of Mining* (3d ed.; New York: McGraw-Hill Book Co., Inc., 1932), p. 165.

78 See W. O. Vanderburg, *Factors Governing the Selection of the Proper Level Interval in Underground Mines* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6613, mimeo., May 1932), 17 pp.

come into use. The $1\frac{1}{2}$-ton storage-battery locomotives will continue to replace hand tramming wherever conditions warrant. However, hand tramming will still be used in many mines where circumstances preclude the use of other forms of motive power. Cars for hand tramming will be more efficient; they will be made of high-tensile-strength steel, equipped with roller bearings, and increased to the maximum size permitted by the drifts. In locomotive haulage many of the cars now in use will be gradually replaced by advanced types - those that will be constructed of durable alloy steels and equipped with antifriction bearings and improved dumping mechanisms. In short, the haulage equipment will be further improved and standardized, and the haulage system will be better coordinated. There is a possibility that belt conveyors, now successfully employed in coal and iron-ore mining and in many respects the most efficient form of large-scale haulage equipment yet devised, may be introduced in block-caving copper mines where they may be used to the best advantage.

Hoisting.- Unless the hoisting system can transfer the ore to the surface as rapidly as it is broken, the mine as a whole cannot function efficiently. Consequently, advances in hoisting equipment and techniques are of utmost importance to efficient operation of underground mines. Within the past 50 years many fundamental improvements, both in equipment and technique, have been introduced to overcome difficulties created by the increasing depth of the mines and rising volume of ore handled. These advances have augmented considerably the output per worker in hoisting operations.

With the advent of steam, compressed air, and electricity have come many improvements in the machinery and equipment that have revolutionized hoisting practices. As a result of these developments it has become possible to conduct hoisting operations in a single large vertical shaft with multiple compartments instead of in a number of small, single-compartment shafts; to handle ore, men, and materials in a few large skips[79] or cages instead of a large number of small ones; and to realize considerable labor savings by reducing the size of the crew engaged in hoisting operations. Further improvements have been effected by scientific planning - by studying carefully the hoisting problem of each mine before adopting any method or equipment and by coordinating underground haulage with hoisting to prevent loss of time.

In the early days, when mines were shallow, hoisting was done largely by horse whims, which could hoist 15 to 20 tons from a depth of 100 feet

[79] A "skip" is a large bucket used for transporting ore or waste.

or 4 to 5 tons from a depth of 300 feet per 8-hour shift.[80] With the development of the steam engine, hoisting larger volumes from greater depths became feasible. Early steam engines were rather inefficient, but with constant improvements they could cope with the problem of increasing depth and growing production. They could handle 10- to 20-ton gross loads from 4,000- to 6,000-foot depths at speeds ranging from 4,000 to 5,000 feet per minute.[81] Even though they were able to hoist large tonnages from greater depths, they were efficient only when the mines were moderately deep. Moreover, operating costs were always high. The steam plants were designed for maximum load conditions, but because the hoists were operated for only a short time at peak (that is, at the start of the load), over-all efficiency was low and costs high. Since many mines were far from the source of fuel, the cost of steam generation also was likely to be excessive.

Electric energy was made available at some mines about 1890. It has been employed either to drive air compressors to supply air to hoists or as a direct source of power. The use of compressed air to run hoists at Butte, Montana, where hydroelectric power is available at low rates, was found to be highly successful and far superior to steam.[82] Electric power at first could only be used efficiently for small hoists. For large hoists it proved to be uneconomical, for the great amount of current required to start and accelerate the load engendered objectionable "peaks" on transmission lines. With the introduction of the Ilgner and similar systems in which energy is stored in large revolving flywheels and is used to start and accelerate the load, the problem of "peaks" was gradually solved; by 1910 electric hoists were replacing steam.[83] Since then many improvements have been made. The electric hoist has been made larger, more powerful, and more durable, and the head frames have been better designed and constructed. As a consequence, hoisting operations can be carried to lower levels, the capacity of cages and skips has been enlarged, and the speed of hoisting has been increased. The use of direct-current motors with variable voltage control has made it possible to accelerate and decelerate smoothly and rapidly. The efficiency of the electric hoist has been enhanced further by the installation of automatic control, which makes it unnecessary to signal and put the machine

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[80] Peele, *op. cit.* (1st ed.; 1918), p. 881.

[81] D. W. Brunton, "Modern Progress in Mining and Metallurgy in the Western United States," *Transactions of the American Institute of Mining Engineers*, Vol. XL (1910), p. 546.

[82] B. V. Nordberg, "The Compressed Air System of the Anaconda Copper Mining Co., Butte, Mont.," *Transactions of the American Institute of Mining Engineers*, Vol. XLVI (1914), pp. 826 ff.

[83] Brunton, *loc. cit.*

in motion by manpower and dispenses with much of the labor of hoist attendants.

Concomitant with the changes in motive power have come equally significant improvements in equipment. The loss of power through transmission from the driving engine to the hoisting mechanism has been reduced to a minimum by replacing the belt, shaft, and pulley with a direct-drive or gear-connected-drive mechanism. A decrease in the transmission losses is, of course, equivalent to an increase in the effective power of the hoist.

Advances in the drum design, too, have helped to increase the efficiency of hoisting. In the early days of the steam hoist a single drum was used to wind the cable. With the development of the system of hoisting in balance considerable economy in personnel and power has been effected. Balanced hoisting is done by means of a double drum (one drum winding a cable while the other unwinds a cable). Thus as one cage or skip is raised another is lowered. The advanced type of double drum usually consists of two drums which revolve upon a common shaft and which may be thrown into or out of action by clutches. The arrangement permits the drums to operate independently of each other or simultaneously, thus making it possible to hoist in balance, partly in balance, or out of balance. Conical and cylindroconical drums, in which the central portion is cylindrical and the ends are conical, have been introduced to reduce the power requirements for starting and acceleration, particularly in steam hoists. For the same reason reels and flat ropes have been employed in some mines.[84] For electric hoists equipped with the Ward-Leonard system of control and large flywheels, however, the present tendency in design is toward straight cylindrical drums, especially those of smaller diameter and greater speed.[85]

Another major improvement is gradual replacement of the cage by the skip.[86] Use of the cage involves loss of time both in loading the cars onto the cage and in removing them at the surface, requires labor to perform the loading and unloading, ties up mine cars in the process, and necessitates hoisting dead weight of the cars. Introduction of the

[84]The use of reels and flat ropes permits the layers of rope to be wound one on top of another. Thus the diameter of the winding section of the reel, which is at a minimum at the start, gradually increases as the cage or skip rises to the surface. This arrangement makes it possible for the hoist to use less power for starting and to accelerate at a faster rate than it otherwise would.

[85]L. Eaton, "Hoist Design and Construction," *Engineering and Mining Journal*, Vol. 133, No. 11 (Nov. 1932), p. 576.

[86]For a more detailed discussion of skips and cages see the serial articles by L. Eaton, published in *Engineering and Mining Journal*, as follows: "Mine Skips," Vol. 133, No. 5 (May 1932), pp. 281-4; "Mine Cages," Vol. 133, No. 6 (June 1932), pp. 345-7; and "Caging and Skip-Loading Devices," Vol. 133, No. 8 (Aug. 1932), pp. 435-41.

electric locomotive underground increases speed and capacity of the haulage system so much that unless an equally efficient hoisting system is devised congestion of the haulageways would result. Use of the skip in place of the cage solves the problem of a congested haulageway. The ore, hauled from various parts of the mine, is dumped directly into skip pockets or bins, where it is stored until hoisting operations may begin. The ore is then loaded into the skips and hoisted to the surface, where it is discharged. These functions may be performed automatically.

Adoption of the skip and skip pocket has brought about a considerable saving in labor and time. It has greatly reduced the labor previously engaged in loading and unloading, eliminated a considerable portion of the delay and congestion both at the shaft and the haulageways, and increased the capacity of the hoist by 25 to 40 percent by obviating the necessity of raising and lowering the deadweight of the mine cars.

Although cages are inferior to skips for handling ores, they have not been displaced entirely by the latter. Cages are used for general hoisting purposes in some of the smaller copper mines and for transporting workmen and material in some of the larger mines. Cages have been increased in capacity by enlarging their size and constructing them with multiple decks. This improvement is particularly important in areas where the 8-hour day runs from "collar to collar" and where it is therefore essential that the workmen be transported to and from their place of work with the greatest possible dispatch. In some mines the hoisting system is so constructed that the skips and cages may be interchanged. This arrangement permits the entire hoisting system to be readily utilized for handling ore at one time and workmen at another. The efficiency of the skips and cages has been greatly improved by the use of aluminum-alloy and nickel-alloy steel in their construction. As a result the weight of these containers has been reduced substantially and the capacity increased accordingly.[87]

Similar progress has been made in the construction of winding cables. Hemp and charcoal-iron ropes have given way to steel cables of higher tensile strength. They have been improved so much that hoisting operations have been carried to deeper levels without decreasing the capacity of the cages or skips or increasing the diameter of the cables.

The advances in hoisting equipment and in the underground haulage system have brought gradual concentration of hoisting activity in one vertical multiple-compartment shaft. By centering hoisting operations in a

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[87] G. Martin, "New Accessories Feature U. S. Smelting Underground Hoist Installations," *Engineering and Mining Journal*, Vol. 136, No. 3 (Mar. 1935), p. 129.

single large shaft instead of in many small ones a substantial gain in efficiency is obtained. The crews required to operate and maintain the many shafts are dispensed with. Power plants, which had to be located at the various shafts and which necessitated a separate personnel to run each of them, are consolidated into one, with a resulting saving in labor and capital. In the centralizing of hoisting operations the trend has been away from inclined shafts and toward vertical ones. It has been found that by substituting a vertical shaft for an inclined one of the same cross section the amount of material hoisted is increased from 15 to 20 percent without a change in the size of the skip or cage. A vertical shaft permits the skip to be filled to capacity, whereas one that is inclined allows the skip to be only partly filled. Moreover, a hoist in a vertical shaft can be run at a much higher speed than one in an inclined shaft. Since the skip or cage in an inclined shaft must travel on tracks, running it at high speed may result in derailment and a tie-up of the system.

Another factor that has explained some of the increase in efficiency in hoisting operations is the careful planning of the hoisting system. In the early days of rather small and shallow mines, hoisting was a relatively simple problem. With growth in size and increase in depth of mines, however, it has become more and more an engineering problem that requires the most thorough studying and planning before the equipment is adopted or designed. For example, in porphyry mines employing the caving method the problem is to devise a hoisting system capable of raising large quantities of ore at a rapid rate - as much as 1,000 tons per hour. The hoists used must therefore have speed, precision, and a large capacity - requirements that can be satisfied by a system operated automatically, powered by direct current with voltage control, and equipped with large skips arranged in balance.[88] In vein mines the problem becomes even more difficult. Although the tonnage to be lifted is smaller, the depth is much greater. In some deep mines the weight of the cable may be heavier than the combined weight of the skip and the ore. The ore must be gathered from a number of levels, moreover, and the cage or skip must be spotted accurately at various levels. An efficient hoist must therefore provide for a constantly changing load. Because of the long distance to be traveled, it must be operated at high speed to raise enough ore to the surface. It must, furthermore, be capable of rapid acceleration and deceleration.[89] Not only do porphyry and vein mines present

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[88] Jurden, *op. cit.*, pp. 155-8.
[89] *Ibid.*, pp. 160-2.

different hoisting problems, but each mine may present a unique problem requiring thorough analysis before a hoisting system is adopted.[90]

The future holds promise of further increases in hoisting efficiency. In porphyry mines no new hoisting problems will be likely to arise. One mine, using an electric hoist and twin shafts, has succeeded in raising 1,000 tons of ore per hour from a 600-foot depth.[91] Another, employing similar equipment but using a single shaft, has attained an average of 17,141 tons per day for 357 operating days.[92] In vein mines hoisting problems will become more difficult as the depth increases. Some copper mines have already installed hoisting plants designed for vertical shafts 5,100 feet deep, which is generally considered the maximum depth of single-stage hoisting for economical operation.[93] From the mechanical standpoint it seems feasible to hoist in a single stage from a depth of 7,000 feet in vertical shafts.[94] It is doubtful, however, whether this depth will be attained by many copper mines in the next 50 years. In inclined shafts, where part of the weight of the ore and skip is resting on the rails and that of the rope on the idlers, it is possible to hoist from a much lower level. One mine in Michigan is equipped with a plant that can operate to a depth of 14,000 feet measured on the incline.[95]

The major obstacle to an increase in depth, so far as hoisting is concerned, is the weight of the cage or skip and cable. Not many mines have yet utilized the new lightweight alloys for the construction of skips and cages, and there appears to be room for much improvement here. One metal mine, for example, by replacing its old steel skips with those made of aluminum alloy, reduced the deadweight to such an extent as to permit it to hoist from an additional depth of 1,200 feet without changing the diameter of the rope or causing greater stresses upon the hoisting drum.[96] Moreover, hoisting experience at the Rand area indicates that the factor of safety[97] of ropes may be safely reduced to as low as 4.7 for ore and 5.0 for men.[98] This means that the capacity of the skip may be held constant while the length of the cable is extended.

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[90]For a discussion of a modern, well-planned system of hoisting in deep copper mines, see the following articles in *Engineering and Mining Journal* by C. M. Harrer and L. S. Farnham: "Ore Hoisting in the Butte District: Part I," Vol. 129, No. 9 (May 8, 1930), pp. 441-5 and "Ore Hoisting in the Butte District: Part II," Vol. 129, No. 11 (June 7, 1930), pp. 555-9.

[91]Jurden, *op. cit.*, p. 155.

[92]A. B. Parsons, *The Porphyry Coppers* (1st ed.; New York: The American Institute of Mining and Metallurgical Engineers, 1933), p. 424.

[93]Harrer and Farnham, "Ore Hoisting in the Butte District: Part I," p. 441.

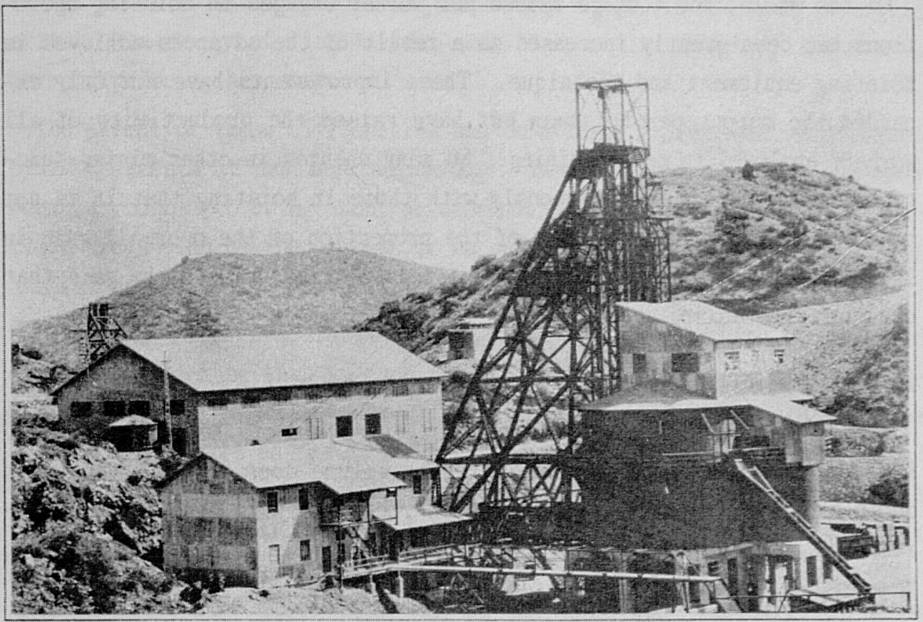
[94]H. H. Dow, "Deep Mine Hoisting," *Engineering and Mining Journal*, Vol. 135, No. 5 (May 1934), p. 200.

[95]Report of the Quincy Mining Company for the year 1930, p. 14.

[96]Martin, *op. cit.*, p. 130.

[97]The factor of safety is the ratio of the ultimate stress (strength per unit of area at the yielding point) to the allowable working stress.

Footnote 98 appears on p. 140.



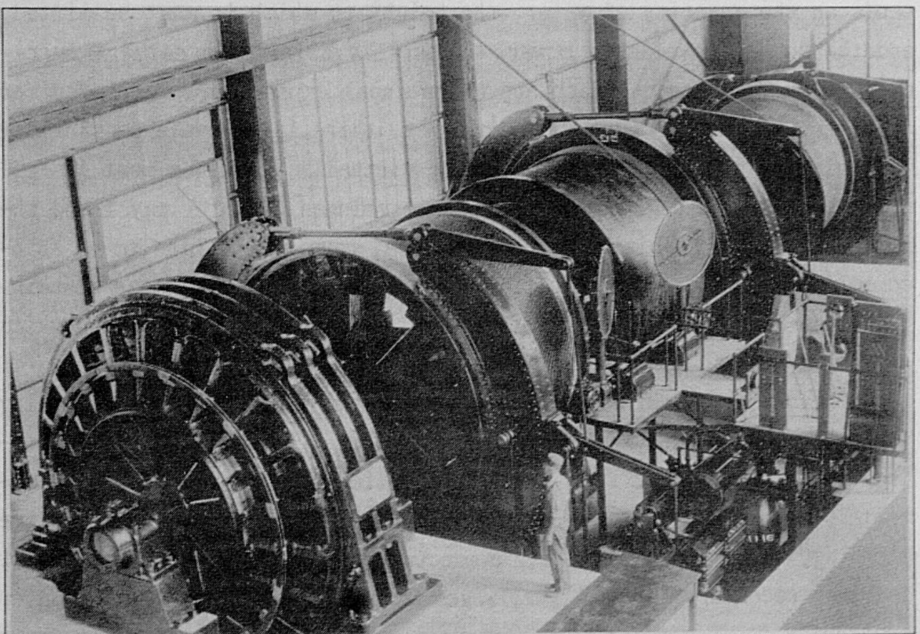


FIGURE 39.- MOVING LARGE TONNAGES TO THE SURFACE REQUIRES EFFICIENT HOISTING PLANTS AND TECHNIQUES

The upper view shows the steel headframe and wheels over which the hoisting cable passes, the cylindrical concrete ore bins into which the skips discharge, and the hoist house (left) which houses the ore and man-cage hoists shown in the lower view. The ore hoists are of the automatic type and can be operated from the underground loading station or from the hoisting platform at the surface.

On the whole, the tonnage raised per worker engaged in hoisting operations has been greatly increased as a result of the advances achieved in hoisting equipment and technique. These improvements have not only expanded the output per hoistman but have raised the productivity of all workers employed in copper mining. So many changes in other mining functions have occurred simultaneously with those in hoisting that it is not possible to state the magnitude of the proportion of the over-all gain in labor productivity in copper mining that is attributable to the gain that occurred in hoisting operations.

Mine Drainage

The presence of water in a mine constitutes one of the great physical handicaps of mining.[99] Fortunately the volume does not increase with depth except in rare instances, because rock pressures tend to close the fissures and open spaces through which subterranean water flows. It is, however, necessary to raise water from an increasingly lower level.

At all mines drainage is an expensive burden, the extent of which depends on the quantity of water, the height to which it must be lifted, and the cost of available power. Generally a large investment is required for the pumping plant and a continual expenditure is needed for its operation and maintenance. Moreover, there is always a possibility that mines may become flooded, so that virtually all mines must install some stand-by drainage equipment. Large volumes of water may flood the mine or entail prohibitive pumping charges, causing the mine to be abandoned. Even small amounts multiply mining difficulties. Water containing acids hastens destruction of pipe lines, rails, and equipment by its corrosive action. When it drips from the roof it may wet the clothing of the men and make working conditions uncomfortable for them, thus tending to lower their productivity. Acid-bearing water rots the miners' clothing and may cause open sores and inflammations on their bodies. In extremely wet workings it may be necessary for the men to wear protective clothing, which may reduce their productivity by about 20 percent.[100] It is often necessary to pay higher wages as compensation for working under such unfavorable conditions. When drill holes become filled with water extreme care must be exercised in placing and firing the explosives, and often special types of explosives and detonating agents must be used.

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[98] W. G. McBride, "Toward Greater Efficiency," *Engineering and Mining Journal*, Vol. 137, No. 2 (Feb. 1936), p. 85.

[99] Mine waters owe their source to surface water that is derived from rain, snow, rivers, and lakes; to deep-seated springs; and to water held in the strata when the rocks were formed. By far the most important is surface water, the volume of which is dependent on weather conditions and varies with the seasons.

[100] Young, *op. cit.*, p. 222.

In hot mines moisture creates high humidity which, in addition to accelerating the decay of mine timber, adds to the discomfort of the miner, thereby reducing his output.

The effect of these difficulties has to a large extent been neutralized by improvements in drainage technology. Several important advances have been introduced to bring about an economical drainage of mines. They include prevention of the ingress of water into the mine and its removal by natural or improved mechanical methods.

Obviously it is preferable and may be more economical to prevent the water from entering than to pump it out, even with the most efficient mechanical method. Operators usually take considerable precaution to place their mine entries as far from the surface water as possible. Ditches, culverts, or small dams are constructed to divert the water around the mine openings, outcrops, and fissures. Watertight collars are built around the shaft to prevent the water from flowing in. Where large volumes of water have found their way to the shaft near the surface, water rings are installed to prevent the water from dropping to the bottom, thus decreasing the height to which the water would have to be raised. Water-bearing strata may be sealed off by cementation.[101] Some mines may find it necessary or desirable to line shafts, crosscuts, and drifts with concrete or gunite to seal off water. Many dams are built underground to confine water to certain localities, to regulate flow to the pumps, and to isolate water in abandoned workings.

Where topographic conditions permit, tunnels may be driven in the mine to drain the water. Most of the mines, however, do not have sufficiently favorable topography and consequently are forced to resort to the use of mechanical means to remove the water. Two methods have been developed: one utilizing buckets and the other pipes.

In the earlier days hoisting in buckets or water skips was widely practiced, largely because of the simplicity of the plant. The engine was on the surface where it could be repaired easily if anything went wrong, long rods and underground steam lines were avoided, the dangers of rockfalls or squeezes that might disrupt pipe lines were obviated, and above all the plant could not be flooded. However, because of high initial and operating costs and the fact that the shaft compartment cannot be used for any other purpose while water is being hoisted, its application is limited. Today water skips are used only in small mines, in large mines where the quantity of water is small, or for emergency purposes.

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[101] The water-bearing strata are punctured by diamond-drill holes into which a thin cement mixture or other plastic material is pumped under pressure in order to fill cracks and crevices.

Air-lift and pump systems have been developed for moving water through pipes. In the former system the water is elevated by the introduction of compressed air at the foot of the column pipe.[102] It can move large quantities of water quickly, and because it has moving parts it is especially adapted for handling acid water. It is, however, limited to dewatering mines and to emergency use because of its low efficiency and the necessity of submerging the intake pipe 30 to 70 percent of its entire length.

Pumping is the most widely used system of mine drainage, and it is here that technology has made the greatest advances. In this system water is collected from the workings by means of ditches or gathering pumps and concentrated in underground reservoirs or sumps from which it is forced to the surface by powerful pumps. Two important types of pumps have been developed for mine drainage: the reciprocating and the centrifugal.

The earliest pumps were bulky, inefficient, and costly to operate and could not force water to a great height. As mines were deepened it became necessary to install sumps on various levels and to relay the water from one sump to another, each having its own pump. This system involved additional costs for additional pump stations and attendants. Constant efforts have been made to render the pumps more efficient and durable. Their design and construction have been improved to reduce power consumption to a minimum, to increase their capacity and the head against which they can work, and to protect the linings and moving parts against wear. Their dimensions have been kept down, thereby reducing the amount of space necessary for installation, and the parts have been made more accessible to facilitate repairs. Moreover, they are designed to meet specific conditions in each mine rather than for general purposes.

The Cornish pump was perhaps one of the earliest types of mechanical pumps to be employed in copper mines. These consisted of one or more single-acting[103] plunger pumps installed in the shaft and connected by long wooden or steel rods to a steam engine on the surface. Originally Cornish beam engines were used as prime movers, but gradually they were replaced by the more efficient Bull type, in which a vertical steam cylinder is mounted at the mouth of the shaft directly in line with the pump. The engine usually was single-acting, the steam raising the rod and plungers that pumped by the weight on the down stroke. Although the Cornish pumps were reliable and comparatively free from the dangers of

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[102]For a description of the air-lift method of drainage see Young, *op. cit.*, pp. 237-41.

[103]Single-acting pumps discharge water in only one direction of the stroke, whereas the double-acting pumps deliver water on both directions of the stroke.

flooding,[104] they were not widely employed, largely because of their high initial, erection, and maintenance costs and the massive machinery required for their operation. However, only a few copper mines were in existence that required deep-level pumping equipment before 1876, when many types of direct-connected steam pumps became available for mine service.

Before 1890 the direct-connected, single-plunger steam pumps, which had been developed in the coal industry in the United States and in the coal and metal mines in Europe, were being used underground. These units were greatly advanced over the Cornish type, for they were compact, direct-connected, more convenient to install and maintain, and low in initial cost. They were nevertheless wasteful in their utilization of steam and were gradually replaced by the more economical duplex, compound, triplex, and quintuplex units. These latter types of pumps are designed for specific purposes.[105]

Meanwhile there was a shift from steam to compressed air as a source of energy. This change was due to several factors. In the first place, as mine workings became deeper the steam losses in transmission increased; in the second place, it was difficult to dispose of the exhaust steam, which fogged the air, increased the temperature, and accelerated "dry rot" of the timbering. The shift not only eliminated these objectionable features but also aided ventilation and permitted pumping operations to be carried on in isolated areas of the mine.

Later, operating and maintenance costs were materially curtailed with the electrification of mines and the application of electric energy for driving pumps. However, the change to electricity came about only gradually. The early motors were high-speed and not suitable for driving the slow-moving pumps. It was consequently necessary to modify the speed of the motors by means of some gearing device. This defect, however, was remedied with the invention of low-speed motors that could be connected either directly or through single-gearing mechanism. The new types of motors were a decided improvement over the steam and compressed-air engines, for they were 20 to 40 percent more efficient, much more compact, easier to install, and subject to remote control, and they could be shifted readily from place to place.[106]

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[104] Because only the pumping parts were located underground and because they could work as well under water as out, there was no danger of drowning out the plant.

[105] For sinking service, for example, the pumps, generally of the plunger type, are made compact, capable of operating under variable heads, and sturdy enough to withstand the severest type of service. These pumps have a capacity of 50 to 1,000 gallons per minute and can pump against a head of 200 to 400 feet (Young, *op. cit.*, p. 231). Station pumps are designed to permit the plungers to work either horizontally or vertically, the latter type being preferred since it reduced the width of the underground pump room. Plunger speed is 100 to 200 feet per minute. Where large capacities are desired they are made to move water on both directions of the stroke.

Footnote 106 appears on following page.

Centrifugal pumps, when first introduced, were restricted in use because they could handle only small amounts of water and because they were single-stage. The height to which they could force water was limited to about 150 feet. Around 1908 multistage units, which had become popular in Europe, were introduced in this country. By 1909 several types were available for mine drainage, namely, single-stage pumps pumping 35,000 gallons per minute against a 150-foot head, 5-stage units pumping 10,000 gallons per minute against a 600-foot head, and 8-stage units pumping 400 gallons per minute against a 1,400-foot head.[107] It was, however, not until about 1914 that the multistage all-centrifugal units were in general use in this country. The centrifugal, upon its introduction, began to replace the reciprocating type as a station pump. By 1929 it was the most-used type in spite of the fact that the reciprocating type has a higher mechanical efficiency.[108]

Comparison of the centrifugal and the reciprocating types indicates that both have advantages and disadvantages, depending on the pumping requirements.[109] The centrifugal pump is compact and simple in design, having few moving parts and requiring little attention. It is designed for an optimum speed and a fixed capacity; operation below and above results in a loss in efficiency. It is especially suited for pumping a large volume of water under a low head. The reciprocating pump, on the other hand, is positive in action, rugged in construction, and capable of working under varying heads. It is well adapted for pumping against an unusually high or greatly varying head. Being positive in action, it is likely to suffer serious damage should it be overloaded, as when the discharge pipe is choked or a valve is closed; the centrifugal pump may suddenly be overloaded without damage to its mechanism.

Another step forward was made when automatic control of pumping was developed in 1922. This cut down station attendance to periodic visits and inspection, curtailed operating and maintenance expenses, and at the same time increased the efficiency of mine drainage.[110] Automatic pumps,

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[106] About 1908 the express type of plunger pump was developed. It could move large volumes of water against a comparatively high head, was driven with a compact, lightweight motor unit, and was run at relatively high rotative speed. By 1909 pumps of this type could raise 1,600 gallons 1,550 feet per minute (Brunton, *op. cit.*, p. 548). Although it was more efficient and compact than the slower-moving plunger pumps, its use did not spread widely because of numerous break-downs resulting from its high speed and intricate mechanism.

[107] *Ibid.*, p. 549.

[108] "Mine, Shop, and Plant," *Engineering and Mining Journal*, Vol. 128, No. 12 (Sept. 21, 1929), pp. 480-1.

[109] For a discussion of the relative merits of the two types of pumps see the following articles in *Engineering and Mining Journal* by L. Weaver: "Mine Pumps and Their Selection: Part I," Vol. 137, No. 12 (Dec. 1936), pp. 621-4 and "Mine Pumps and Their Selection: Part II," Vol. 138, No. 1 (Jan. 1937), pp. 15-9.

[110] A. H. Hubbell, "Controlling Centrifugal Pumps Automatically," *Engineering and Mining Journal*, Vol. 121, No. 25 (June 19, 1926), pp. 997-1001 and Hubbell, "Economies and Efficiencies in Mine and Plant," p. 9.

which may be operated for many days without attendance, are particularly useful in preventing the mine from being flooded if a cave-in that may prevent the entrance of the operating crews should occur.[111]

The presence of acid or acid salts in the water of some mines may cause rapid corrosion of the pumping equipment unless proper precaution is taken. Many improvements have been introduced to prevent corrosion. In some mines that employ plunger-type pumps the pump interior and the pipes are lined with corrosion-resistant materials, caulked, and painted with acid-resisting paint. The plunger, which cannot be lined, is constructed of such acid-resisting alloys as brass, phosphor bronze, or sometimes porcelain. In other mines, the pump and pipes are lined with lead or rubber. Where the centrifugal type is employed the entire pump, except for the shaft and casing, is constructed of bronze. Chemical treatment of the water before pumping is practiced in some mines where acid content is high and soluble copper is present. Some mines neutralize the water with slacked lime, and others add scrap iron to precipitate the copper out of the water.[112]

With the development of larger capacity and more efficient high-lift pumps, efforts have been made to concentrate the water at fewer pumping stations.[113] This practice has made possible replacement of many small pumping units with a smaller number of larger, more efficient ones; this, of course, has brought about a reduction in operating costs and a decrease in the amount of labor employed.[114] The water from each level is concentrated at one point and transferred through water boxes in the shaft or through boreholes to the main stations from which it is pumped to the surface. As a result it has become necessary to increase the capacity of the underground sumps or reservoirs. Where it is possible to do so, sumps are constructed with a capacity large enough to take care of the inflow of water while pumping equipment is being repaired. Large

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[111]W. B. Clark, "Modern Automatic Pumping at Consolidated Coppermines," *Mining and Metallurgy*, Vol. 14, No. 319 (July 1933), pp. 295-6.

[112]H. L. Lavender, "Water Treatment and Underground Leaching at Bisbee Mines of Calumet and Arizona Mining Company," *The Mining Congress Journal*, Vol. 12, No. 9 (Sept. 1926), p. 663.

[113]For example, at Butte, Montana, practically all of the underground water from 23 or more mines is drained into five central pumping stations located on the 1,200-, 2,200-, and 2,800-foot levels. The greatest quantity of water is delivered to the 2,800-foot level, where it is forced to the surface in several stages. See W. B. Daly and Others, "Mining Methods in the Butte District," *Transactions of the American Institute of Mining and Metallurgical Engineers*, Vol. LXXII (1925), p. 271.

[114]With the increase in depth it became necessary in 1922 for the Calumet and Arizona mine to improve its pumping equipment and methods. Prior to this time, 1 electric and 10 steam pumps had been in use. The former could pump water from the 1,800-foot level in a single lift. The steam units had to pump the water in two stages, five of them forcing it from the lower level to the 1,000-foot level and five others lifting it to the surface. The installation of three motor-driven pumps at the 2,200-foot level and the transfer of another from the 1,800-foot level to the lower level eliminated all the steam units and reduced the number of operators from 17 to 7. See E. F. Irving, "Deep Mine Pumping at Calumet and Arizona," *The Mining Congress Journal*, Vol. 12, No. 11 (Nov. 1926), pp. 805-6.

sumps offer a number of advantages, the most important of which are that emergency or stand-by equipment can be eliminated; operating costs can be reduced, for pumping can be done during "off-peak" power periods; the mine is safeguarded against flooding in the event of a break-down of the pumping equipment; and the sand and silt, which are destructive to pumps and valves, are permitted to settle, resulting in prolonging the life of the pumping equipment and curtailing expenditures for repairs and maintenance.[115]

Another important step toward increasing pumping efficiency has been made by more careful planning and selection of the equipment for each station unit. Since the maximum flow occurs only part of the year, pumping is usually not concentrated in a single unit, for equipment that could handle the water during the wet season would be much too large during the dry. Moreover, it would require a 100-percent duplication of equipment to safeguard against pump failure. The present practice is to study the drainage problem of a mine thoroughly before devising a pumping system. The maximum and minimum requirements are estimated and allocated to several pumps, which are designed to meet specific drainage conditions. Only as many units as are needed at any particular time are utilized. This scheme means that only a fraction of the pumping capacity is affected if one of the pumps should fail. It is also common practice to have all pumps of the same type and of equal capacity. Although this is not a necessity, it is desirable from the standpoint of repairs, since fewer parts need be carried in stock.

It may be said that the improvements in drainage technology have greatly increased the productivity of labor engaged in drainage operations. Inasmuch as only a very small proportion of the total number of mine workers is actually occupied with mine drainage, the most significant fact is not that these advances have raised productivity of drainage operations but that in overcoming some of the mining handicaps created by the presence of water in underground workings they have raised the over-all output per worker, reduced mining costs, and made it possible to extract ore which otherwise could not have been mined.

Mine Supports

Improvements in the materials for mine supports and in the method of constructing the supports for underground openings, although less spectacular than those in the other phases of copper mining already discussed, have nevertheless had an important effect on the productivity

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[115] See L. Weaver, "Mine Pumps and Their Selection," *Engineering and Mining Journal*, Vol. 138, No. 3 (Mar. 1937), pp. 143-6.

of labor. The task of obtaining, transporting, erecting, and replacing artificial supports has always consumed a substantial portion of the miners' time. The amount of labor required for this purpose, of course, depends on the mining methods employed and the kind of ground being mined. Some methods virtually dispense with artificial support, others use temporary structures which are abandoned as soon as the ore is removed, and still others must employ more carefully constructed props. Where the ground is heavy, stronger and better supports must be constructed. In all mines, however, regardless of the methods utilized, some artificial supports are necessary in the shaft, drifts and crosscuts, and airways.

Changes in the design and construction of mine supports, together with the tendency to utilize mining methods that employ as little timbering as possible, have reduced progressively the labor time used in erecting supports, thus permitting the miners to spend more time in drilling, blasting, and mucking and to increase their daily output. At the same time, however, physical handicaps, particularly depth, have been increasing in some mines, necessitating the use of stronger, more elaborate supports and hence a greater expenditure of labor. Thus the gain in productivity resulting from advances in the technique of constructing supports has been to some extent offset by increasing mining difficulties.

Timber, steel, and concrete are the materials employed most commonly in the construction of supports for underground copper mines. Installations of steel and concrete, although constantly increasing, particularly in places where the service is severe, have been rather limited,[116] largely owing to the fact that the initial cost is high, that few mine openings have a long enough life to require the durability of steel or concrete supports, and that these materials fail rapidly once initial failure has set in. Steel and concrete are used as substitutes for timber largely in shafts, main haulageways, and main crosscuts, wherever the ground is solid and not subject to movement. In heavy ground, however, they are poor substitutes for timber. Although they have great strength, they are relatively inelastic, and are likely to fail rapidly when the load exceeds their carrying capacity. By far the most common and most important material for the construction of mine supports is timber. Because of its availability, low initial cost, light weight, high strength, and the ease with which it is cut to shape or length, its tendency to fail gradually when subject to overload[117] and to warn the miners of impending

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[116] It is estimated that only 4 percent of the timber in metal and coal mines has been displaced by steel and concrete. See W. D. Brush, *Timber Requirements for Mines in the United States* (U. S. Dept. Agr., Forest Service, mimeo., Jan. 1938), p. 4.

Footnote 117 appears on following page.

failure by cracking sounds, it is still the most suitable material. Moreover, it may be cheaper than steel or concrete even after allowance is made for frequent replacements.

One of the most important advances in the utilization of timber for mine supports is the standardization of timber sets, which has reduced timber costs and at the same time increased the productivity of labor. In the early days of copper mining unpeeled logs were sent underground, where they were cut to fit by miners who used hand tools and worked under poor light in cramped quarters. With standardization of timber joints and uniform spacing of timber sets the task was transferred to the surface, where the framing was done by carpenters who, although still employing hand tools, had a far higher productivity than the miners who did their framing underground. The introduction of framing machines and the concentrating of timber framing in a few shops have further increased the productivity of the workers engaged in constructing supports.[118] Sawed timber, which is uniform in length, lighter in weight, simpler to store and transport, and easier to frame, has been replacing the round, unpeeled logs. Standardization of sets has not only increased the productivity of labor engaged in framing timber but has also reduced the amount of labor required to replace broken timber. As will be seen later, the labor cost for timber replacement usually is much higher than that for installing the timber.

Another important advance is the improvement in the method of handling timber. Formerly the general practice was to lower each day's supply of timber to the working level in cages or skips, transfer it to supply cars, and haul it to the working place where it was installed by hand. Little attention was paid to the length of the logs; in consequence, the timber that was unsuitable for installation was left to rot in working stopes. Since the advent of scientific planning the handling of timber has been revolutionized. The proper amount of timber of the appropriate sizes is loaded on special timber trucks at the surface and lowered to the mines in cages. In some instances it is lowered from the surface to the underground levels through special supply shafts and is hauled directly to the place where it is to be installed. In some mines enough

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[117] Timber, because of its cellular and fibrous structure, is relatively pliable. This quality makes it particularly well adapted for use in heavy grounds where the mine support must not only resist pressure but must at the same time yield sufficiently so as to avoid a sudden collapse. See R. D. Parks, "Recent Developments in Methods of Mining in the Michigan Iron Mines and Timbering Practice in the Michigan Iron Mines," *Bulletin 1928-1929*, Vol. 2, No. 3 (Houghton, Mich.: Michigan College of Mining and Technology, 1929), p. 55.

[118] A carpenter using hand tools could frame 20 pieces of timber in 8 hours. When the single-end framer was introduced two men could frame 250 pieces in 8 hours; with the double framer two sawyers could frame 300 pieces of timber; with the cutter-head type of framer two men could frame 900 to 1,000 pieces (Peele, *op. cit.* [1st ed.; 1918], p. 552).

timber for several days' use is stored underground. In the more modern mines portable air or electric hoists are used to lower and raise timber through winzes, raises, and other underground workings.

Still another significant improvement is the extension of the life of mine timber.[119] The life of mine timber may be shortened by fire, wear and abrasion, breakage from crushing and squeezing, destruction by insects, and decay caused by molds and fungi. The danger from fire has been minimized considerably by inaugurating fire-prevention measures, by using a coating of concrete or gunite, and by treating the timber with noninflammable preservatives. Wear and abrasion have been abated by protecting the timbers with steel sheeting. Although crushing and squeezing because of overloading are beyond human control it is nevertheless possible to safeguard against these contingencies to some extent by selecting oversize timber, using special covering sets, commonly known as jacket sets, and installing the timber sets with more care. Decay and destruction by insects have been reduced substantially by peeling, seasoning, prompt shipment, careful storing, improved ventilation, and preservative methods. Of the various means of prolonging the life of mine timber, treatment with preservatives is the most important.[120]

It should be pointed out that it is unnecessary to treat timber that is to be used in temporary working places. In stopes, where operations are carried on for a comparatively short time, it is more economical to install untreated timber, which is certain to outlast these working places. However, in the more permanent openings, such as the shafts, ventilation arteries, and haulageways, which are to be in service for 5 or more years, treated timber is unquestionably advantageous. The average life of untreated timber in such semipermanent openings, where crushing is absent, is about 3 years.[121] Where conditions promote decay (such as the presence of moisture and stagnant, warm air) the life of untreated timber is, of course, much more limited. The length of life of preserved timber, like that of untreated timber, depends on the condition of the mine and type of service. Such fragmentary service records as are available indicate that in openings in which crushing is not a factor timber well treated with preservatives outlives three or more replacements of

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[119] For a detailed discussion of the methods of prolonging the life of mine timber see R. R. Horner and H. E. Tufft, *Mine Timber; Its Selection, Storage, Treatment and Use* (U. S. Dept. Int., Bur. Mines, Bull. No. 235, 1925), pp. 65 ff.

[120] Timber treatment has also undergone many changes since its introduction at metal mines in 1910. At present many preservatives are available. They include tars, creosotes, petroleum, zinc chlorides, sodium fluoride, copper sulphate, mercuric chloride, and proprietary compounds which are supplied by methods varying from the simple brushing or open-tank processes to the more complex methods of vacuum or pressure systems using heat. (*Ibid.*, pp. 80-98.)

[121] *Ibid.*, p. 25.

untreated timber. If it is assumed that the average mine shaft and haulageway will be in service for 12 years, one installation of treated timber will suffice, whereas three or more renewals of untreated timber will be necessary. In general the labor cost of installing the original timber ranges up to the cost of the timber or even more. In timber replacements, however, the labor cost usually is at least double and, in some instances, many times the cost of the timber,[122] notwithstanding the fact that standardization of timber sets has decreased materially the amount of labor required to replace deteriorated timber. To this high cost of timber renewal must be added the loss of labor time caused by interruption of mining operations while the timber is being replaced. As a result of timber treatment that eliminates or delays the necessity of replacing mine timber, the productivity of labor in underground copper mining has been enhanced substantially.

In summary it may be said that improvements in the methods of constructing mine supports have joined with other technologic changes to promote productivity in underground copper mining. The increasing use of steel and concrete in mines, where the service is severe; the standardization of timber sets, which has permitted the framing of timber to be transferred from the mines to the surface, where machinery may be employed; improvement in the method of handling and placing timber in the mines; and the lengthening of the life of mine timber by introducing fire-prevention measures, selecting timber with a sufficient tolerance to withstand crushing, improving ventilation of the mines, and, especially, treating timber with efficient preservatives have played an important part in counteracting the growth in physical handicaps and in increasing the output per worker in underground copper mines.

Further improvements may be expected. More steel and concrete will be employed in those mines in which the service will warrant their use. As only a beginning has been made in prolonging the life of the timber used in mine openings in which supports of great durability may be profitably employed, and as the cost of timber may be expected to increase with the growing scarcity of the timber supply, mining concerns undoubtedly will lay greater emphasis on the extension of the life of mine timber, particularly by means of chemical treatment.

Ventilation

Improved ventilation facilities have been, in recent years, important factors tending to increase output per worker in deep underground mines

[122] See H. E. Tufft, "Treated Timber As a Mining Asset," *Engineering and Mining Journal*, Vol. 136, No. 12 (Dec. 1935), p. 608 and E. H. Rieman, "Efficient and Effective Mine Timbering," *The Mining Congress Journal*, Vol. 21, No. 7 (July 1935), p. 53.

where air is affected by high temperatures and humidities, injurious gases, dust, and smoke. Advances in ventilation techniques have to a large extent offset these increasing difficulties by affording cleaner and cooler air, drier workings, and a clearer atmosphere. The resulting improvements in working conditions reduce labor turn-over, decrease the number of accidents, make for greater comfort and better health of workers, and increase productivity of labor. With the adoption of mechanical ventilation systems compressed air which was formerly used to some extent for ventilation is no longer needed for this purpose. Finally, timber decay is reduced materially and a positive control of air circulation is maintained.[123]

In the nineteenth century, when copper mines in the United States were shallow, natural ventilation and human endurance were made to suffice. The concentration of noxious gases from blasting was the only apparent difficulty to be overcome, and natural ventilation, if given time, ultimately would dilute the foul air to such an extent that it could be endured. However, as workings were driven to greater depths and farther from the main entries the loss of labor efficiency, resulting in higher operating costs, became more apparent, and the need for mechanical ventilation began to be recognized by mine operators. Poor ventilation was responsible for much lost time. In mine workings where the air was smoky, dusty, and deficient in oxygen miners had to come to the surface or to the main air courses for a breathing spell. In many small isolated stopes a crew actually worked only about 50 percent of the time or less.

One of the earliest ventilation devices consisted of a small wooden duct which was brought through the tunnel to the vicinity of the face and which set up a small circulation of air by taking advantage of the difference between the temperature in the room and that at the surface. Sometimes the duct was led into the base of a stove built of stones on the hillside near the portal of the mine. The heated air in the stove created a suction which drew the air from the mine. Later the wooden duct was displaced by galvanized-iron pipe that conducted air to or from the working faces of the levels, but the pipes, as well as the wooden ducts, seldom were extended into stopes.

Introduction of the air drill improved working conditions greatly. The exhaust of the drill did not only furnish fresh air to the stopes where it was most needed but also tended to stimulate natural circulation. Moreover, the presence of compressed-air lines throughout the mine

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[123] Positive control of air circulation may be defined as the ability to maintain air currents in a desired direction regardless of the difference in temperature between the surface and the underground.

permitted the workmen to stop at outlet stations to revive themselves when necessary.

Nearly all of the underground copper mines operated with natural or crude artificial ventilation until the heat, humidity, gas, smoke, and dust became so intolerable that large bonuses had to be paid to induce the miners to remain. Despite the payment of bonuses labor productivity declined. Mechanical ventilation thus became almost a necessity. Since the underlying causes for the need of mechanical ventilation differ widely, depending on the district in which the mine is situated, geologic structure of the ore body, and the method of mining used, it may be advisable to outline briefly the operating conditions in the principal producing areas of Michigan, Arizona, and Montana, where most of the deep mines are situated.

Michigan mines have little need for primary mechanical ventilation (except for the positive control in the event of fire), but utilize auxiliary ventilation to improve conditions at the working faces.[124] The open-stope method of mining, low surface-air temperatures, and low humidities - natural advantages with respect to ventilation - are enjoyed by virtually all mines of the district. Even though operating zones are 2,000 to 6,000 feet vertically below the surface, there are relatively few working places in the district that are not cool, comfortable, and well ventilated. Investigations in two of the deepest mines in the district showed that the lower stopes and development faces are in rock zones having virgin temperatures of 90 to 100 degrees; however, the natural dryness of the workings and the presence of saline mine waters result in such low humidities, and the air temperatures are so much lower, that working conditions here correspond to a "warm" rather than a "hot" mine.[125] Uncomfortable workings in localized areas, however, do require relief by auxiliary ventilation practices.

The mines of Arizona and Montana are confronted not only with high humidities and increasing rock temperatures but also with additional ventilation problems created by the method of mining and, at times, by mine fires. Underground mines of Arizona fall roughly into two broad groups: (1) large-scale caving operations of low-grade ore, in which the handicaps in the forms of temperature and humidity are at a minimum, but in which the workings are filled with dust, smoke, and gases produced continuously by the mining method employed; and (2) moderately deep mines

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[124] The circulation of air through the principal openings of a mine may be called primary ventilation. Circulation induced in working places away from the main channels by means of fans and other mechanical methods may be termed auxiliary ventilation.

[125] G. E. McElroy, *Natural Ventilation of Michigan Copper Mines* (U. S. Dept. Com., Bur. Mines, Tech. Paper No. 516, 1932).

operating on fissure veins or irregular replacement deposits under high temperatures that are attributable to the high geothermal gradient,[126] compression of the air column,[127] airway resistance,[128] heat generated by the oxidation of sulphide ores and by the decay of mine timber, and heat given off by electrical motors and men. In the Montana mines, which are similar to the second type of mines in Arizona, the conditions in some cases were so uncomfortable that mining companies paid bonuses to attract workers. In Butte, for example, a bonus of 25 cents a day over the regular wage was paid for work in sections of a mine having unusually poor ventilation, the working shift was reduced from 8 to 7 hours, and while one man was working, his partner sat near a compressed-air outlet to cool and to recuperate. The maximum efficiency of the worker, who had to be a man of exceptional endurance, was much less than 50 percent, probably not over 30 percent on the average; yet, he received a bonus of $12\frac{1}{2}$ percent in terms of a reduced length of shift, and an increase of 25 cents per shift.[129]

Even after the needs for effective ventilation were recognized the change from natural ventilation, which was at times supplemented by crude methods of artificial ventilation, to mechanically driven fans was gradual. This was due not only to the lack of knowledge by operators of the principles of proper mine ventilation and of the effects of heat, humidity, and mine gases on the human system, but also to the view, especially among old-time mining men, that circulation of air at the working faces was necessary only at coal mines.

Shortly after 1900 metal mines found it necessary to install electrically driven auxiliary fans (United Verde in 1906, Anaconda in 1910) to combat the high temperatures and humidities as the depth of the mines increased, to remove the smoky and dusty air, and to aid in fighting and

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[126] Geothermal gradient may be defined as the rise in rock temperature with depth. It is practically constant for any given locality, but it is extremely variable for different parts of the country. In the Michigan mines the rise in temperature was about 1 degree F. per 95 feet in depth. The initial rock temperature was rather low, being only 90 degrees at a vertical depth of 5,000 feet. (McElroy, *op. cit.*, p. 8.) In the Montana mines the gradient of increase was about 1 degree per 100 feet in depth. The rock temperature was much higher than that of the Michigan mines. It was 100 degrees at the 3,200-foot level. (D. Harrington, *Underground Ventilation at Butte* [U. S. Dept. Int., Bur. Mines, Bull. No. 204, 1923], p. 21.) In Arizona the increase in the rock temperature varies with the different localities, ranging from 1 degree per 50 feet to 1 degree per 85 feet (G. E. McElroy, *Ventilation of the Large Copper Mines of Arizona* [U. S. Dept. Com., Bur. Mines, Bull. No. 330, 1930], p. 136). The initial temperature was higher than that of Michigan or Montana. At the Magma mine, for example, it was 140 degrees at the 4,000-foot level. (C. B. Foraker, "Ventilation and Air Conditioning of the Magma Mine," *Mining Technology*, Vol. 2, No. 5 [Sept. 1938], A.I.M.E. Tech. Pub. No. 979, p. 1.)

[127] As air descends into a deep mine shaft its temperature increases about 5.5 degrees F. per 1,000 feet. This rise in the temperature is due to the adiabatic compression of the air column.

[128] In passing through the mine labyrinth the air encounters resistance which causes friction and generates heat.

[129] Harrington, *op. cit.*, p. 115.

controlling mine fires. Later, with the development of galvanized-iron pipe and canvas tubing that replaced the clumsy wooden duct as an air conductor, auxiliary units were placed at strategic points throughout the mine, particularly after it was noted that the cooling effect produced by resultant good circulation greatly increased labor efficiency. In 1916 the Anaconda Company transferred its ventilation problems to ventilation engineers who, together with the safety engineers, introduced systematic planning of ventilation. By 1918 this concern had installed in its mines at Butte, Montana, approximately 150 fans. Most of them were placed underground, some acting as blowers and some as suction units.[130] By 1933 Anaconda had installed 450 fans;[131] by 1938, over 600 fans.[132]

With the driving of additional shafts required by safety laws, surface exhaust fans of larger capacity have been installed. These supplemental shafts not only have improved ventilation by permitting natural circulation and the installation of more fans but also have provided additional exits through which the workmen may escape from the mine if a disaster occurs. Friction of moving air, which tends to create heat, is decreased by guniting downcast shafts and by placing smooth-surfaced boards in upcast shafts. Before the beginning of the World War practically no attempt was made to conserve and regulate the air supply and to force it to vital working faces in blind ends of raises, drifts, and stopes. At present abandoned areas are completely sealed off and sufficient air is provided at each active place by systematic air splitting.[133] Mechanical devices alone are not enough, however, to combat high temperatures and humidities in some of the hot mines.[134] In such mines it has been necessary to install air-conditioning systems. In the Anaconda group of mines experimental work in air conditioning has been conducted for the past 8 years, and satisfactory results have been obtained.[135] The lower levels of the Magma mine in Arizona have been completely air-conditioned,[136] obviating the former practice of laying off men during the summer.

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[130] *Ibid.*, p. 8.

[131] Daly and Richardson, *loc. cit.*

[132] W. B. Daly, "Metal Mine Hazards," *Safety Engineering*, Vol. 76, No. 3 (Sept. 1938), p. 12.

[133] Air splitting may be defined as the diversion of the air from the main channels of the mine into raises, drifts, and stopes. The volume of air entering the various workings is regulated by doors, bulkheads, and other devices.

[134] The surface air, no matter how cool or how free from humidity, will not be efficacious in ventilating a deep mine in which the thermal gradient is high. In Butte, for example, in winter the air leaving the surface at a temperature of 20 degrees F. below zero is heated to a temperature of 65 degrees or more before it leaves the inlet shaft at stations below the 3,500-foot level. A further increase of 5 to 15 degrees may be expected in the crosscuts and laterals before the air reaches the working zones. See A. S. Richardson, "Air Conditioning for the Ventilation of Butte Mines," *Engineering and Mining Journal*, Vol. 139, No. 10 (Oct. 1938), p. 29.

[135] See Daly and Richardson, *op. cit.*, pp. 231-44 and Richardson, *op. cit.*, pp. 29-34.

[136] E. P. Palmatier, "Cooling Magma's Lower Levels by Mechanical Refrigeration," *Mining and Metallurgy*, Vol. 18, No. 368 (Aug. 1937), p. 385.

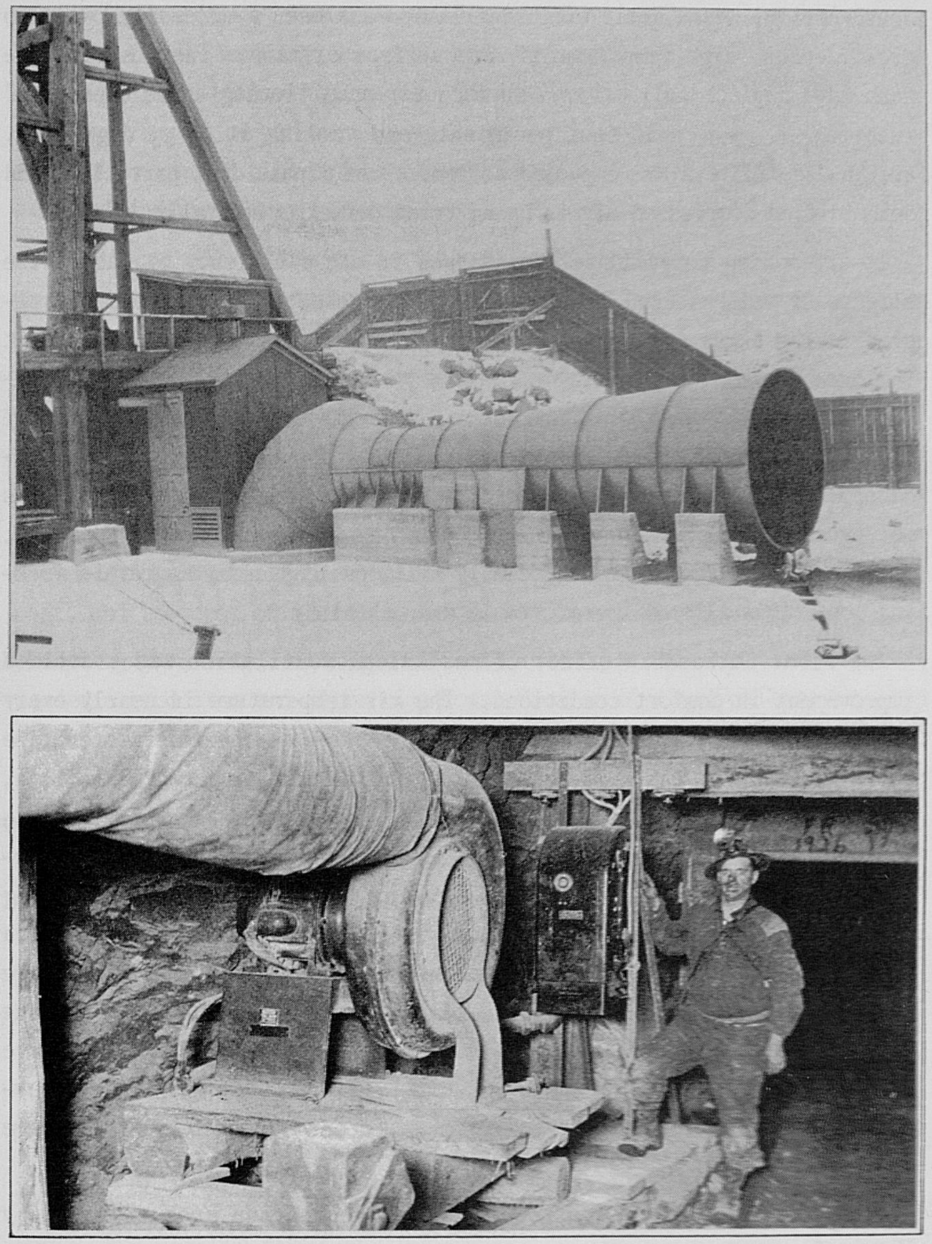

FIGURE 40.- INCREASED USE OF MECHANICAL VENTILATION DEVICES WAS NECESSARY WITH THE EXPANSION OF OPERATIONS AT UNDERGROUND MINES

The upper view shows surface-type fan equipment used to force air down the mine shaft. The lower view shows a small blower fan, to which canvas tubing is connected, used to ventilate the dead ends of the crosscuts and raises. At some copper mines air-conditioning equipment has been installed.

Future developments in mechanical ventilation, together with air conditioning, probably will follow the pattern developed in the Rand area in South Africa, where artificial ventilation has been a necessity owing to great depths, high temperatures, and surface climate. There screw-type fans have replaced all others, surface air-conditioning units capable of treating 400,000 cubic feet per minute and cooling it to 33 degrees F. are in use, filters are employed to remove the minute dust particles, and dehumidified compressed air is being tried out experimentally.[137]

In attempting to evaluate the changes in the efficiency of labor, the effects of temperature, humidity, injurious gases, and dust-laden particles on the human system must be considered. It has been estimated that in mines where the temperatures are above 75 degrees F., where humidities are above 85 percent, and where there is little or no movement of air workers may lose 25 percent to as much as 75 percent of their efficiency.[138] Their health, too, is likely to be affected adversely unless moving currents of fresh air are supplied to working places. Moreover, the presence of fine dust, especially siliceous dust, may aggravate working conditions already uncomfortable and unhealthy.

The first noticeable effect of mechanical ventilation was a decided improvement in comfort conditions. The air temperature in nearly every working place in one mine at Butte in 1916 was above the rock temperature of 78 degrees and the relative humidity was above 90 percent. In 1924, when the mine was ventilated mechanically, the average temperature in 109 working places was 70 degrees, the average relative humidity was 88 percent, and fresh air was found to be present in practically every working place that had been filled with hot, stagnant air having a low oxygen content. In another mine the almost unendurable working conditions were changed between 1916 and 1924 to fairly comfortable conditions; the average temperature was reduced from 83 to 72 degrees and the average relative humidity from 85 to 81 percent. In still another mine the average temperature was decreased from 80 to 66 degrees during the same interval.[139]

In 586 working places in the mines of the Montana area examined by United States Bureau of Mines officials in 1918 it was found that the average air temperature was 81 degrees F. and the average relative humidity 87 percent; a survey in 1923-24 of 676 working places in the 16 mines

[137] McBride, *op. cit.*, p. 86.

[138] D. Harrington, *Metal-Mine Ventilation* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6734, mimeo., June 1933), p. 8.

[139] G. S. Rice and R. R. Sayers, *Review of Safety and Health Conditions in the Mines at Butte* (U. S. Dept. Com., Bur. Mines, Bull. No. 257, 1925), p. 18.

that had adopted improved methods of ventilation revealed that the air temperature was 75 degrees F. and the average relative humidity 81 percent. These average reductions of 6 degrees in temperature and 6 percent in the relative humidity, combined with the movement of fresh air, had in all instances substantially improved the comfort of the miners[140] and thus contributed to an increase in labor efficiency. Since then, as more and better ventilation equipment was installed, further improvements in working conditions were effected.[141]

In Arizona all of the mines that adopted mechanical ventilation reported an increase in comfort conditions resulting in a higher output per man and significant reductions in mining costs. The installation of mechanical ventilation was credited with most of the 50-percent increase in labor productivity and 50-percent decrease in the use of compressed air at the Miami mine.[142] After an efficient mechanical ventilating system was installed at the Copper Queen mine in Bisbee, Arizona, the quantity of compressed air used was reduced by almost 50 percent, the tonnage per man-shift increased from about $2\frac{1}{2}$ to 7 tons, or approximately 180 percent, and the cost of mining ore decreased about 45 percent.[143]

Air conditioning, although still in the experimental stage, has afforded a good standard of working conditions in sections of deep mines in which it would have been extremely difficult to operate with a mechanical system of ventilation. This is indicated in the change in air temperature of working places after air conditioning has been put into operation. For example, in a Butte mine, after a plant was installed in the 3,600-foot level, the average temperature in 14 working places declined from 90 to 77 degrees F. and 88 to 74 degrees wet-bulb.[144] Not only is a considerable part of the heat and moisture in the mine air removed, but a great proportion of the dust is also eliminated. Thus the air in sections of the mine affected by the air-cooling device is not only cooler and less humid but also cleaner and fresher.[145] As a result of this

[140] *Ibid.*, p. 19.

[141] A. S. Richardson, "Ventilation and Dust Prevention in Butte Mines," *Mining Technology*, Vol. 2, No. 5 (Sept. 1938), A.I.M.E. Tech. Pub. No. 969, p. 3.

[142] McElroy, *Ventilation of the Large Copper Mines of Arizona*, p. 108.

[143] C. A. Mitke, *Standardization of Mining Methods* (New York: McGraw-Hill Book Co., Inc., 1919), pp. 41-2.

[144] The comfort conditions of a mine may be measured by an ordinary or dry-bulb thermometer and a wet-bulb thermometer. The dry-bulb thermometer merely records the sensible heat of the air. The wet-bulb thermometer measures the evaporating power of the air. Its bulb is covered with a layer of muslin which is kept moist with water. When the air is not saturated, evaporation will occur; as the water on the bulb is evaporated, heat is absorbed, the thermometer is cooled, and the readings fall below those of the dry bulb. When the air is saturated, no evaporation will take place, and the readings of the dry bulb and the wet bulb will be identical. The katathermometer, which records the cooling effect of the air per unit of time, may also be used to register the comfort conditions in a mine.

[145] Richardson, "Air Conditioning for the Ventilation of Butte Mines," p. 34.

amelioration of working conditions, the productivity of labor has been greatly increased.[146]

Installation of a modern ventilation system not only increases the productivity of labor but also reduces operating expenses. Better ventilation tends to decrease labor turn-over, which is extremely high in poorly ventilated mines, and very costly.[147] Reducing the amount of labor turnover, therefore, has brought about a considerable saving in labor cost. In mines with such unfavorable working conditions as the presence of fog, smoke, and dust, which obscure the vision of the workers, the accident rate is materially higher than that in well-ventilated mines. The elimination of these conditions by proper ventilation has lowered the accident rate and hence decreased human suffering and reduced the amount of compensation paid to the injured and disabled miners.

Improved ventilation has also increased the efficiency of air drills. In poorly ventilated mines that have to resort to the use of compressed air to stimulate air circulation the working pressure is often as low as 30 to 40 pounds per square inch, whereas in well-ventilated mines, where the drills are the only consumers of compressed air, the working pressure is maintained at 60 to 80 pounds per square inch. The increase in air pressure attributable to improvements in mine ventilation has greatly promoted the efficiency of drilling in underground mines. This gain can well be appreciated when it is remembered that drilling is one of the largest items in mining cost. Moreover, where compressed air is no longer needed for ventilation purposes the expense required for compressing the air is decreased. It was estimated that in 30 mines in the Butte area before ventilation was improved 50,000 cubic feet of air per minute was used in compressed-air blowers to obtain relief from bad air conditions.

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[146] At the Magma mine, too, installation of an air-conditioning system in the 3,600-foot level has reduced the air temperature from 102 to 89 degrees and the relative humidity from 74 to 64 percent, and has permitted the miners to work in the lower levels all year around under fairly comfortable conditions (Foraker, *op. cit.*, p. 15).

In most of the uncomfortable places of the Butte mines a good day's work for a mucker consisted of 15 cars of ore or waste, whereas, in cool, well-ventilated places contract men load 45 to 60 cars. Mine officials reported that their men worked not more than half of the shifts and that conditions in many places made it physically impossible for their miners to work steadily and efficiently. The workers in ill-ventilated mines not only give a poor performance but set a standard for the area, so that miners working under conditions conducive to greater productivity refuse to deliver more work than those in less comfortable places. (Harrington, *Underground Ventilation at Butte*, p. 116.)

[147] It is estimated that the minimum cost of hiring and familiarizing a miner with the workings is $10 (Harrington, *Underground Ventilation at Butte*, p. 117).

In sinking a shaft in the Butte area, where the average rock temperature was 100 degrees or over, the average rate was 55 feet per month. After auxiliary ventilation was installed the shaft was sunk at the rate of 130 feet per month. In another mine at Butte, where the temperature was slightly under 80 degrees, it was almost impossible to obtain men to drive a drift about 2,000 feet from air circulation. After the installation of a fan and canvas tubing it became relatively easy to attract and hold workers, and the cost of driving the drift was decreased by over 40 percent. (*Ibid.*, pp. 115-6.)

The cost of compressing 50,000 cubic feet per minute during two 8-hour shifts runs well over $30,000 per month.[148]

An appreciable saving in maintenance labor for timber replacement has also been brought about by effective circulation of air. Warm, damp, stagnant places stimulate fungus growth, causing timber decay which weakens timber structures and may ultimately result in the failure of supports.

SUMMARY AND CONCLUSIONS

The over-all output of copper per underground-mine worker was increasing virtually throughout the period 1917-37. From about 101 pounds in 1917 the output per man-day rose to a maximum of 228 pounds in 1935. The rate of increase, though substantial, was low relative to that at open-cut operations. The trends of the output per worker by four of the five principal mining methods into which all underground mines were classified were rising at even more moderate rates. For the fifth method - shrinkage - the trend was downward since 1921. During this period the proportion of the production coming from mines employing the more productive methods increased, and the output per worker of all underground mining therefore tended to increase at a somewhat faster rate than that by any of the several mining methods.

Increasing natural difficulties on the one hand and advances in mining technology on the other were the main factors affecting the output per worker in underground mines. Progressive exhaustion of the richer, more accessible deposits is indicated by the decrease in the yield of copper and the increase in the depth of mines. Although the full extent of the decline in the quality of the ore was obscured by the fact that the recovery of copper was becoming increasingly more efficient and that selective mining was widely practiced at times, the trend of the average yield of copper for the period 1917-30 was downward, declining at an average annual rate of about 0.3 pound per ton of ore. In Michigan, Montana, and Arizona, where most of the deep vein mines are situated, mine depth has been increasing constantly. Statistics on depth of mines are incomplete, but such fragmentary data as are available indicate the necessity of descending farther and farther underground to get copper. In the past 30 years the average depth of shafts in the vein mines of these three principal producing States has increased notably. The increase in depth, together with depletion (as reflected in the decline in yield), tended to neutralize some of the effects of the improvements in mining methods and

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[148] *Ibid.*, p. 117.

equipment. Had it not been for the fact that natural difficulties were becoming increasingly more adverse, the trends of output per worker for all underground mines as well as for the groups of mines classified by principal mining methods would have risen at much faster rates under the stimulus of technologic improvements.

Technologic advances have been achieved both in the methods of mining and in the equipment used. Perhaps the greatest improvement may be attributed to the invention and development of the caving systems of mining, which have made possible exploitation of the low-grade porphyry deposits on a mass-production basis. Other significant changes in mining methods contributing to technologic progress include utilization of the force of gravity to break and transport the ore to the loading chutes; adoption of the retreating system of stoping in which the ore farthest from the shaft is removed first, thereby reducing haulage maintenance labor; alteration of the arrangement of mine workings to permit the maximum use of scrapers, mechanical loaders, and other labor-saving devices; and adoption of stoping methods which would minimize, if not eliminate, the use of costly artificial supports and the amount of hand shoveling. The improvements in mining methods have contributed materially to offsetting the depressing effect of increasing mining difficulties on the productivity of labor.

A large share of the progress in counteracting the growth of physical handicaps and in raising the output per worker, however, must also be assigned to the technologic advances in mechanical devices.

The introduction of mechanical drills, particularly of the one-man drifter, stoper, and jackhammer types, has greatly increased the speed of drilling. The efficiency of drills has been enhanced further by the constant efforts made to develop special models for particular ground and operating conditions, to improve designs, and to use stronger and more durable materials for the construction of the drilling machines. The wearing qualities of the drill steel and bits have been improved with the use of better-grade steels and the application of special hardening and sharpening processes. Further savings in drill steel have been realized through utilization of detachable bits which permits the bits to be made of special wear-resisting alloys and which eliminates the handling of a large quantity of steel. Blasting techniques also have made great strides, and blasting agents have been improved to such an extent that many types are available having a wide range of strength, size, and characteristics suitable for any condition. Detonators have been improved to eliminate costly misfires and to explode the charge efficiently. As a result of these advances in drilling and blasting there has been a substantial decrease in the labor required to break the ore. It is estimated that about two-thirds of the underground labor was engaged in shattering rocks when hand methods of drilling were used, and that at present only 20 to 30 percent of the total underground labor is employed in drilling and blasting operations.

Mucking or loading the broken ore, too, has been rendered less time-consuming and costly by the advances made in hand-shoveling techniques and by the introduction and continual improvement of mechanical shovels and scrapers. The improved, small mobile loaders of the compressed-air types have been especially important in some of the deep and hot mines where manual work is exceedingly difficult or almost impossible because of the heat and humidity. They have been particularly effective in development work. Although these loaders have been successful in some mines, scrapers have been equally so in others, especially in sublevel development work and in stopes where mechanical shovels cannot be used. The construction of the scrapers and auxiliary equipment has been constantly improved, the designs have been modified from time to time to suit mining conditions, and, where necessary, mining methods have been altered to permit the use of these labor-saving loaders. The effect of utilizing scrapers in underground copper mines, like that of employing mechanical shovels, has been to increase the productivity of labor.

Progress in underground haulage has also been important. Hand tramming and animal hauling have been replaced almost entirely by the electric and battery locomotives, which, in turn, have been vastly improved since their introduction. The modern locomotive, larger and heavier than its predecessors, is equipped with sturdier, speedier, and more powerful motors; dust- and dirt-proof antifriction bearings; better control apparatus; and more efficient brakes. These advances have reduced the cost of maintenance and increased the speed and hauling capacity of the locomotives. The mine cars, too, have been improved. Their dead weight in relation to their pay load has been lowered by changing their design and constructing them with high-tensile-strength steel, their size has been increased, their rolling friction greatly reduced by substituting antifriction bearings for plain bearings, and their design altered to facilitate dumping operations and to suit mining conditions. The haulageways have been enlarged and straightened, track gages widened, heavier rails used, lay-outs of track systems better planned, and block-signal systems and automatic-switching devices introduced to speed the movement of trains. Mining operations have been concentrated on fewer levels wherever conditions permit, and the utilization of scrapers and interconnected chutes to gather the ore and distribute the waste have permitted use of larger, more efficient haulage equipment. Better coordination of the entire transportation system - haulage as well as hoisting - is another important factor.

In hoisting there have been significant improvements in equipment and techniques. The horse whim was displaced by the steam hoist. Despite many improvements, the latter was costly to operate and consequently was replaced by the compressed-air or the electric hoist. The electric hoist, which has undergone a series of improvements, has been gaining in relative importance. The introduction of the Ilgner system and Ward-Leonard control has reduced the problem of "peaks" and permitted the hoist to accelerate and decelerate smoothly and rapidly. The motor has been made larger, more durable, and more powerful. The development of the system of hoisting in balance has resulted in considerable economy in personnel and power. The gradual replacement of the cage by the skip, which has eliminated hoisting the dead weight of the mine cars, has increased the capacity of the hoist 25 to 40 percent. Where cages are required their capacity has been increased by equipping them with multiple decks. In recent years the tendency has been to use light but durable materials for the construction of cages and skips. As a result, the weight of containers has been lowered and capacity raised. Winding cables, too, have been improved by the use of steel of higher tensile strength for their construction. A further gain in efficiency has been attained by concentrating hoisting operations in large, multiple-compartment shafts and by carefully planning the hoisting system so that it will meet the important requirements of each particular mine or group of mines. The construction and design of head frames have been improved and their height increased to accommodate skip hoisting and skip changers and to prevent overwinding. In consequence of these advances, hoisting operations have been carried to progressively lower levels, the capacity of cages and skips has been enlarged, the speed of hoists has been raised, a considerable saving in labor and other costs has been effected, and the productivity of labor not only of the hoistmen but also of all workers employed in underground mines has been increased.

Advances in drainage equipment and technique have also played an important part in overcoming some of the mining difficulties created by the presence of water in the underground workings, in reducing mining costs, and in increasing the labor productivity of not only those engaged in pumping operations but all concerned with underground mining. The pumps have been made more efficient and durable. Their design has been improved, their power consumption has been reduced, their capacities and the head against which they can work have been increased, and their lining and mechanism have been protected against wear and corrosion. The introduction of automatic control in some mines has curtailed station attendance to periodic visits and at the same time has raised the efficiency of mine drainage. By concentrating the water at a few pumping stations and replacing many small pumping units with a smaller number of those that are larger, more efficient, and designed for specific conditions, it has been possible to reduce operating costs and labor requirements. Underground sumps have been enlarged to minimize the amount of emergency equipment and safeguard the mines in case of flooding or breakdown of the pumping equipment.

With the growth in depth of mines, stronger and more elaborate supports are required and a greater expenditure of labor is necessary. Any improvement that simplifies this task or lengthens the life of the supports has, therefore, a significant bearing on the productivity of labor. The use of steel and concrete in place of timber has greatly increased the life of mine supports. Although steel and concrete supports have been gaining acceptance, they are mainly confined to mine openings which have a potential long life. Consequently, timber still predominates as the chief material for supports in copper mines. An important advance in the utilization of timber for mine supports is the standardization of timber sets, a practice that has permitted the framing of timber to be transferred from underground to the surface, where machinery may be utilized. Another significant improvement is the prolongation of the life of mine supports by treating the timber with preservatives, improving the ventilation of the mines, and fireproofing the timber. These changes have reduced progressively the amount of labor required to construct and especially to replace mine supports, notwithstanding the fact that physical difficulties, particularly the depth of mines, have been increasing steadily.

Advances in ventilation facilities and techniques have been another important factor tending to counteract the depressing influence of increasing mining difficulties and to raise the productivity of labor in deep underground mines. Natural ventilation, which had been relied upon from time immemorial to ventilate working places in mines, became insufficient as the temperature and humidity rose with an increase in the depth of operations. Some of the larger mining companies have placed their ventilation problems in the hands of experts who have introduced many improvements. Increasing numbers of large, electrically driven fans have been installed at the shafts, and booster fans have been placed throughout the mine to force the air to the workings which were located away from the main air courses. Small auxiliary blowers, which force the air through galvanized-iron pipes and canvas tubing, have also been utilized wherever necessary to conduct the air to the working face. The air supply is carefully regulated by systematic air splitting, and abandoned sections of the mines are sealed off to prevent the fresh air from escaping into the dead workings. Two mines have been experimenting with air conditioning which has afforded satisfactory working conditions in places where, because of the high temperatures and humidities, successful conduct of manual labor was not possible. The effect of the installation of modern ventilation and air-conditioning systems in deep mines has been an improvement in comfort conditions, a reduction in labor turn-over, a lowering in the accident rate, and an increase in the output per worker.

The gain in productivity ascribable to improvements in the art of mining and progress in mechanization is due in no small measure to scientific planning - a topic that has been mentioned in passing in the preceding sections. After a mining method has been selected to suit individual conditions, its application must be so planned as to attain the lowest possible unit costs. To this end scientific-management principles are applied to standardize the work of the miner. In the early days the miner planned and executed his work to the best of his own knowledge and ability. With more rigorous engineering control, however, his work is laid out for him by the management, which employs a staff of experts to investigate, devise plans, and supervise operations in accordance with those plans. The number and placing of drill holes, the quantity of powder to use in blasting, the spacing and framing of timber, and the kind of equipment to be used in a given situation are all prescribed for the miner to increase his output per unit of time. Similarly, all operations beyond the stope (hauling, hoisting) are planned to balance the maximum output expected from the miners in the stopes. A well organized mine today is not unlike a modern factory in which the task of each worker is set and the parts are so coordinated as to operate as a whole.

This, then, summarizes the changes in mining conditions on the one hand and advances in the mining methods and in mechanization, aided by scientific management, on the other. Because these changes have occurred simultaneously it is not possible to determine statistically the precise effect of each on the productivity of labor. However, the net result of the struggle between increasing physical handicaps and technologic progress has been a substantial and rather steady rise in the over-all output per worker.

The increase in output per worker would undoubtedly have been much greater had all mines adopted the improved engineering practices as soon as they became known. It is not always possible, however, for all operators to make use of the latest inventions economically. Where a mine has been in operation for some time the adoption of the most modern equipment may be conditioned by a number of factors. Among the more important of these are the outlays already made for existing equipment that is still serviceable, the size of mine openings, the mining methods employed, and the life of the ore reserve. For example, the introduction of the larger and improved locomotives and mine cars would call for the widening of track gages, enlarging the haulageways, and discarding some of the old equipment. Having already invested a large amount of capital in developing and equipping a mine, an operator naturally would be reluctant to install the latest equipment, which would involve a considerable outlay. Unless he can be satisfied that the new expenditure is a profitable investment, the new equipment will not be introduced. Numerous factors contrive to cast doubt on the wisdom of making costly commitments for the future; among the most important of these are declining prices that accompany recurring depression. Technologic changes over an entire industry are therefore more likely to be made piecemeal than all at once, and often take the form of modified old equipment rather than new installations. For this reason there is usually a time lag between the first development of a new piece of equipment and its general acceptance. This failure of industrial practice to keep abreast of engineering progress is universal, but it is especially true of the mining industry where risks are likely to be particularly great.

In view of the lag between the first development of a mining device and its general adoption, further gain in labor productivity may be expected as the use of existing inventions becomes more widespread. And a greater increase may be anticipated as new inventions and better techniques are developed. In the field of drilling and blasting progress will probably continue along its former paths: the effective life of the drills will be lengthened, their drilling speed increased, their power consumption reduced, and their mobility improved; detachable bits will be more widely adopted; and drilling and blasting operations will be more carefully planned and supervised. In the realm of mucking there will be more improvements in mechanical shovels and scrapers, a wider application of these devices, and greater efforts to utilize gravity to load the ore. Future advances in underground haulage will consist mainly of further improvement and standardization of the equipment and better coordination of the haulage system; both locomotives and mine cars will be made larger, more efficient, and more durable; the small storage-battery locomotives will continue to replace hand tramming wherever conditions permit; and belt conveyors may possibly be introduced in some mines. There are also prospects for further gains in efficiency in hoisting, notwithstanding the increase in the depth of mines. More powerful and higher-speed motors will probably be developed; the weight of the cage or skip will be reduced by the use of lightweight alloys and that of the cable either by tapering or by the use of higher-tensile-strength steels. Further advances in mine drainage may be expected when more mine operators begin to modernize their pumping equipment by installing more efficient pumps and resorting to automatic control. There will also be some improvement in mine supports. More steel and concrete will be used in permanent mine openings, and greater efforts will be made to prolong the life of mine timber by chemical treatment and other means. Finally, there will be a further amelioration of working conditions despite the growth in mining depth. More and better mechanical ventilation systems will be installed, and in the deeper and hotter mines air conditioning will be practiced.

CHAPTER V

ORE DRESSING[1]

Changes in the techniques of ore dressing have had a profound influence on mining practices and an important bearing on the output of copper per worker. As had already been pointed out, advances in the technology of ore dressing have been largely responsible for profitable exploitation of low-grade porphyry ores. As these ores have been extracted by application of the more productive methods - open-cut and block-caving - the output per worker, as a consequence, has been materially increased. Improvements in the method of ore dressing, moreover, have not only an indirect but also a direct effect on the output per worker; with increased efficiency in the recovery of copper from the ore, a greater quantity of metal is credited to the labor expended in mining a ton of ore. Thus progress in ore-dressing technique has enhanced mining productivity indirectly by permitting more productive methods to be applied to win copper from low-grade ore and directly by increasing the recovery of metal from the ore mined. It is for this reason that a brief discussion of ore dressing is appropriate here even though the employment afforded by ore dressing is not included in the basic data of this report.

Essentially, ore dressing consists of eliminating a portion of the waste material contained in the ore, thus increasing the metal content of the product to be sent to the smelter. In the past and, to some extent, even at present, partial concentration of ore has been accomplished by hand sorting at the mine. In this process, which is applicable to vein mines, the ore is broken into fragments small enough for hand sorting, and those high in metal content are selected for smelting. There always remain, however, considerable quantities of ore too low in copper content for direct smelting but containing, nevertheless, many valuable particles of metal. These particles usually are disseminated throughout a fragment of ore. This fragment must be finely crushed before the particles can be separated and removed from the waste material or gangue. With the mining of large tonnages of low-grade prophyry ores by mass methods, hand sorting could not be employed and mechanical methods of treating ore have become necessary.

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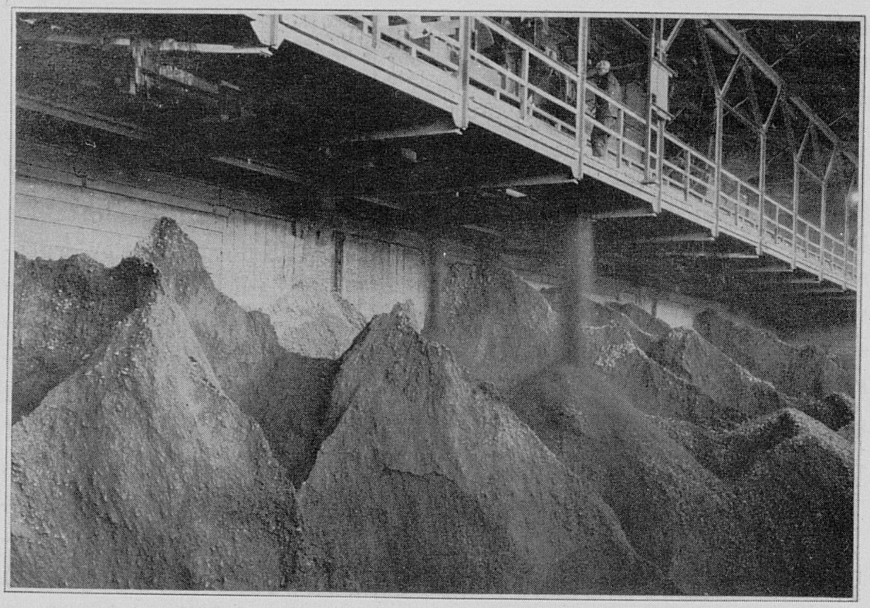
[1]By Emil Erdreich, Y. S. Leong, C. E. Nighman, and J. C. Burritt. No attempt is made here to treat the subject of ore dressing exhaustively. For a more detailed treatment of the techniques of ore dressing see T. G. Chapman, *Concentration of Copper Ores in North America* (U. S. Dept. Int., Bur. Mines, Bull. No. 392, 1936); A. F. Taggart, *Handbook of Ore Dressing* (New York: John Wiley & Sons, Inc., 1927); *Transactions of the American Institute of Mining and Metallurgical Engineers*, Vol. 112 (1935); and A. M. Gaudin, *Flotation* (New York: McGraw-Hill Book Co., Inc., 1932).



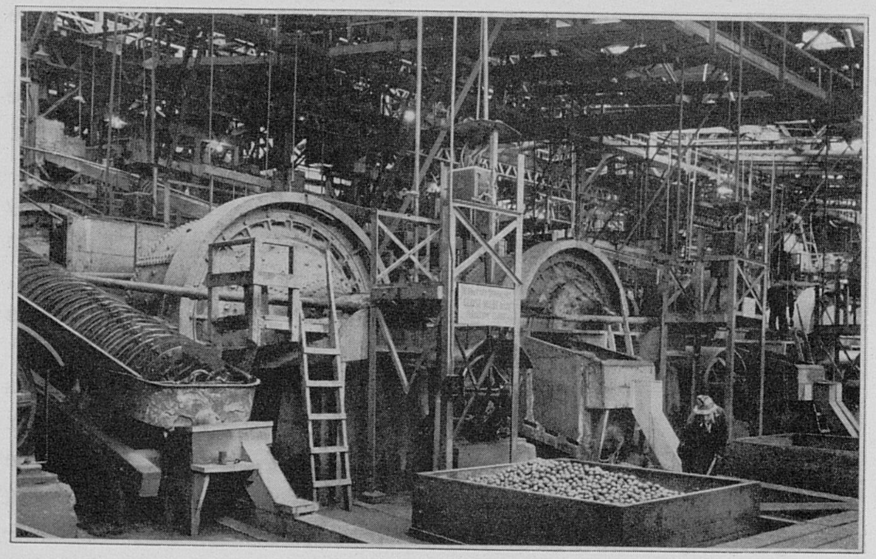


FIGURE 41.— MODERN CONCENTRATION METHODS REQUIRE THAT ORE BE CRUSHED AND GROUND UNTIL THE LARGEST PARTICLES RESEMBLE FINE SAND

The large chunks of ore are first crushed in coarse crushers and this coarsely crushed ore is fed to secondary crushers. It is then stored in bins, as shown in the upper view. Later the ore is reduced further by fine-grinding units such as the ball mills in the lower view.

After the ore is extracted from the mine it must undergo a series of treatments before the valuable mineral can be recovered. The method of treatment depends on the proportion of the mineral in the ore and the chemical state in which it is found. High-grade ores (containing at least 3 to 4 percent copper) usually are smelted without preliminary treatment. Some low-grade ores are treated by "leaching", that is, by dissolution of the metal in some solvent that does not dissolve the gangue and recovery of the metal from the solution either by precipitating it on scrap iron or by electrolysis. Most copper ores, however, must pass through several treatments, in which the ore is separated into a "concentrate" containing the bulk of the metal and "tailings" consisting of worthless material with but a small residue of valuable mineral. The concentrated product usually contains 30 to 60 percent copper. The concentrate is next sent to a smelter where it is transformed into blister copper, which is finally converted into refined copper at a refinery.

Concentration greatly reduces the volume of material that must be treated by expensive metallurgical processes. For instance, 100 tons of porphyry ores assaying about 1 percent copper will yield about 3 tons of concentrate containing 30 percent copper, the loss in tailings usually being less than one-tenth of the total copper. Thus, instead of treating 100 tons of original ore, only 3 tons of concentrate need to pass through the smelting process.

Labor requirements in ore dressing are much smaller than in mining operations. In the last decade about 32 tons, on the average, were treated per man-day of labor expended in ore-dressing operations, whereas the output per worker at the mines averaged, during the same period, about 6 tons per day. These figures indicate that the labor requirements in ore-dressing operations are even lower than at the open-cut mines, where the output per man-day averaged about 20 tons.

Mechanical treatment of ore can be divided broadly into two major operations: milling[2] (reduction in the size of ore particles by means of crushing and grinding, and classification of these particles by size and specific gravity) and concentration (increasing the metal content of the product by eliminating waste material or gangue).

Crushing and grinding are the most expensive part of dressing most ores, for the valuable minerals are almost invariably disseminated in the ore to such an extent that the grinding must be carried to a fraction of a millimeter[3] or a few thousandths of an inch to free metallic from waste

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[2]The term "milling" is used by some writers synonymously with "concentration" or "ore dressing."

[3]A millimeter is equal to 0.03937 inch.

particles. Crushing usually proceeds in steps; all particles which are already as fine as the final product of a crusher generally are removed from the feed of that machine. The ease with which the ore is ground to any particular size depends largely on the type of ore. Some of the crushing and grinding machines have a screen to assure that the product will not be larger than a specified size. The coarse crushers, which receive only the larger fragments of ore, can reduce lumps as large as 5 feet in diameter to a product 1-1$\frac{1}{2}$ inches in size. Most of the ore that comes from an underground mine is less than 1 foot in diameter, whereas that from an open-pit mine is much larger. The size of the feed to intermediate crushers, which include rolls, steam stamps, and disk and cone crushers, ranges from 6 to 1 inches; the final product ranges from $\frac{1}{2}$ to $\frac{1}{16}$ inch. Fine crushers or grinders receive the product from the intermediate crushers and reduce it to any desired size down to about 0.05 millimeter. Fine grinding usually requires wetting of the product in contrast to dry crushing in the coarse and intermediate crushers. Of the many types of machines that have been used in intermediate and fine grinding the most important are steam stamps, rolls, ball mills, rod mills, and tube mills. Final separation of the crushed material involves sizing and classification by a series of screens and classifiers to yield groups of graded products suitable for various types of concentrating equipment. Screens usually are employed for separating products above 1 to 2 millimeters and classifiers for those below 1 to 2 millimeters in size.

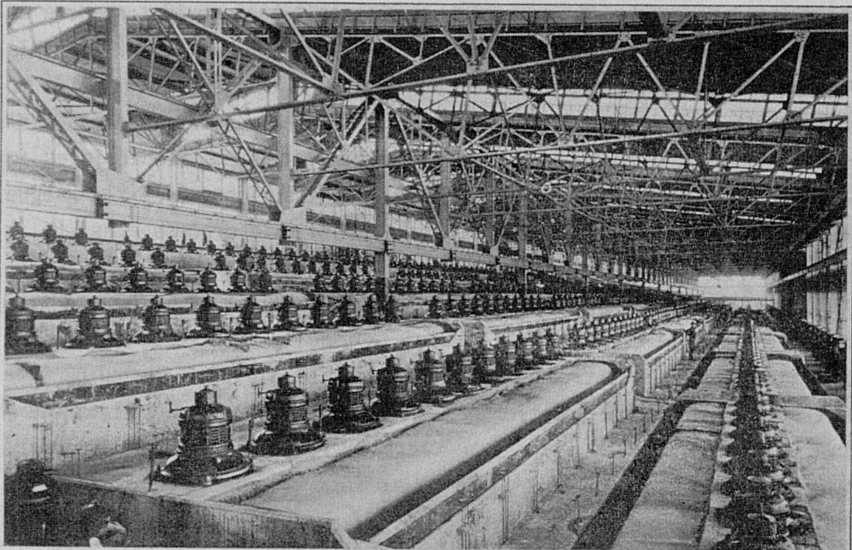
In the process of concentration the ground product usually is subjected to several operations that utilize differences in physical properties of the mineral and the gangue. The two methods employed most commonly are gravity concentration, which utilizes the differences in specific gravity, and flotation, which is based upon such properties as surface tension, adhesion, and absorption.

The process of separating the valuable mineral from waste material by gravity concentration is based upon the principle that when two particles of different specific gravity are allowed to fall through a resisting fluid the metallic particles, having a higher specific gravity, develop a faster falling rate than those containing primarily waste material, of relatively low specific gravity. Numerous devices are employed in gravity concentration. With water as a resistive medium the machines employed most commonly are mechanical and hydraulic classifiers, jigs, shaking tables, vanners, and slime tables. The last two, however, are used much less since the development of the flotation process.

Froth flotation, which was developed in 1912 and is the only form of the flotation process of commercial importance,[4] has proved to be one of the most outstanding achievements in ore-dressing technology. In this process finely ground ore is mixed with water, 4 to 13 parts of water to 1 of ore (by volume), to form a pulp to which reagents are added. Mineral is separated from the gangue by causing the mineral particles to attach themselves to air bubbles rising to the surface while the gangue remains submerged. One of the first steps in this process consists in adding to the pulp a "frothing" agent which assists the formation of bubbles and strengthens the surface of these bubbles by forming a film around them. The evolution of bubbles is usually induced by mechanical agitation or by the introduction of a stream of air into the pulp. The next step involves the addition of a "collecting" agent (usually a chemical compound, such as one of the xanthates) to the mixture of ore and water. The function of this agent is to form an oil-seeking coating on the surface of mineral particles which, as a result, become less easily wetted by the water of the pulp. This coating increases the force of adhesion between mineral particles and a bubble surface. Frequently the same substance may serve the function of frothing as well as that of collecting agent. The bubbles plus the attached mineral particles rise to the surface in the form of mineralized froth, which is then collected as concentrate. In some instances it is necessary to add "depressing agents" to lessen the floatability of one or more of the minerals when they occur with the mineral to be separated. For instance, copper sulphides often are mixed with pyrite, and to isolate the former it is necessary to employ a depressing agent to get rid of the latter.

The froth-flotation method is effective in separating mineral particles less than 0.5 millimeter in diameter, much of which would have been lost in the slime under gravity methods. Moreover, it permits separation of minerals other than copper which have nearly the same specific gravity. The development of this process, which is generally known as selective or differential flotation, is one of the most significant advances in metallurgical technology. It has made it possible to recover, by successive application of different reagents, the greater portion of accessory minerals which were formerly lost. Furthermore, it has permitted the treatment of complex ores, that is, ores containing sizeable quantities of more than one metallic mineral. Until the advent of flotation these ores could not be treated effectively, for their specific gravities do not

[4] There are two other varieties of flotation process: film flotation and oil-buoyancy flotation. For description of these varieties see Taggart, *op. cit.*, pp. 779-90.



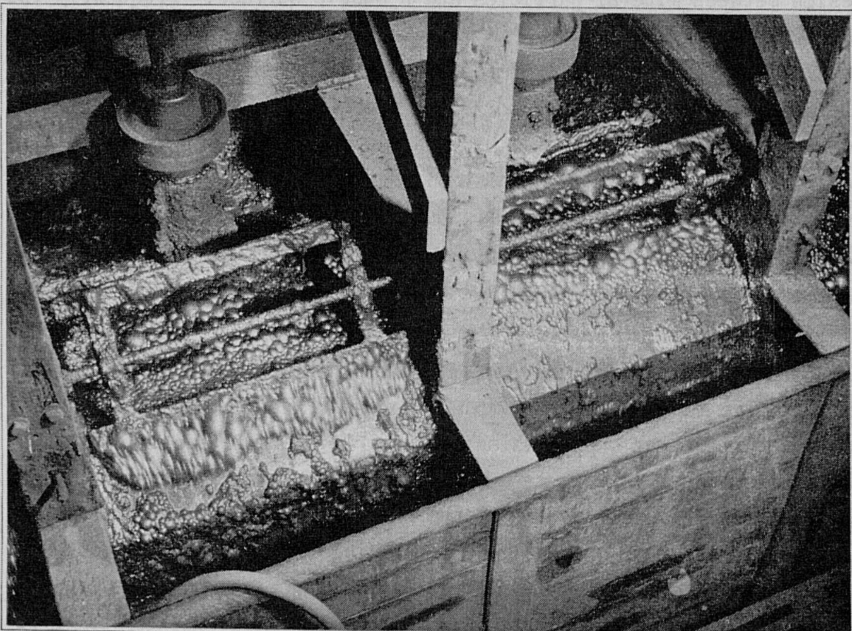


FIGURE 42.- TYPICAL LARGE COPPER CONCENTRATING MILLS USE FLOTATION CELLS

The upper view shows a battery of flotation cells in a large copper mill; the lower picture is a closer view of a different type of flotation cell. In these units the lighter waste material sinks to the bottom where it can be diverted as waste. The heavier metal particles, because of their affinity for flotation agents, attach themselves to bubbles that are produced by the agitation and aeration of the pulp and ride or float to the surface where they are skimmed off as concentrates.

differ to any large extent and separation could not have been accomplished by gravity concentration. Thus the flotation process brought into exploitation many deposits that formerly could not be mined commercially, thereby increasing the available copper reserves. It has not only enhanced the efficiency in the recovery of copper and of other metals from copper-bearing ores but has also greatly reduced the cost of smelting per unit of copper metal by freeing copper concentrates from undesirable constitutents.[5]

Generally ore-dressing practices vary with the type of ore which is to be treated. Sulphide ores (those that carry the copper as a sulphide, commonly chalcocite, bornite, or chalcopyrite) are crushed and ground down to $\frac{1}{4}$ millimeter or finer by the use of ball and rod mills, and the product is then treated by flotation.[6] If the ore occurs in the form of oxides it is usually treated by leaching, although in some instances gravity concentration and flotation are also employed. Complex ores containing sulphides of two or more metals generally are treated, after reduction to a fine product, by differential flotation which separates the metals. Some concentrators have adopted the practice of grinding the ore rather coarsely and treating the ground material by bulk flotation and then regrinding and treating the concentrate by selective methods.[7] This procedure eliminates the necessity of fine grinding of all the ore and yet retains the advantages of high recovery and high copper grade of concentrate. The conglomerate copper ores of the Lake Superior district, containing native copper, are first reduced to about $\frac{3}{8}$ inch in size. The coarser products are treated in jigs and the finer ones by means of shaking tables.[8] The coarser material derived from the last operation may be reground and retabled. The flotation process has also been used in the Lake Superior region to treat slimes and tailings.[9]

The significance of technologic advances in ore dressing is indicated by the increased efficiency in the recovery of copper at the three typical mills for which a long-time record is available (see figure 43). In

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[5] See Chapman, *op. cit.*, pp. 14-5.

[6] Because of unusual conditions at the plants, a few concentrators still treat sulphide copper ores by gravity methods.

[7] The Miami Copper mill, for example, has been regrinding and treating its primary concentrate by selective flotation since 1925. See H. D. Hunt, *Mining Methods and Costs at the Concentrator of the Miami Copper Co., Miami, Ariz.* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6573, mimeo., Aug. 1932), p. 8.

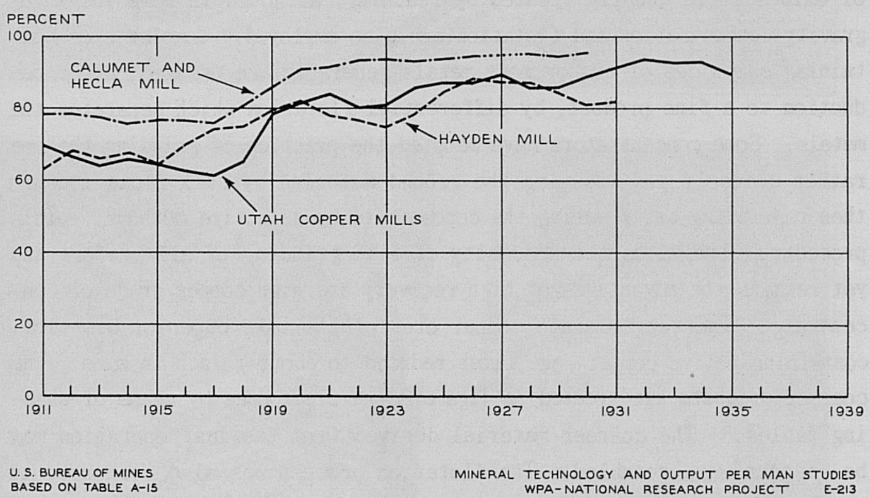
[8] The prevalence of jigs and tables in the Lake Superior region, despite their abandonment in favor of all flotation elsewhere, is attributable to the unique characteristics of the ore - the presence of native copper which resists reduction in size when liberated and the presence of particles of native copper which are tougher than the ore and hence more resistant to sliming. See C. H. Benedict, *Milling Method and Cost at the Conglomerate Mill of the Calumet & Hecla Consolidated Copper Co.* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6364, mimeo., Nov. 1930), pp. 3-4.

[9] *Ibid.*, p. 9.

all three mills the greatest gains were registered since the introduction of the flotation process, although improvements in fine grinding and gravity concentration were also responsible for increasing the metallurgical efficiency of these plants.

At the Utah Copper Company concentrators (Arthur and Magna Mills) flotation was first introduced in 1914 but did not come into full operation until May 1918. The improvements in recovery between 1917 and 1919 can therefore be attributed to the adoption of the flotation process. Since 1919, however, part of the increase in the efficiency of copper recovery was due to better classification, finer grinding, and improvement in the character of ore mined.[10]

Figure 43.- PERCENTAGE OF COPPER CONTENT OF ORE RECOVERED AT TYPICAL CONCENTRATORS, 1911-35



At the Hayden mill of the Nevada Consolidated Copper Company the introduction of flotation in 1915 for the treatment of gravity-mill tailings increased copper recovery in subsequent years to about 75 percent as compared with less than 70 percent prior to the change. Between 1918 and 1924 the improvement in recovery was largely attributable to finer grinding and the use of more efficient classifiers and better reagents. With the construction of new flotation units in 1925 and the consequent elimination of shaking tables mill recovery ranged from 87 to 88 percent in the years 1926 to 1928.

The Calumet and Hecla Concentrator in Michigan shows an improvement in recovery from 78 percent in 1911 to about 91 in 1920 and to about 96

[10] H. S. Martin, *Milling Methods and Costs at the Arthur and Magna Concentrators of the Utah Copper Co.* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6479, mimeo., July 1931), p. 8.

in 1930. The gains in the efficiency in the first instance were due to three causes: finer grinding, adoption of the leaching process for treating fine sands, and introduction of flotation for treating slimes. Since 1919 no major changes have been made in the treatment of the ores, and the increased recovery is to be attributed to general improvements in metallurgical efficiency.[11]

Rather remarkable gains in mill recovery of copper were achieved through many improvements in crushing and concentration techniques. Within the period 1911-35, for which the recovery figures are plotted in figure 43, important changes have taken place both in the type of machinery employed and the processes used. Among notable advances in equipment is the introduction of the Symons cone crusher for intermediate crushing. One advantage that this type of machine affords is the greater capacity in terms of tons crushed per unit of time and the ability to reduce lumps as large as 8 inches in diameter to less than ½ inch in one operation. Pebble mills, which had eliminated Chilean and Huntington mills, in turn were replaced by ball and rod mills. These latter mills, which are used almost universally now for fine grinding, have increased speed and capacity of grinding circuits and lowered the cost of grinding. The adoption of improved hydraulic and mechanical classifiers has made sizing of the product more efficient and accurate and has resulted in an increase in the recovery of copper in concentration. Gravity concentration machines have been replaced almost completely by flotation equipment except in the Lake Superior district where, owing to the peculiar characteristics of the ore, jigs and shaking tables are retained. The introduction of the flotation process has not only revolutionized the method of ore treatment but has also stimulated most of the advances in crushing and grinding practices.

The use of gravity methods, as practiced before flotation was adopted, has resulted in extremely high losses of copper in tailings. Various attempts were made to overcome the defects of gravity concentration. The Anaconda Copper Company, after considerable experimentation, utilized the round table in 1914 to recover the fine particles of mineral in the slime products, and succeeded in reducing the copper losses in tailings by about 50 percent.[12] The Miami Copper Company resorted to regrinding the coarser sand-table tailings and treating the fine slime material on multiple-deck tilting slime tables.[13] Although the recovery of fine free

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11 Benedict, *op. cit.*, pp. 4-5.

12 F. Laist and A. E. Wiggin, "The Slime-Concentrating Plant at Anaconda," *Transactions of the American Institute of Mining Engineers*, Vol. XLIX (1915), pp. 473-9.

13 Hunt, *op. cit.*, p. 2.

mineral in the tailings was improved by these processes, the losses in copper still were high when flotation came on the scene; flotation was therefore first adopted primarily to reduce the losses of copper in the tailings and slimes of gravity concentrators. This innovation was instantly successful. The metallurgical results were so much improved by flotation at the Anaconda Copper Company that by 1915 it was decided to replace the round tables with the flotation process.[14] With the improvements in milling methods, which permitted finer grinding and increased the volume of fine product, flotation gradually became a major process, and gravity was relegated to the position of an adjunct to flotation. Between 1923 and 1927 "all-flotation" methods were introduced at most copper concentrators and have almost entirely displaced gravity concentration in treatment of sulphide ores. A few mills, however, still retain gravity methods for reasons peculiar to each plant.[15]

Flotation machines have been much improved since their adoption by the copper-mining industry.[16] The earlier machines used by copper concentrators were of two general types: straight mechanical-agitation and pneumatic. The mechanical-agitation type has been replaced by the subaeration and mechanical-air types, with a resulting saving in the consumption of power and an improvement in metallurgical results. The original pneumatic machine was equipped with a stationary mat having a canvas bottom through which air is blown by a compressor. With the successful depression of the pyrite and the development of selective flotation, operating difficulties were encountered with this type of machine. Where the ore contained a considerable amount of pyrite, the larger amount of lime required to depress the iron increased the deposition of lime salts in the pores of the mats, and the depression of the pyrite resulted in a heavy accumulation of waste material on the mat. As a consequence aeration was reduced, consumption of power was high, and repair cost was increased. To overcome these defects the stationary-mat machine was modified to the pneumatic rotating mat or matless types. Although the old mechanical-agitation and stationary-mat types are still in use at some copper concentrators, the trend in recent years has been toward subaeration and matless pneumatic machines.[17]

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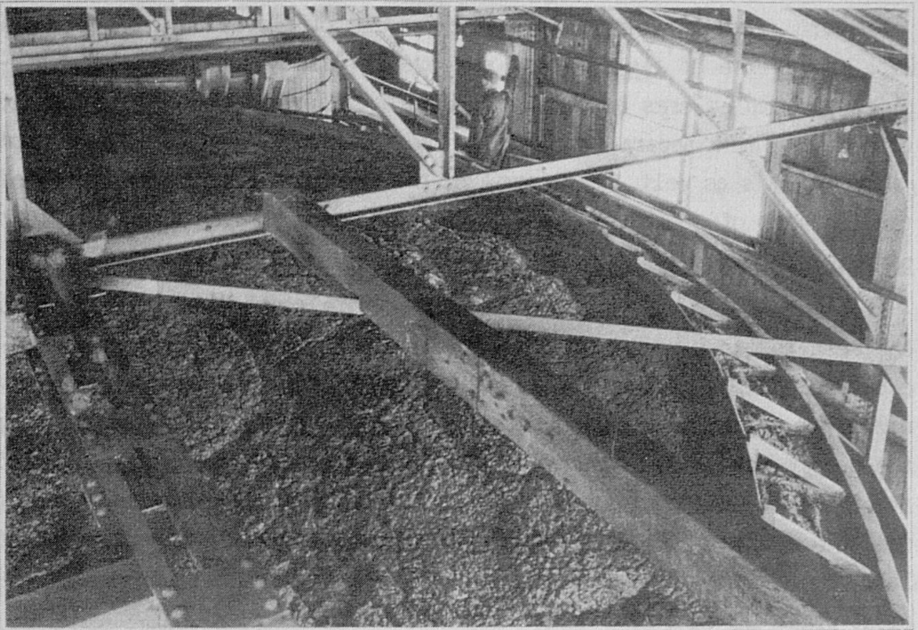
[14] The Anaconda company had been experimenting with flotation since 1913. In February 1915 it signed a contract with Minerals Separation agreeing to pay royalty to the patent-owning company for the privilege of using the flotation process. At the close of 1915 the Anaconda company reported that as a result of the use of flotation an increase of 55,000,000 pounds per year in production was obtained without an increase in the volume or grade of ore that had been mined in the past. Of this increase, approximately 40,000,000 pounds could be realized without adding to the cost per ton of ore treated. See T. A. Rickard's editorial in the *Mining and Scientific Press*, Vol. 112 (Feb. 26, 1916), p. 296.

[15] Chapman, *op. cit.*, p. 5.

[16] For a description of flotation machines see Gaudin, *op. cit.*, pp. 387-401.

[17] *Ibid.*, p. 401. See also Chapman, *op. cit.*, pp. 78-9, 97.



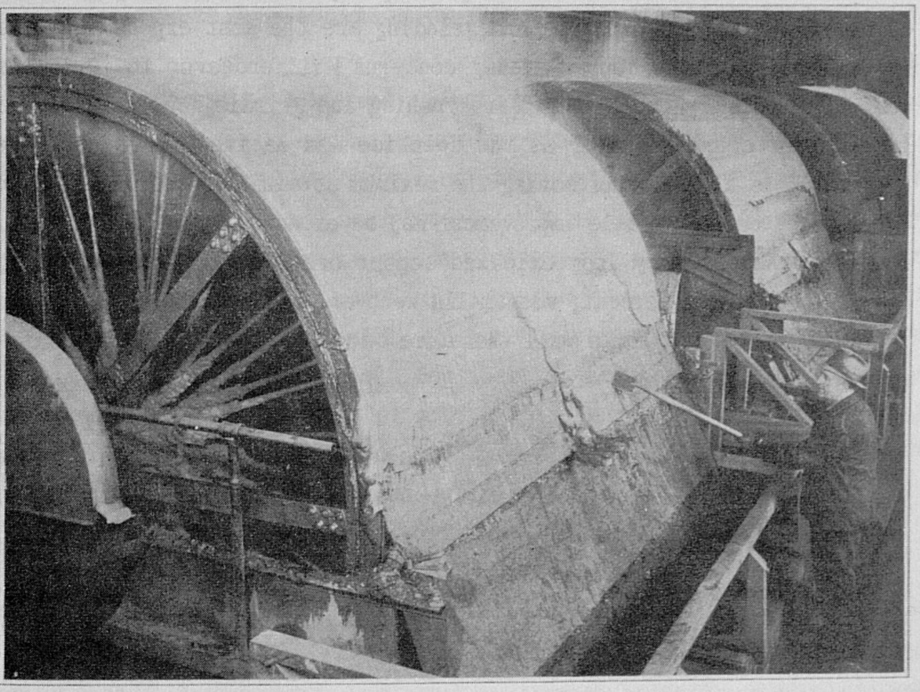


FIGURE 44.- CONCENTRATES MUST BE DEHYDRATED

After flotation, concentrates containing about 25 percent solid matter and 75 percent water are thickened in mechanical thickening tanks, shown in the upper view, to constitute about 65 percent solids and 35 percent water. The thickened concentrate is then conveyed to continuous mechanical vacuum-pressure filters, shown in the lower view, where the moisture content is reduced to 9 to 12 percent. The "cake", removed from the filter by scrapers, falls into a trough and is carried away by conveyor belts.

The most recent developments in the process of flotation are concerned with the choice of the reagents employed. Selective flotation as applied to complex ores has been possible because of the use of suitable reagents. These separate copper not only from the gangue but also from other minerals as well. The growth of the process has not only increased the recovery of copper but has also succeeded in salvaging the greater part of other metals which were previously lost in tailings.[18]

In addition to the increased recovery of metals, the flotation process has resulted in a reduction of milling and concentration costs, an increase in plant capacities, and a simplification of mining operations by making it less necessary to resort to selective mining and permitting the profitable exploitation of porphyry and complex ores. Attainment of these objectives has been facilitated by general improvements in plant design and construction of ore-dressing plants, such as installation of larger operating units, automatic control of machinery, and more efficient plant lay-out.

Future efforts will be directed principally toward reducing the cost of ore dressing. As crushing and grinding are the most expensive items in ore dressing, the copper-mining concerns will endeavor to lower the cost of milling by improving their crushing and grinding equipment. The recovery of copper, as far as the sulphide and native-copper ores are concerned, is already approaching the maximum attainable, being well over 90 percent, so that little improvement may be expected in this direction. However, the recovery from oxidized copper ores is still low, ranging only from 15 to 60 percent; most mills recovered less than 40 percent in 1929 and 1930.[19] Attempts will therefore be made to better the metallurgical results from these ores.

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[18] The recovery of molybdenum in the copper ore by the concentrators of Utah Copper is a case in point. With heads averaging only 0.04 percent, the production of a high-grade molybdenite concentrate low in copper is proving to be profitable. See D. D. Moffat, "Mining Utah Copper," *The Mines Magazine*, Vol. XXVII, No. 11 (Nov. 1937), p. 53.

[19] Chapman, *op. cit.*, p. 13.

CHAPTER VI

SOME EFFECTS OF TECHNOLOGIC CHANGES ON THE MINERS[1]

Widespread mechanization and the resultant gain in the productivity of labor on the one hand and the increase in physical difficulties on the other have affected employment opportunities, modified skill requirements, and multiplied occupational hazards. These changes, moreover, have had other repercussions: on the wages earned and hours worked by miners, the security and duration of their employment, their safety and health, the legislation enacted for their protection, and their relations with the operators. Although, as a rule, the miners have scored important gains through improvements in working conditions and more recently through legal recognition of their right to self-organization and collective bargaining, they have, since the end of the World War, sustained a serious loss of opportunities for employment and experienced considerable economic insecurity.

NUMBER AND LOCATION OF EMPLOYMENT OPPORTUNITIES

Until the end of the World War the number of men engaged in the mining of copper ore had been increasing almost continually, reaching a peak of 61,300 men in 1917. Since then employment opportunities have been declining. By 1929, when copper production for the second time reached a peak of almost 2,000,000,000 pounds (table 5), only 37,100 workers were required. The 1,200,000,000 pounds produced in 1936 required only 14,100 men as against the 35,300 who were employed in 1920 when output was at the same level.[2] It may safely be said that advances in technology were to a large extent responsible for these differences in employment opportunities at given levels of production. In addition to productivity change, sharp contractions of production have periodically diminished employment opportunities greatly. Thus output in 1921 was less than one-quarter of the 1916 peak. In 1933 production was less than one-fifth of the 1929 level, and even in 1936 it was only 61 percent of 1929.

Important changes have taken place in the location of jobs. Whereas in the earlier days of the industry (prior to 1880) Michigan provided by far the greater portion of total employment in copper mining, in recent years it has accounted for about 15 percent, and there are definite

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[1]By Emil Erdreich and Y. S. Leong.

[2]Data on numbers of men given in this paragraph are from table 5 and are not adjusted for differences in number of days worked in the various years. If adjustment is made on the basis of 300-day man-years, the number of workers would be 63,600 in 1917, 39,900 in 1929, 14,500 in 1936, and 37,300 in 1920.

Table 5.- PRODUCTION, EMPLOYMENT, MAN-SHIFTS, AND UNIT LABOR REQUIREMENTS IN COPPER MINING, 1914-36[a]

Year	Copper production (thousands of pounds)	Number of men employed	Number of man-shifts worked (thousands)	Number of man-shifts per pound of copper produced[b]
1914	1,127,259	44,686	12,845	0.011395
1915	1,453,912	47,174	14,211	.009774
1916	1,969,403	61,228	18,932	.009613
1917	1,866,079	61,275	19,086	.010228
1918	1,884,055	59,447	19,104	.010140
1919	1,191,292	39,327	11,856	.009952
1920	1,201,687	35,254	11,182	.009305
1921	455,708	18,300	4,461	.009789
1922	947,299	25,739	7,505	.007923
1923	1,449,780	32,477	10,306	.007109
1924	1,585,020	32,477	10,228	.006453
1925	1,650,291	33,266	10,421	.006315
1926	1,690,043	32,723	10,512	.006220
1927	1,615,928	30,724	9,625	.005956
1928	1,774,120	30,561	9,901	.005581
1929	1,961,560	37,147	11,984	.006109
1930	1,385,169	27,692	8,250	.005956
1931	1,042,531	19,687	5,076	.004869
1932	464,857	9,555	2,291	.004928
1933	368,225	6,976	1,690	.004590
1934	457,646	8,084	1,841	.004023
1935	737,778	10,188	2,787	.003778
1936	1,203,202	14,102	4,355	.003620

[a]Based on table A-1.
[b]Number of man-shifts divided by copper production.

indications of a further decline in the near future because of rapid depletion of available reserves in that area. The importance of Montana as an employment center is also waning. In the nineties Montana displaced Michigan as the chief source of employment in the mining of copper ore, but during the last two decades its importance has declined considerably.

The decrease of employment opportunities in Michigan and Montana has been compensated to some extent by an increase in the number of jobs in Arizona, Nevada, and Utah. The development of ore bodies in the latter States has been made possible by technologic advances that have permitted the exploitation of porphyry ores that chiefly occur in these States. Thus the general decrease in the number of available jobs has been accompanied by a shift in employment opportunities from the North to the Southwest.

The effect of increasing depth and gradual depletion on a mining community may be observed from a study of the Lake Superior district covering Houghton and Keweenaw Counties, Michigan. With the rapid expansion of mining operations the population of these counties increased from 27,000 in 1880 to 95,000 in 1910. Meanwhile the richer and more accessible ores were gradually worked out, the mines were getting constantly deeper, the veins were decreasing in width, and competitive mining districts were opened up in the West where conditions were more favorable and costs lower. The output of copper from Michigan ore began to shrink from a war peak of 268,000,000 pounds in 1916 to 153,000,000 pounds in 1929. Employment, too, fell from 15,000 workers in 1916 to 6,800 in 1929 and to about 1,900 in 1936; but for the fact that technology had enabled operators to recover copper from old tailings, employment would have declined further. The population of the two counties declined from 95,000 in 1910 to 58,000 in 1930. It is estimated that their 1936 population was about 50,000. Had it not been for the depression, more inhabitants probably would have migrated. As it is, there were approximately 3,000 copper miners, together with their families representing about 12,000 persons, left stranded near the mines with little prospect for reemployment.[3]

SKILL AND AGE REQUIREMENTS

Widespread mechanization and large-scale operations have affected both the skill and age requirements of miners. In the earlier days a good miner had to be a "jack-of-all-trades", able to handle nearly any task encountered in mining. Of course, unskilled labor was also employed on a large scale for digging and hand shoveling. With the development of mass production and division of labor such universality in skill was no longer required. Men are now trained for specific tasks, and mining is carried on under the immediate supervision of experienced foremen in accordance with plans prepared by technically trained mine superintendents and mining engineers. In drilling, for instance, the Cornishman, who typified the driller in the early days, was an artist in his trade. With the introduction of many drills devised for specific conditions, less dependence was placed on skills of individual drillers and more on the selection of proper drills and explosives. Similar situations exist in other operations. In framing timber most of the work is done in surface shops where machines produce standardized timber sets with interchangeable parts. As

[3]C. Goodrich and Others, *Migration and Economic Opportunity* (Philadelphia, Pa.: University of Pennsylvania Press, 1936), pp. 186-7, 272-3, 434-5.

a result much less skill is required of timbermen today than formerly, when each timber part had to be finished by hand in the mine and when each frame might be different from the others.

On the other hand, mechanization has increased the demand for skilled mechanics, enginemen, and technicians and has eliminated a large number of totally unskilled laborers, especially since electricity was introduced in mines. As animal haulage has given way in most mines to electric transportation and as scraping and loading operations have become more and more mechanized the old skills have become virtually obsolete. Younger men, trained in the use of particular machines, have come to replace the older, jack-of-all-trades, hand miners.

Although records showing the age distribution of miners are not available, there are indications that at most mines younger men have been preferred to older ones in recent years. Before the advent of machines, knowledge of mining and skill in performing a multiplicity of tasks were indispensable. Long years of training were essential. Consequently, older men with long, broad experience usually were better miners than the younger men. In these days of specialization, on the other hand, a miner can be trained to do his task efficiently within a relatively short time. His work is confined to a narrow field wherein he can readily acquire a high degree of skill and wherein he is not handicapped by not having years of experience at the mine.

Although the average age of miners probably has dropped with the reduction in the number of older workers, the number of workers below 18 years of age has also decreased. States have enacted legislation that has either prohibited or limited the employment of child labor in mining occupations; in consequence no State now permits the employment at mines of minors under 16 and in some instances under 18 years old. Under the NRA code no person under 18 years of age was allowed to be employed in mining.

Greater specialization among workers and wider application of scientific production methods have also tended to increase the proportion of employees engaged in supervision, planning, record-keeping, and other occupations not directly concerned with the mining of ore. For example, in 1902 salaried employees having the above occupations amounted to 3 percent of the wage earners actually engaged in mining, whereas in 1929 they amounted to 8 percent.[4]

[4]The data for salaried employees and wage earners are from *Special Reports of the Census Office*, "Mines and Quarries: 1902" (U. S. Dept. Com. and Labor, Bur. Census, 1905), p. 504 and *Fifteenth Census of the United States: 1930*, "Mines and Quarries: 1929" (U. S. Dept. Com., Bur. Census, 1933), p. 311. Salaried employees for 1902 included "salaried officials, clerks, etc.," minus "foremen below ground."

WAGES AND HOURS

The annual earnings of full-time wage earners averaged $1,645 in 1929 and $532 in 1880.[5] The increase in real wages, however, was smaller, amounting to 111 percent during the same period.[6] Introduction of the contract or piece-rate system of wage payments early in this century, as well as the payment of bonuses and premiums for the amount of work done above an established standard, tended to increase both labor productivity and workers' earnings.

The major changes in the basic wage scales for miners per 8-hour shift during the years 1922-35 in three areas which together employed about 85 percent of the total number of workers in the industry are given in the tabulation below:[7]

Date	Butte	Arizona	Michigan
Oct. 1922	$4.75	$4.95	$3.65
Nov. 1923	4.75	4.95	4.25
Oct. 1928	5.25	5.45	4.25
June 1929	5.50	5.69	4.75
Nov. 1930	4.75	4.95	3.85
Oct. 1931	4.75	4.50	3.00
May 1932	4.25	4.05	-
Sept. 1935	4.75	4.95	-

Butte is the only major area where the base rate is the same for all ordinary classifications of underground workers. At other places the rates vary; muckers and trammers usually get about 50 cents less than miners, whereas certain groups of skilled mechanics such as hoistmen receive a rate above the base. The average wage at open-cut mines is somewhat lower than that for similar classification of workers underground. Generally the wages at open-cut mines range from about $2.50 for common labor to about $8.00 for shovel operators.

At many mines wage scales vary with the price of copper, a method of wage payment originally introduced at Anaconda Copper Company mines around 1907. This principle was embodied in the agreement between Anaconda and the Butte Miners' Union No. 1 of the International Union of Mine, Mill, and Smelter Workers. This agreement, which was entered into in 1934[8] and

[5] Annual earnings obtained by dividing the total wages paid by the average number of wage earners employed (*Special Reports of the Census Office*, "Mines and Quarries: 1902," p. 469 and *Fifteenth Census of the United States: 1930*, "Mines and Quarries: 1929," p. 311).

[6] Annual earnings deflated by the wholesale-price index published in the *Statistical Abstract of the United States: 1934* (U. S. Dept. Com., Bur. For. Dom. Com., 1934), pp. 284-5.

[7] From E. D. Gardner, C. H. Johnson, and B. S. Butler, *Copper Mining in North America* (U. S. Dept. Int., Bur. Mines, Bull. No. 405, 1938), p. 253.

[8] See "Mediation Effects Settlement of Major Labor Dispute in the Mining Industry," *Engineering and Mining Journal*, Vol. 135, No. 10 (Oct. 1934), pp. 436-7.

since renewed, set the minimum base wage for underground miners for an 8-hour day at \$4.25, with a provision that when the price of copper attains or exceeds certain levels the minimum wage is to be increased by specified amounts.[9] The basic wage agreed upon was somewhat higher than the minimum wage of $47\frac{1}{2}$ cents per hour for underground labor set by the NRA for the Northwestern Wage District,[10] the highest wage level for the copper industry.

As output per man increased the wages paid per ton of ore declined. It will be noted from the following tabulation that whereas the wages paid per ton mined amounted to \$3.19 in 1880, in 1929 they were reduced to \$1.07.[11]

Item	1880	1889	1902	1909	1919	1929
Wages paid per ton of ore produced	\$3.19	\$1.99	\$1.80	\$1.77	\$1.84	\$1.07
Wages as percent of value of ore produced	36.3	36.3	41.3	39.8	36.9	25.8
Wages as percent of total principal expenses	69.8	52.6	61.8	54.4	53.5	50.4

If the purchasing power of the dollar were taken into account, the reduction would have been even greater. More extensive mechanization, however, necessitated an increase in operating expenses other than wages, with the result that the percentage of total principal expenses represented by wages declined steadily from about 70 percent in 1880 to about 50 percent in 1929. With a rather rapid rise in productivity after 1909 following the development of porphyry mining the percentage that wages comprised of the total value of the ore mined shows a material decline - from about 40 percent in 1909 to only about 26 percent in 1929.

Although no accurate data are available on the length of the working day in the earlier period, it is generally known that the men used to

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9 See ftn. 18 of this chapter.

10 "Code of Fair Competition for the Copper Industry," *Codes of Fair Competition, Nos. 374-416* (National Recovery Administration, 1934), IX, 396.

11 The data for "wages paid", 1880, 1889, and 1902; "value of ore produced", 1880 and 1902; "tons of ore produced", 1880, 1889, and 1902; and "total principal expenses" (which includes salaries, wages, contract work, supplies, fuel, and purchased electric energy), 1880, 1889, and 1902, are from *Special Reports of the Census Office*, "Mines and Quarries: 1902," p. 469. The "value of ore" for 1889 is not reported and is estimated by the following method: The average of the ratios of the price per unit of ore to the price per unit of copper for 1880 and 1902 was multiplied by the price per unit of copper for 1889, and the product was multiplied by the quantity of ore produced in 1889. "Wages paid", "value of ore", and "total principal expenses", 1909, 1919, and 1929, are from *Fifteenth Census of the United States: 1930*, "Mines and Quarries: 1929," p. 294. Data for "tons of ore produced" for 1909, 1919, and 1929 are from *Mineral Resources of the United States, 1909* (U. S. Dept. Int., Geol. Survey, 1911), pt. I, p. 162; same, *1919* (1922), pt. I, p. 554; and same, *1930* (U. S. Dept. Com., Bur. Mines, 1933), pt. I, p. 709.

work an average of 10 hours per shift.[12] Few mines had a longer shift, and at mines where physical conditions were particularly severe the men worked 8 hours and perhaps even less per shift.

During the early part of this century a number of States enacted legislation limiting the hours of labor at nonferrous mines. All copper-producing States except Michigan and New Mexico have passed "collar-to-collar" laws that limit the workshift to 8 hours. The working time is computed from the time the men leave the surface until they have returned. Despite the absence of such legislation, copper mines in Michigan and New Mexico largely adhere to an 8-hour workshift. Under the NRA code the maximum hours were set at 40 per week or 8 in any 24-hour period, with special provisions for emergency work.[13]

EMPLOYMENT TENURE AND ECONOMIC SECURITY

Although there are definite indications of a longer tenure of employment at present than in former days, except during periods of business depressions when unemployment is widespread, no accurate records as to duration of employment are available. It is generally known, however, that as late as the early part of this century copper mines depended to a very large extent on tramp or migratory miners for their labor supply. It was a commonplace occurrence for a miner to work at one job for several weeks until he had accumulated a stake large enough to take him to another camp.[14] As most of the mines in those days operated on a relatively small scale, they were not particularly handicapped by this state of affairs. With operations conducted on a larger scale and with increased use of machinery requiring a trained personnel, high labor turnover became costly. For example, at Anaconda mines in Butte, which are the largest single employer of labor in copper mining, it has been estimated that in 1916 the cost of replacing one man ranged between $10 and $25.[15] With a labor turn-over at some mines as high as 500 percent annually, the expense incurred in replacing personnel tended to swell production costs materially. Furthermore, the requirements for skill and for a thorough knowledge of mechanical trades place certain limitations on the labor supply from which miners are recruited. Faced with this situation, the mine operators have been forced to offer inducements in

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[12] *Tenth Census of the United States: 1880*, Vol. XIII, "Statistics and Technology of the Precious Metals" (U. S. Dept. Int., Census Office, 1885), p. 157.

[13] "Code of Fair Competition for the Copper Industry," pp. 394-5.

[14] A. B. Parsons, *The Porphyry Coppers* (1st ed.; New York: The American Institute of Mining and Metallurgical Engineers, 1933), p. 561.

[15] D. Harrington, *Underground Ventilation at Butte* (U. S. Dept. Int., Bur. Mines, Bull. No. 204, 1923), p. 117.

order to prevent the miners from leaving their jobs, with the result that the average duration of employment has been increased.

The economic security of mine workers has been materially enhanced recently by the enactment of social-security and unemployment-compensation laws which tend to compensate for instability of job tenure.[16] Moreover, the tenure of employment has been protected by the National Labor Relations Act of 1935 and by similar State laws enacted subsequently. Under these acts labor has been given the legal right to bargain collectively and has been guaranteed protection against interference or coercion from employers in the administration of labor organizations and against discrimination because of union affiliations.

The miners themselves have been responsible in no small measure for increasing their economic security. By organizing themselves into the powerful International Union of Mine, Mill, and Smelter Workers and utilizing collective bargaining they have improved their wages and working conditions and have lengthened their tenure of employment.[17] With the quick growth in membership since 1934, when it won its strike against Anaconda, and with the aid of national and State laws giving labor the legal right to bargain collectively, this union has been quite successful in reaching agreements with the operators. Important provisions of these contracts are summarized below.

Most of the agreements provide for a closed shop, and an employee not in good standing in the union is not permitted to work. Wages are commonly based upon the price of copper. Minimum wages are established for underground miners, with differentials set for other workers.[18] Overtime pay is provided at one and one-half times the regular rate, and double time generally is paid for work done on holidays. In some mines 50 cents extra is paid per shift to men who work in wet locations requiring boots and slickers. The 8-hour day for underground miners, which has long been established by law in most copper-mining States, is retained. Generally discharge and reemployment of workers are based upon seniority. Those having the greatest length of service are to be given preference in retaining their jobs and in regaining employment after a shut-down.

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[16] See "Federal Labor Legislation" and "State Labor Legislation" appearing in various issues of the *Monthly Labor Review.*

[17] For an account of the growth of unionization of copper mines see "Development of Collective Bargaining in Metal Mining," *Monthly Labor Review,* Vol. 47, No. 3 (Sept. 1938), pp. 591-8.

[18] For example, at the Anaconda mines an increase of 50 cents per day is added to the basic wage when the price of electrolytic copper exceeds 9 cents per pound for a period of 30 successive days; another 50-cent increase is made when the price exceeds 9.75 cents; thereafter raises of 25 cents per day are to be given for each 1.5-cent rise in the average price of copper for any 30-day period. Reductions in wages are made on the same conditions, but each newly established rate must remain unchanged for at least 30 successive days.

All contracts provide machinery for the settlement of disputes. Usually grievances that cannot be settled between the employee involved and the foreman are referred to the mine manager or to some high official of the company. Back pay is guaranteed to workers found to have been discharged without cause. Lock-outs and strikes are barred unless all means of settlement have been exhausted. It is generally provided that should it become necessary to stop production men engaged in the operation of pumps or machinery required to protect the mine from danger or destruction continue to perform their functions. However, if the company attempts to produce, these maintenance men are not required to work. The union usually reserves the right to order these workers out if negotiations are not completed within 15 days.

Thus the tenure of employment and the economic security of the workers in the copper-mining industry have been increased materially as a result of a combination of factors. Because of the high cost of labor turn-over and the shortage of workers skilled and experienced in the art and technology of mining, operators have been compelled to offer inducements to their employees. Both the Federal and State governments have enacted social-security and labor-relations legislation, and the miners themselves have lengthened their job tenure and strengthened their economic position in the industry through unionization and the signing of agreements with the operators arrived at through collective bargaining.

On the other hand, there are forces that have affected the security of employment adversely. Chief among these are the decline in the rate of growth of production and the displacement of labor resulting from increased output per worker. Moreover, scientific methods of production and the division of labor have tended to reduce the importance of the role played by the individual employee and to render his services easily dispensable should the necessity for curtailment of personnel arise.

SAFETY

Factors that accompanied the expansion of the copper-mining industry also made copper mining more hazardous. The increase in mine hazards is attributable mainly to two causes: advances in mechanization and the necessity for mining at greater depth. The widespread use of electricity, for instance, has brought with it a host of new risks; the trolley line, in particular, has been a great source of danger and the potential cause of mine fires. Increased employment of high-speed locomotives underground has multiplied haulage accidents, and the greater utilization of power drills has increased dust hazards manyfold. As the depth of mines increased, rock bursts and rock falls become more frequent, and the high temperature and humidity in these deep workings affect the health and efficiency of the miners.

The high accident rate resulting from these new perils has focused the attention of management on the necessity of preventing injuries. The promotion of safety, apart from ethical or humanitarian considerations, has a practical appeal for the mining concerns, for it is generally realized that injuries constitute an important element of cost. Mines having a high accident rate usually must pay more for compensation, hospitalization, or insurance. This direct outlay represents but a small part of the total cost of accidents. Consideration must also be given to indirect costs resulting from the disruption of the activities of the mines; the loss of services of trained employees, either permanently or during the period of the injury; and damages to mine property.[19] It is obvious that unless the frequency and severity of injuries are reduced, the cost of production will be materially increased.[20]

One of the constructive steps has been the education of mine workers in safety, mine rescue, and first aid. Since its establishment in 1910 the United States Bureau of Mines has played a prominent part in this educational campaign. By publishing the results of its researches on safety, by having its officials lecture at safety meetings, and by sending its engineers and instructors to teach courses in accident prevention, mine rescue, and first aid to mine officials and miners, the Bureau, working in cooperation with the mining companies and State mining bureaus, has done much to reduce the frequency and severity of injuries and to improve health conditions at the mines. Another significant development has been the establishment of safety organizations by the mining companies.[21] Such organizations usually are governed by a committee consisting of some high ranking officials of the mines and representatives of different groups of workers. The work of formulating safety rules, making mine inspections, enforcing discipline, and giving safety instruction is carried on under the direction of a safety director or engineer.

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[19] It is estimated that the indirect cost of accidents is at least four times the direct cost. See D. Harrington, "Safety in Mining," *Minerals Yearbook, 1935* (U. S. Dept. Int., Bur. Mines, 1935), p. 1242.

[20] "Frequency rate" may be defined as the average number of disabling injuries per million hours of exposure; "severity rate" is measured in terms of days of disability caused by industrial injuries per thousand hours of exposure.

[21] The Anaconda Copper Mining Company, for example, has established a Bureau of Safety and has employed at each mine a safety director who works under the chairman of the Bureau and the superintendents and whose duty it is to make daily inspections of the underground workings. The company also trains its miners in first aid and rescue work and endeavors to safeguard its employees against such health hazards as silicosis by ventilating and air-conditioning its mines. See W. B. Daly, "Reducing Health and Accident Hazards in Industry" (address before the 26th Annual Meeting of the Chamber of Commerce of the United States, Washington, D. C., May 3, 1938).

Various precautions have been taken to protect miners from injury. In many mines workmen must wear hard hats, heavy gloves, hard-toed shoes, goggles, and safety belts. Blasting is delayed until the end of a shift to minimize the danger of dust diseases as well as to safeguard against falling rocks, which caused the largest number of accidents. Where dust conditions are serious other precautionary measures are introduced: drilling with water-injection drills, wetting down the broken ore before it is loaded, equipping the workmen with respirators, and improving the ventilation of the mine. Where the ground is soft and is likely to cave, the supports in working places, particularly the main haulageways, are strengthened by steel, reinforced concrete, or oversized timber. Illumination has been improved. To safeguard against the danger of fires, most mines have been equipped with fire-fighting apparatus and provided with at least two shafts, the down-cast one being used as the entrance and exit for the workmen and the up-cast for hoisting materials and supplies. Some of the mines have substituted electric cap lamps for open-flame ones, have lined the shafts with noninflammable materials, and have installed ventilation systems that may so regulate the air flow as to permit the miners to escape to the surface without injury in case of a fire and to aid in fighting a mine fire. Fast-moving parts of machines are shielded to reduce mechanical hazards, and machines are inspected frequently. Some mines have installed block signals on the haulage lines and automatic controls in the hoisting shafts to decrease hazards of transportation.

Public interest in the welfare of the miners has manifested itself in the adoption of the State safety codes. One of the main provisions of these codes is examination of mines by State inspectors. Unfortunately the inspection personnel is frequently inadequate and no State has more than five inspectors for metal mines; some States have only one.[22] In these circumstances effective enforcement of safety regulations is difficult.[23]

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[22] J. B. Andrews, "State Mine Inspection Bureaus - Are They Adequate?" *The American Labor Legislation Review*, Vol. XXVII, No. 3 (Sept. 1937), pp. 137-44.

[23] It is complicated further in many States by the provisions governing the selection of mine inspectors. California is the only State among the foremost producing States in which inspectors are appointed in accordance with accepted civil-service procedure. Although candidates for the inspectorship must prove their qualification in four other States (Montana, New Mexico, Tennessee, and Utah), they are appointed by the State administrations for a period usually not to exceed 4 years. With every change in the administration at least part of the inspection staff is also likely to be changed. Moreover, some States permit the removal of mine inspectors at the will of the appointing officer. In Arizona, Michigan, and Nevada the mine inspectors are elected by popular vote for terms of 2 or 4 years, with no training prerequisites for qualification. (J. B. Andrews, "What is Wrong With Mine Safety Legislation?" *The American Labor Legislation Review*, Vol. XXVII, No. 2 [June 1937], pp. 73-9.)

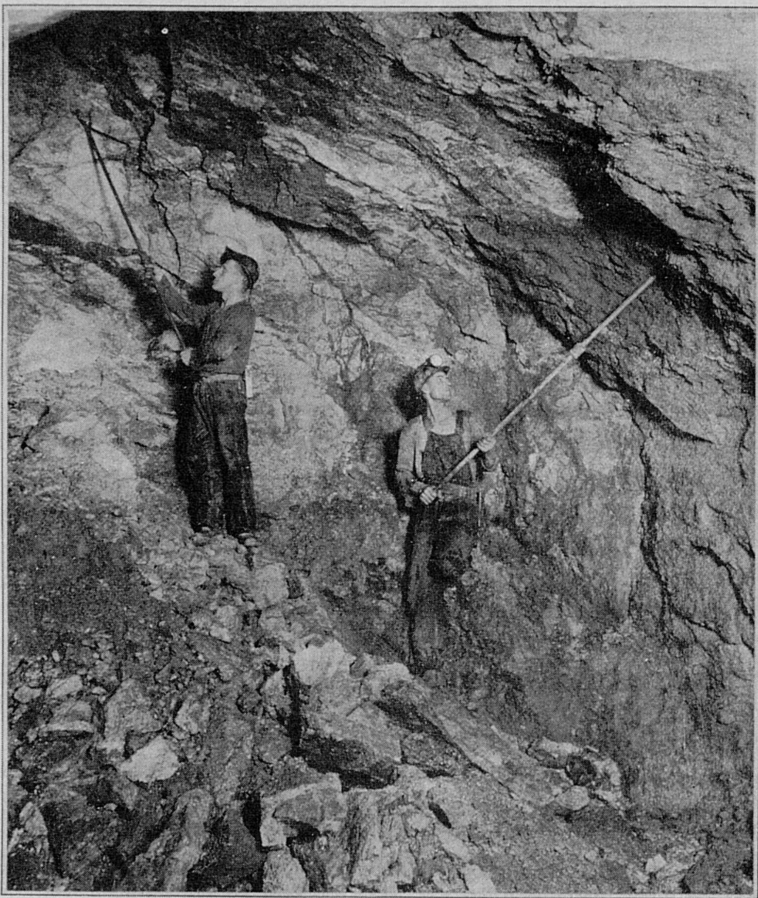


FIGURE 45.— BARRING DOWN LOOSE ROCKS IN UNDERGROUND WORKING PLACES

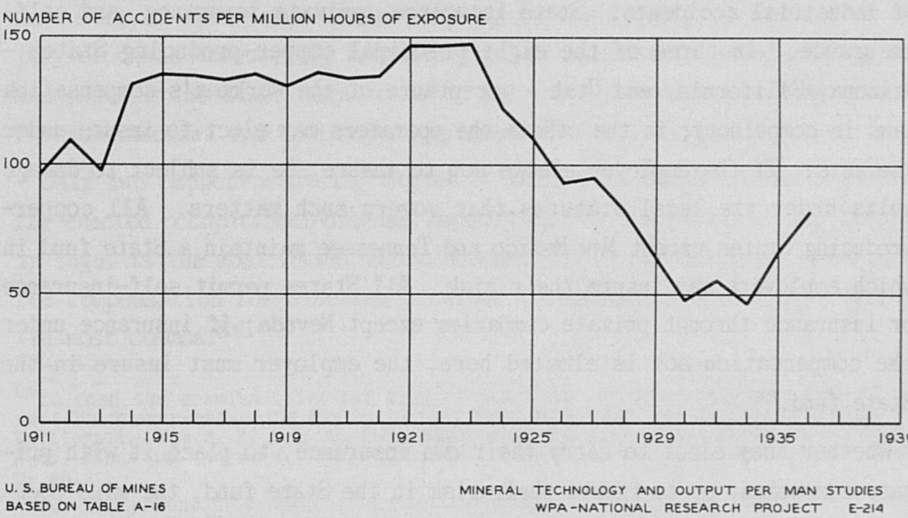
The greatest number of accidents in underground copper mines results from falling rocks. The above view shows men using long steel rods to bar down loose rocks that might fall. Such falls may be initiated by vibrations from machinery or from blasting in another section of the mine.

The thoroughness of inspection varies from State to State, depending not only on the number but also on the size, nature, and situation of mines. Most States require at least three or four inspections per year per mine, with a higher frequency for the unusually hazardous operations. As the size of the inspection staff in most instances does not even permit these minimum standards to be maintained, the primary responsibility for making the mines safe rests with the operator. The efforts of the various mine bureaus therefore are centered mainly on making certain that each mine operator is meeting his full responsibility for accident prevention. However, the greatest incentive to the operator for introducing safety devices comes from the workmen's-compensation laws and not from the efforts of the inspection agencies.[24] Recently, therefore, the tendency has been to place greater legal responsibility for the adoption of safety practices upon the mine operator.

The more recent code provisions, such as those embodied in the California code of 1937, prescribe the number and the nature of inspections to be observed by mine operators themselves.[25] Most State mine codes also prescribe certain qualifications for the mine personnel. Thus the licensing of hoisting engineers is required by nearly all States. Some States do not allow, for safety reasons, employment of persons who cannot clearly speak and readily understand the English language. In other States this regulation applies only to supervising personnel and persons engaged in responsible occupations, such as hoisting engineers. A number of States also require all mine foremen, assistant foremen, shot inspectors, and fire bosses to conform to prescribed standards of age, habit, and experience and to pass an examination on their technical qualifications before a mining board.

Because more attention has been devoted to increasing safety in mines, the accident rate during the last decade, as illustrated in figure 46,

Figure 46.- ACCIDENT RATE AT COPPER MINES, 1911-36



[24] *Ibid.*; Andrews, "State Mine Inspection Bureaus - Are They Adequate?" pp. 137-44.

[25] The California code requires operators to make at least two inspections during the working shift of those parts of the mine that are either worked or used for haulage purposes, a biweekly inspection of all parts of the mine, and a monthly inspection of all electrical equipment used. It also requires that a safety committee of at least 5 men be organized at each mine employing 25 or more men to devise means of reducing the frequency of accidents and of carrying on safety education among the men. It contains numerous provisions to insure protection against fires, the more important being those dealing with safety exits, emergency doors, alarm systems, fire-fighting equipment, and the handling of inflammable materials. For the protection of miners' health the code contains extensive provisions relating to sanitation facilities, drinking water, and supply of air in mines.

has been declining, showing a decrease of over 50 percent in the period 1925-35 in open-cut and underground operations. At many individual mines the reduction in the accident rate has been even greater. The low accident record at a number of mines indicates that a further reduction in accidents can be achieved if safety rules are observed by mine operators and workers.

Workmen's Compensation

Enactment of workmen's-compensation laws has greatly stimulated the promotion of mine safety. In fact, the provision for compensating workers for sustained injuries has been more instrumental than any other factor in reducing frequency and severity rates; it is thus making mining a safer occupation. It has made hazardous mining an uneconomical and costly way of producing copper. Moreover, it has made safety pay, for in the long run greater safety tends to reduce the cost of production materially.

All copper-producing States have legal provisions for insuring the risk of injury to employees. These laws vary considerably in their scope, amount of compensation prescribed, and whether or not their provisions are compulsory. They provide three general methods for insuring the risk of industrial accidents: State insurance, private insurance, and self-insurance. In three of the eight principal copper-producing States - Arizona, California, and Utah - acceptance of the workmen's-compensation laws is compulsory; in the others the operators may elect to insure under the acts. If the employer elects not to insure, he is subject to damage suits under the legal statutes that govern such matters. All copper-producing States except New Mexico and Tennessee maintain a State fund in which employers may insure their risk. All States permit self-insurance or insurance through private companies except Nevada; if insurance under the compensation act is elected here, the employer must insure in the State fund.[26]

Whether they elect to carry their own insurance, to place it with private companies, or to insure their risk in the State fund, the mine operators have a strong inducement to promote safety - that of lower costs.[27]

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[26]For additional details see the *Handbook of Labor Statistics, 1936 Edition* (U. S. Dept. Labor, Bur. Labor Statistics, Bull. No. 616, 1936), p. 1118. The data were originally compiled by the Bureau of Labor Statistics as of January 1, 1936. The Bureau indicates, however, that no changes in these provisions have been made since that date.

[27]It is estimated that the direct cost of accidents, comprising the compensation paid to the injured worker or his family, medical and hospital fees, and other expenses involved, amounts to about 5 percent of the pay roll. The indirect cost attributable to accidents is estimated at about four times the direct cost. See Harrington, "Safety in Mining," p. 1242.

As self-insurers, it is to their direct interest to prevent the occurrence of accidents, and to this end they have taken an active part in safety-promotion work. When the insurance against the risk of accidents is carried with private companies or in a State fund, the mine operators try as much as possible to avoid the occurrence of accidents in and about mines because, with the decrease in the frequency and severity of accidents in a particular area, the insurance rate for that area is reduced.[28] This incentive to promote safety is strengthened in some States by provisions for allowing individual employers a credit in the form of lower insurance premiums for the adoption of safety measures and equipment and for achieving a favorable accident record. Private insurance companies are also an active factor in promoting safety in mines. When they undertake to insure operators against accident risks they insist upon wider application of safety devices and stricter observances of safety codes. These companies maintain a staff of safety inspectors who visit each mine at least once a month and thus render valuable assistance in the prevention of accidents.

Workmen's compensation not only effects greater safety in mining but also increases the economic security of the workers. Although in many instances the compensation paid may not be adequate, the knowledge that some sort of assistance is forthcoming to their families in case of injury or death undoubtedly promotes greater contentment among the workers and reduces the labor turn-over.[29] All State compensation laws provide for some medical aid to injured employees.[30]

Only two copper-producing States - California and Michigan - have so far enacted occupational-disease legislation.[31] The Michigan law, enacted in 1937, is the most recent and comprehensive.[32] It includes a provision for compensation for diseases peculiar to mining, of which silicosis is the most common.[33]

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[28] In 1930 the premium rates per $100 of pay roll for metal mines in some of the western copper-producing States varied from $4.57 for New Mexico to $10.54 for California. See S. H. Ash, *Accident Experience and Cost of Accidents at Washington Metal Mines and Quarries* (U. S. Dept. Com., Bur. Mines, Tech. Paper No. 514, 1932), p. 29.

[29] For the amount of compensation and the method of payment see *Handbook of Labor Statistics, 1936 Edition*, pp. 1123-30.

[30] *Ibid.*, p. 1131.

[31] *State Labor Legislation, 1937* (U. S. Dept. Labor, Bur. Labor Statistics, Bull. No. 654, Jan. 1938), p. 18.

[32] *Occupational-Disease Legislation in the United States, 1936* (U. S. Dept. Labor, Bur. Labor Statistics, Bull. No. 652, Dec. 1938), pp. 78-81.

[33] In the case of death or total disability the amount of compensation is limited to a maximum of $3,000. Other provisions are essentially similar to those included in the State Workmen's Compensation Act. Under the California law enacted in 1931, occupational diseases are covered by the term "injury" and are thus subject to the provisions of the State Workmen's Compensation Act. (*Ibid.*, p. 6.)

CHAPTER VII

FUTURE OUTLOOK[1]

Future outlook for employment opportunities in the copper-mining industry is determined by a number of factors whose behavior cannot be predicted with certainty. These factors may be grouped into two broad categories - those affecting copper demand and those affecting the output per worker in the industry. The former group will be discussed later in this chapter. The output per worker is affected by the availability and quality of ore reserves and by the success with which technologic advances will be able to cope with ever-increasing natural difficulties in the extraction of copper ores. Thus the number of future employment opportunities depends mainly on the availability and character of ore reserves, the trend of technologic advances in the industry, and the domestic mine production as determined by demand.

COPPER RESERVES

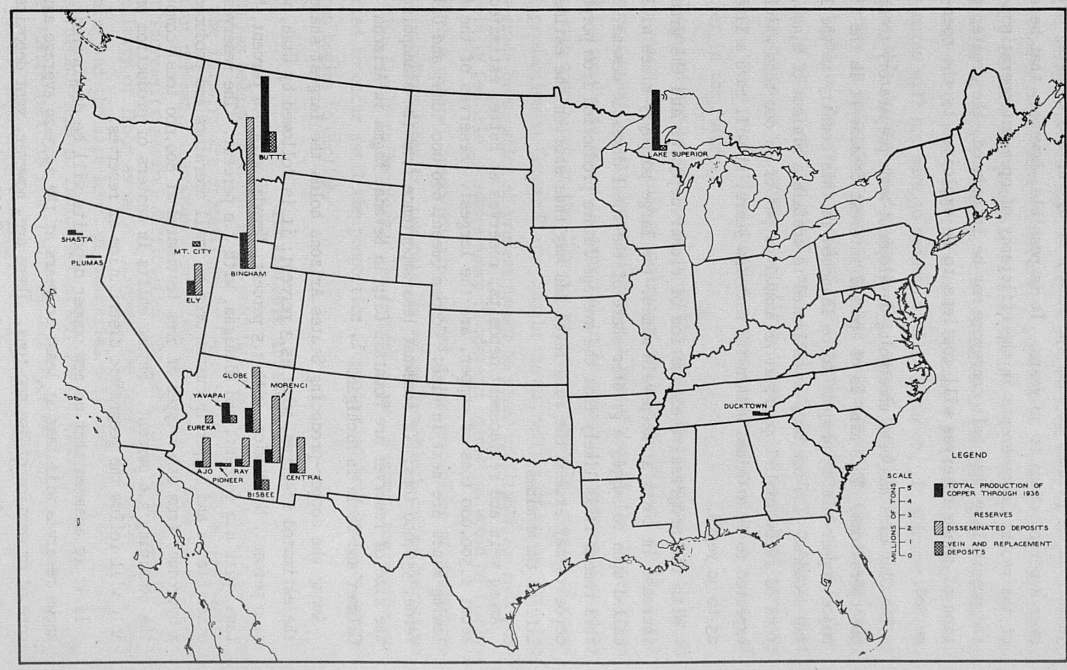
An estimate of copper resources must consider the market price of copper. Ore deposits that cannot be exploited profitably at the prevailing price of the metal cannot be considered as reserves. The volume of reserves would thus vary with the price of copper. The estimate of copper reserves in the United States, as shown in table A-18, represents the number of tons of recoverable copper that can be mined under conditions of the technology of today to be sold within the price range of 8 to 17 cents per pound. Figure 47 illustrates the geographical distribution of these reserves.

By far the greater part of the known deposits is of the disseminated or porphyry type. Porphyry deposits constitute 31,000,000 tons of the total estimated reserves of about 36,000,000 tons of recoverable copper; about two-thirds are likely to be mined by the open-pit method. These deposits are largely confined to five mines: Utah Copper, Chino, New Cornelia, Copper Flat-Ruth, and Clay.

The known reserves of Utah Copper head the list; at the rate of copper production in 1929 they will, as of 1939, last about 75 years. On the same basis, the reserves of the Chino and New Cornelia will have a probable life of about 60 years. At the Nevada Consolidated mine (Copper Flat-Ruth) at Ely, Nevada, the original plans called for the mining of the ore by caving methods. At the 1929 rate of extraction, that part of

[1]By Y. S. Leong, Emil Erdreich, J. C. Burritt, O. E. Kiessling, and C. E. Nighman.

Figure 47.- ESTIMATED COPPER RESERVES AND QUANTITY OF COPPER EXTRACTED THROUGH 1936, BY DISTRICT



U.S. BUREAU OF MINES
BASED ON TABLE A-18 AND PUBLICATIONS AND RECORDS OF BUREAU OF MINES

MINERAL TECHNOLOGY AND OUTPUT PER MAN STUDIES
WPA-NATIONAL RESEARCH PROJECT E-215

the reserves to be excavated by the open-cut method will last 10 to 15 years. It is planned to mine the remainder by the block-caving method over a period of 20 to 30 years. It is possible, however, that because of the recent improvements in the efficiency of open-cut operations and the possibility that belt conveyors may be introduced, the greater portion of these reserves will continue to be exploited by the open-cut method.

The Clay mine has been undergoing development work preparatory to open-cut operations. This mine is of special interest because it is the first major copper-mining project to be launched in this country in the last two decades. The ore body is estimated to contain a minimum of 2,440,000 tons of recoverable copper; at an annual output of 90,000 tons which is expected to be attained within the next few years, it will have a life of 25 to 30 years.

With the progressive exhaustion of richer deposits and the gradual increase in costs at deep-vein mines, the large porphyry mines will be called upon to supply a greater share of the total domestic demand. For this reason it is likely that the average future production from porphyry deposits may exceed the 1929 level and may thus shorten the estimated life of these mines.

Among vein and replacement deposits, reserves at Butte, estimated at about 1,500,000 tons of copper, are the largest. Reserves of the following mines are next in size: Copper Queen, 670,000 tons; and United Verde, 550,000 tons. Of somewhat less importance from the standpoint of the size of reserves are Mountain City in Nevada, Magma in Arizona, and Calumet and Hecla in Michigan.

Among the copper-producing States Arizona holds the largest share of the estimated reserves, with 45.3 percent; it is followed by Utah, with 31.0 percent, New Mexico, with 7.5 percent, Nevada, with 7.4 percent, Montana, with 4.2 percent, and Michigan, with 1.0 percent. The reserves in other States and copper in custom ores of small operators and produced as a byproduct from other types of ores (estimated at 800,000 tons) comprise the remaining 3.6 percent. Future shifts in centers of production probably will follow the geographic distribution of reserves.

If it is assumed that no new copper deposits will be discovered, the known reserves will last at least 40 years at the 1923-29 average annual rate of production of 850,000 tons. There are, however, vast deposits of porphyry ores known to exist in Arizona and California, but these are of such low grade that exploitation at present is not commercially feasible. With further improvements in milling and concentration practices, with cheaper power resulting from the growth of hydroelectric plants, and with higher copper prices the recovery of copper from some of these deposits eventually may become commercially possible. In addition, allowance should be made for possible future discoveries. In the past, various estimates of copper reserves were consistently revised upward because of the new discoveries, and there is no reason to believe that the present estimates will not have to undergo similar revision. It appears that, as far as this country is concerned, the fear of rapid depletion of copper reserves is unfounded, even though the richer deposits are on the wane. The trend in the average yield of ore, however, is downward and will continue in this direction partly as a result of the gradual exhaustion of richer deposits but more especially because of the growth of porphyry mining, which is destined to play a more important role as a source of copper in the future.

Copper mining by the open-cut method is likely to expand for 15 to 20 more years and to attain a maximum of perhaps 60 percent of the total copper output. Curtailment of open-cut operations at the Copper Queen and United Verde mines is expected to be more than counterbalanced by open-cut operations at the Clay ore body and a possible increase of open-cut production at Copper Flat. Should the extraction of copper from the low-grade ores, which at present cannot be mined profitably, be made commercially feasible, a sudden spurt in open-cut operations would be likely.

With a general trend pointing toward a greater volume of production from porphyry deposits, the relative volume from block-caving operations is also likely to increase. The largest gains, however, may not be expected to occur until the proportion of production from open-cut operations begins to decline.

TECHNOLOGIC ADVANCES

As has been pointed out in the preceding chapters, technologic advances in the industry are unquestionably the outstanding factor influencing output per man. Improvements in technology of mining have not only offset increasing natural difficulties but have materially improved productivity of labor as well. However, the rapid progress in the art of mining and the high level of managerial efficiency attained within the last decade tend to indicate that further technologic advances in the near future (next 10 years) will be less important than in the past 10 to 15 years. Important technologic changes that probably will occur in the copper-mining industry have been indicated in the preceding chapters. It is sufficient at this point, to recall and summarize some of the impending developments that are likely to have a significant effect on the productivity of labor.

At the four open-cut mines, Utah Copper, New Cornelia, Chino, and Copper Flat, further increase in the productivity of labor will depend mainly on the replacement of old by new equipment. In view of the fact that mine operators for the most part have been deferring their replacements during the past 10 years because of the prolonged depression and recurring recessions, purchases of new equipment in the near future may be substantial if business conditions improve enough to induce the outlay. Modern, full-revolving electric shovels will be substituted for some of the old-style power excavators, many of which were acquired at least 20 years ago and some of which would undoubtedly have been replaced but for the discouraging business prospects of the past decade. In transportation, too, there will probably be some extensive replacements. Larger, heavier rolling stock of better design and construction will supplant the old and obsolete equipment. The use of trucks will probably expand, particularly as the depth of mines increases. As this type of haulage equipment can negotiate steeper grades and sharper curves than railroad locomotives, it may prove to be the solution of the transportation problem at pits where depth will be increasing. There is the possibility that belt conveyors may be adopted at one or more of the mines. In the realm of drilling, the new type of churn drills, which can bore holes as large as 9 inches in diameter at high speed, will find wider application and will displace some of the other types of drills now in use. Of course, additional mining machinery will be purchased if the existing open-cut mines expand their output sufficiently far above the peak level of 1929. At the Clay mine, a substantial volume of new drilling, loading, and transportation equipment will be required when large-scale production is commenced.

At underground mines there is a decided trend toward further increase in mechanization. More and improved labor-saving machinery will be installed, and animal and human power will be further displaced. The use of electricity will become more widespread. Scrapers, which have been increasing in size and efficiency, and mechanical shovels, which have been rapidly displacing hand shoveling, will be adopted more widely. More extensive use and constant improvements of these loading devices will raise the labor productivity of those engaged in loading and will lessen the drudgery of mucking. Transportation equipment will be further improved and standardized, and the haulage system will be better coordinated. Locomotives and mine cars will be enlarged and made more efficient; storage-battery locomotives will continue to supplant hand tramming; and belt conveyors, which have been utilized successfully at iron mines, may be introduced at some copper mines. Despite the growth in mining depth, hoisting efficiency will continue to increase. Speedier and more powerful equipment will be introduced, and multiple-compartment shafts will be utilized more extensively. The weight of the cable will be decreased by the use of steels of higher tensile strength and that of the cage or skip by the use of light-weight alloys. This reduction in dead weight makes possible the hoisting of a larger volume of ore or hoisting from a greater depth without an increase in the size of the hoist or cable. In the field of drilling the principal advances will consist of an increase in drilling speed, reduction in power consumption, improvement in the mobility of the drills, and wider application of detachable bits. The development of large rotary drills that can cut a core as large as 5 feet in diameter may change methods of shaft sinking. Mine illumination will be improved further with wider adoption of portable flood lights and electric cap lamps. Increased efforts will be made to ameliorate working conditions, mechanical ventilation will be improved, and as the heat and humidity rise with increasing depth of mines, air-conditioning systems will be installed.

No great changes are anticipated in mining methods in the near future although further refinements can be expected in the adaptation of the most effective methods to the appropriate types of deposits. Over a longer period, however, there are many forces that tend to stimulate improvements in mining techniques. The more important of these include the increasing natural difficulties arising from greater depths at which new ore bodies may be developed, the exploitation of ores much poorer in copper content than those that are now being mined, the increasing cost of labor and supplies, and a general reduction in the market value of copper products resulting from competition of other products or from other causes. Such factors make for higher production costs and narrower margins of profit, and thus serve as a constant incentive for improving mining techniques. As a result, mining operations are becoming more and more standarized, particularly at mines employing artificial supports. Standardization, of course, tends to raise labor productivity. Future progress in mining methods probably will be in the direction of obviating the necessity of filling with waste, restricting the use of square sets, and applying the retreating system of mining more extensively. Eventually a system may be developed whereby large ore bodies may be mined in vertical slices starting from the extreme ends or sides of the deposit and retreating laterally, and from a lower level upward, rather than by the usual method of mining horizontal slices starting at the upper levels and near the shaft.[2] Careful advance planning of mining operations and geological control will decrease the volume of dead work and the tonnage of waste handled, and this in turn will be reflected in an increase in the output per man.

Mining difficulties at open-cut mines will increase somewhat, but they are unlikely to constitute an important problem, at least in the next 10 years. At underground mines, however, physical handicaps will become more serious. Some of the vein mines, which have already descended to great depths or have depleted their richer deposits, may find it necessary to close down partly or entirely. With partial or complete abandonment of some of these mines, a larger share of the output will be forthcoming from the more productive vein mines and, of course, from porphyry mines employing the caving system. Whatever increase in physical difficulties there may be will therefore be unlikely in the near future to stem the tide of technologic progress.

The introduction of technologic improvements in the past has been fostered by the strengthening of the financial position of operating concerns through consolidations of small properties into larger enterprises. Although it seems probable that this general tendency toward fewer and larger enterprises will continue, it must be remembered that the number and the size of operating units will depend mainly on the character and occurrence of ore deposits, a fact that places a definite limitation on the potentiality for further consolidations.

The selection and training of mine labor have a significant bearing on the output per man. Wider use of mechanical devices requires better (though more narrowly) trained and quickly responsive men. Technically trained men are likely to command a higher premium, because knowledge of how to handle a piece of complicated machinery not only results in increasing output but also in reducing wear and tear on the machine and in decreasing the number of break-downs and delays. It may be expected, therefore, that greater emphasis will be placed on training of personnel at the mine.

OUTPUT PER MAN

The trend of technologic advances and the nature and geologic structure of available deposits will largely determine the future output per worker in copper mining. These factors, together with past trends of output per worker, may serve as indications of future productivity of labor.

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[2] C. W. Wright, *Mining Methods and Costs at Metal Mines of the United States* (U. S. Dept. Com., Bur. Mines, Inf. Cir. No. 6503, mimeo., June 1931), p. 33.

Essentially, output per man is determined by two elements: the relative volume of production by each mining method (or, roughly, the proportion of total output by underground mining and by open-cut operations), and the productivity of each method as measured by the quantity of copper recovered per unit of labor expended.

It may be estimated that in open-cut mines the average output per man-hour in 1947[3] or thereabouts is likely to reach 3.04 tons of ore.[4] This estimate is not overoptimistic in view of the fact that it represents only a 45-percent increase over the 1923-31 average. Technologic improvements that have enabled these mines to increase their output per worker undoubtedly will continue to exert their stimulating influence in the future. It is expected that yield of ore will decrease as a result of the decline in the grade of ore. On the assumption that the yield of a ton of ore will be 17 pounds of copper, or a reduction of 2 pounds from the 1923-31 average, the output of recoverable copper per man-hour in 1947 is estimated at 52 pounds.

In underground mines an output of 0.74 ton of ore per man-hour may be estimated for 1947[5] compared with 0.44 ton for 1923-31 and 0.45 ton for 1929. Physical difficulties, particularly at deep vein mines, may increase, but technologic advances, together with the shift in the output from the less productive mines to the more productive vein and especially porphyry mines employing the caving method, will more than counterbalance the depressing effect of mining handicaps on the output per worker. If a yield of 35 pounds of recoverable copper per ton of ore is assumed, which is about 4 pounds less than the average yield for 1923-31, the output per man-hour may be expected to reach 26 pounds of copper in 1947. As ore dressing has already attained a high degree of efficiency, further improvements in metallurgy will be of minor importance and consequently need not be considered in the present estimate.

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[3] All estimates for 1947 refer to the last year of the approximate decade following the most recent year (1936) for which detailed statistics were available for consideration in this report.

[4] This estimate is obtained by assuming that the average annual increment of increase during the period 1917-31 will continue to 1947. The trend line upon which the estimate is based was fitted by the method of least squares to the output of ore per man-hour for the period 1917-31. The equation with the origin at 1924 is $y = 2.0240 + 0.0442x$, where y represents tons of ore and x represents a time unit of 1 year. Because of abnormal conditions the period after 1931 is omitted from the computation of the line of growth. The output per man after 1931 was greatly distorted by a number of factors, among the more important of which are selective mining, changes in the stripping ratios, operation of some of the mines at partial capacity, and closing down of one or more of the mines. It is assumed that as production approaches predepression levels by 1947 (see section on "Production" later in this chapter) these factors will no longer be operative.

[5] This estimate is derived on the assumption that the average yearly increment of increase during the period 1917-30 will be carried forward to 1947. The equation for the trend line is $y = 0.2805 + 0.0147x$. As in the case of the open-cut mines, the period during the depression is disregarded in the computation of the line of estimation because of the presence of factors which tended to distort output per worker for this period.

The relative volume of output that will be produced at open-cut and underground mines in 1947 is more difficult to estimate. The decline in open-cut production resulting from the cessation of open-pit operations at the United Verde and Sacramento Hill mines will probably be more than counterbalanced by the future output from the Clay mine, which may begin large-scale excavations in the next few years. Since the other open-cut mines have already reached maturity, having undergone the preliminary stages of development, no significant changes in their share of the total output can be expected, at least in the near future. In recent years the open-pit mines contributed about 40 percent of the total production for all mines. In about 1947 the four producing open-cut mines together with the Clay mine will probably produce about 45 percent of the total copper output. By weighting the estimated output per man for open-cut mines by 45 and that for underground mines by 55, an estimate of 38 pounds of recoverable copper, or roughly 300 pounds per 8-hour shift, may be obtained for the industry as a whole in about 1947.

Over a longer period, the over-all output per man can be estimated only with a great degree of uncertainty. It is safe to state, however, that the upper limit[6] is set by the output per man at all porphyry mines. The relatively high level of productivity now reached at these mines will probably not be attained by all mines, for even if the porphyry deposits become the chief source of new copper, a portion of the total supply will continue to be derived from vein and replacement deposits which will be mined with greater expenditure of labor.

PRODUCTION

Among the factors that must be taken into consideration in estimating future employment opportunities, that of future production is the most uncertain. So many developments in so many different fields may influence it that, even at best, only the approximate range of their effects on production can be indicated here. Three primary factors must be taken into account: domestic consumption requirements, secondary production from old scrap, and foreign markets and foreign competition. These factors are discussed in reverse order.

Foreign Markets and Foreign Competition

In all probability foreign demand for copper in the immediate future will increase if the European war becomes more intensified. With the cessation of hostilities, however, foreign markets probably will offer

[6] This is a variable limit which depends upon the increase in labor productivity in open-cut operations. About all that can be done is to hazard a guess of the actual limit of output per man in such operations a decade hence.

little opportunity for domestic producers. As many countries are striving for self-sufficiency in copper and as the mines in Rhodesia, Chile, and Canada are expanding their production, United States producers will have to look largely to domestic consumers for disposition of their products.[7] During the past few years foreign production of copper has been more than adequate to meet the demand abroad notwithstanding heavy purchases for armaments and war reserves.[8] With the return of nations to peaceful pursuits, this supply situation in foreign markets is likely to prevail once more.[9]

On the other hand, there is no great danger of foreign competition in the domestic market. As a result of the net flow of copper into this country in 1930-32, a tariff of 4 cents per pound was levied on copper imports in 1932, and it has been continued. Should foreign competition increase, higher tariff barriers may be erected to protect home industry. It seems likely, then, that the mines of this country will, in normal times, operate primarily to supply domestic demand. On these assumptions, imports and exports of copper can be largely neglected in estimating the future trend of domestic production.

Secondary Copper[10]

Secondary production from old scrap is an important source of copper supply; its future significance, however, often has been exaggerated.

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[7]The United States used to supply a large part of the foreign demand for copper, but in recent years Rhodesia, Chile, and Canada have been displacing this country in foreign markets. Apparently the mines in these countries, particularly those in Rhodesia and Chile, have been able to produce copper at a much lower cost than the average in the United States. It may be observed in the following tabulation, showing the relative position of the principal copper-producing countries (as represented by the percentage of the mine output of each country to the world mine output) that the United States lost considerable ground between 1929 and 1937.

Country	Percent of total	
	1929	1937
United States	46.7	32.8
Chile	16.5	17.7
Rhodesia	0.3	10.7
Canada	5.8	10.3
Belgian Congo	7.1	6.5

In 1929 this country produced about 47 percent of the mine production of the world, whereas in 1937 it contributed only 33 percent. Rhodesia, Canada, and to some extent Chile, gained at the expense of the United States. Data for the tabulation above are from *Mineral Resources of the United States, 1930* (U. S. Dept. Com., Bur. Mines, 1933), pt. I, pp. 693-4; *Minerals Yearbook, 1939* (U. S. Dept. Int., Bur. Mines, 1939), p. 115.

[8]By entering into an international cartel agreement, the larger copper producers outside of the United States have had some success in adjusting production to demand, so that except at short intervals when the supply and demand were temporarily out of balance, there could not have been overproduction. These producers, however, have a combined productive capacity which, if unrestricted, will supply copper far in excess of the armament and industrial demand outside of the United States, except perhaps during active and extensive warfare.

[9]At the end of a war there may be an outflow of copper from the reserves accumulated by both warring and neutral countries.

[10]Secondary copper is derived from two sources: old scrap and new scrap. Old copper scrap may be defined as copper reclaimed from discarded used articles, such as old trolley wires or transmission lines; new copper scrap, as copper derived [*Con.*]

The true role of this secondary copper is better indicated by its relationship to total copper consumption than by its relationship to domestic mine output. In the post-war period the production of secondary copper from old scrap was increasing at about the same rate as the consumption of copper in all forms. This tends to indicate that there is no disproportionate growth of secondary production.[11] It is true that during depression periods the production of copper from old scrap often is greater than the volume of new production, but this phenomenon is due chiefly to the fact that primary production can be much more readily controlled than can secondary production. The larger volume of secondary output in relation to the greatly curtailed mine production in periods of depression, when demand for copper slackens, tends to magnify at least statistically the importance of the secondary supply. Moreover, over a longer period, reclamation of copper from old scrap is certain to decline unless there is an increase in the consumption of new copper or a change in the character of consumption permitting a greater proportionate recovery of scrap copper.

During the 1923–29 period secondary production averaged roughly 35 percent of total consumption,[12] with only minor deviations from this figure (see table 6). This average may be taken as a safe estimate for the next 10 years. Over a longer period perhaps a moderate increase in the relative volume of secondary copper may be expected.[13]

Consumption Requirements

A number of inherent difficulties are involved in attempting to chart the future course of copper consumption. Growth and distribution of national income, increase in population, a wider industrial and commercial

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[10] *[Con.]* from the refuse produced in the fabrication of articles (such as clippings) and from unused finished and semifinished articles which were discarded because of faulty manufacture, change in design, excessive inventory, and other causes which render these products unsalable. The output of copper from new scrap tends to vary directly with the volume of copper manufactures and with general business conditions. The output of copper from old scrap, on the other hand, is less responsive to the business cycle, for the collection of old scrap is likely to continue in prosperity and in depression with relatively little change.

[11] For an interesting study of secondary copper in the United States see P. E. Barbour, *Secondary Copper: Its Production, Consumption, and Market Effect* (New York: Mining and Metallurgical Society of America, 1936).

[12] Barbour (*ibid.*, p. 59) places secondary production at about 25 percent of total consumption. He excludes copper content of brass and other copper alloys, and includes both old and new scrap in his estimate, whereas the figure of 35 percent is computed on the basis of old scrap including the copper content of brass and other copper alloys.

[13] As the copper-consuming industries gradually approach maturity, secondary copper will become relatively and progressively more important than mine or primary output. With the attainment of maturity and stability by these industries, scrap copper will become the dominant factor in the supply situation. Except for that part of the copper which is dissipated by use and must be replaced and that part which goes into such new uses as may be developed the entire amount of domestic consumption will then be derived from secondary sources. So far as the important copper-using industries are concerned, however, maturity will not be reached for some time to come.

Table 6.- TOTAL COPPER CONSUMPTION AND SECONDARY COPPER PRODUCTION, 1919-37

Year	Copper consumed[a] (short tons)	Secondary copper produced[b] (short tons)	Secondary production as percent of total consumption
1919	621,375	152,600	24.6
1920	684,035	168,960	24.7
1921	459,865	131,990	28.7
1922	539,113	202,800	37.6
1923	737,700	270,900	36.7
1924	765,950	266,200	34.8
1925	836,000	291,010	34.8
1926	909,900	337,300	37.1
1927	862,450	339,400	39.4
1928	990,700	365,500	36.9
1929	1,159,800	404,350	34.9
1930	956,150	342,200	35.8
1931	650,000	261,300	40.2
1932	368,000	180,980	49.2
1933	415,000	260,300	62.7
1934	463,000	310,900	67.1
1935	574,000	361,700	63.0
1936	749,000	382,700	51.1
1937	860,000	408,900	47.5

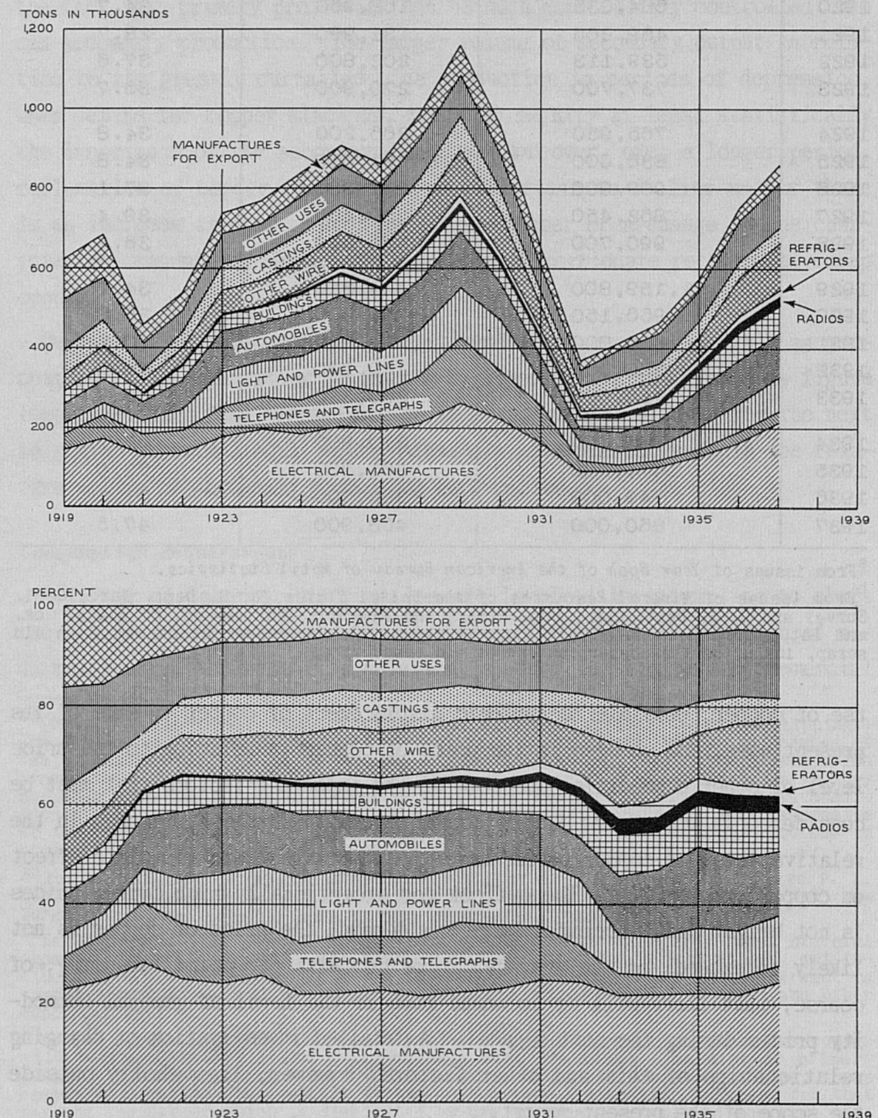
[a]From issues of *Year Book of the American Bureau of Metal Statistics*.

[b]From issues of *Mineral Resources of the United States* (U. S. Dept. Int., Geol. Survey and U. S. Dept. Com., Bur. Mines) and *Minerals Yearbook* (U. S. Depts. Com. and Int., Bur. Mines). These figures represent the copper recovered from old scrap, including the copper content of old brass scrap.

use of copper, possible technologic displacement of copper in some of its present uses, and changes in the price of copper in relation to the price level of other commodities are some of the important factors that must be considered in estimating future copper demand. Of these, changes in the relative level of copper prices have the most direct and immediate effect on copper consumption. The outlook for exceptionally high copper prices is not bright and, barring a large war demand, the price of copper is not likely to exceed, on the average, 15 or 16 cents per pound (assuming, of course, that there will be no undue rise in the level of general commodity prices during the next decade). A detailed investigation of changing relationships of this and other factors, however, lies wholly outside the scope of the present report.

There are certain specific indications as to the probable future trend of copper consumption. The predepression trend of per-capita copper consumption was steadily upward. New uses of copper are constantly being developed, thus adding cumulatively to the existing consumption requirements. Estimated consumption by principal copper-using industries is illustrated in figure 48. A brief survey of copper utilization by these industries clearly points to a larger demand for copper in the future.

Figure 48.- ESTIMATED USE OF COPPER IN THE UNITED STATES, BY PRINCIPAL USE, 1919-37



U. S. BUREAU OF MINES
BASED ON TABLE A-17

MINERAL TECHNOLOGY AND OUTPUT PER MAN STUDIES
WPA-NATIONAL RESEARCH PROJECT E-216

With the existence of a general tendency toward industrial and farm electrification, copper requirements for electrical machinery, equipment, and light and power lines seem certain to increase greatly. Electrification of steam railroads is likely to be particularly important, although until now only about 3 percent of the total railway mileage has been electrified. This general tendency toward electrification is accentuated by a rapid development of hydroelectric plants which make electric energy cheaply available to scattered communities often located far from the sources of power. Not only do the low power rates increase consumption of electric energy where it is already available but in many instances they also make it possible to start industrial development in areas where, in the absence of cheap fuel, important industries have been unable to secure a foothold.

The consumption of copper by telephone and telegraph industries, however, is not likely to reach the high level of 1929 for some time to come. Even with an appreciable increase in the number of telephone users, additional connections can be made with but very few new installations of main cable lines. In most communities such trunk lines have already been laid, and an extension of service can be achieved by providing only branch outlets. Moreover, the number of messages that can be sent over a single wire has been materially increased, thus obviating in many instances the necessity of installing additional cables. Most of the copper demand will therefore arise from replacement requirements, but considering the long life of cables, increase in the use of copper for this purpose will not be likely to be significant. The telegraph network is already dense. Moreover, an expansion in the use of telegraph facilities requires practically no additional copper, particularly since it is now possible, as in the case of the telephone, to send many more messages simultaneously over a single wire. Thus most of the consumption requirements will be largely for replacement and maintenance uses.

The consumption of copper by the automobile industry will increase practically in direct relation to the number of new automobiles produced. As the production of automobiles per capita is gaining, there is every reason to believe that, with growth of the national economy, rise in the national income, and general improvement in the standard of living, the consumption of copper by this industry in the next decade will continue to increase.

Mechanical-refrigeration, air-conditioning, and radio-television industries, which are still in their infancy or perhaps in their youth, are certain to grow and to increase their use of copper in the near future.

As they develop and continue to mature into the mass-production stage there will be important price reductions, and with lower prices demand for these products will rise and the use of copper will expand accordingly. The television industry, which is perhaps more in the embryonic than in the infancy stage, is a particularly promising consumer of copper.

The building-construction industry will continue to be an important user of copper. In recent years the consumption of copper and brass per housing unit has been growing steadily. Copper is used not only for plumbing and heating installations but for the construction of houses as well. Prefabricated houses, in particular, are heavy consumers of this metal. It is also widely used for interior decoration and for safeguarding buildings against destruction by termites. With a general trend toward higher housing standards, the building-construction industry is therefore very likely to play an increasingly important role in creating a demand for copper in the not-too-distant future.

There are a number of other fields where copper is utilized rather extensively. A few of these are railway equipment, shipbuilding, castings, wire, alloys, coinage, clocks and watches, washing machines, and fire-fighting apparatus. These uses take 20 to 30 percent of the total copper consumed. Assuming a gradually rising level of industrial development, nearly all of the above fields are likely to increase their consumption.

Although copper requirements for armaments and ammunition have been an important item of copper consumption, the future trend is unpredictable. However, in view of the fact that this country, together with the rest of the world, has embarked on a rearmament program, the use of copper for naval construction, land and air armaments, and ammunition will expand during the next few years at least.

Some factors adverse to an increase in consumption of copper should be mentioned. The rate of increase of the population is gradually declining. In certain fields there is a tendency to use less copper than formerly; for instance, trolley lines are being replaced by busses. Moreover, in some industries competing metals are being substituted for copper; for example, aluminum is replacing copper in the construction of high-tension transmission lines.

When so many variables influence the demand for copper, it is of course not possible to estimate future consumption accurately. Nevertheless an approximation of the future level of consumption or production is indispensable if an attempt at forecasting employment is to be undertaken.

One way of arriving at an estimate of the future level of consumption is to compute the trend line for the per-capita consumption of copper and to assume a figure somewhat higher than the trend value for 1930. By multiplying this figure by the estimated population in the year for which the estimate is made, the amount of copper that is to be consumed in that year may be approximated. For the present purpose, 22 pounds may be accepted as the probable consumption in 1947. The trend line computed for 1919-30 indicates that the per-capita consumption of copper was increasing by 4.6 percent per year.[14] If this line is extended, it will reach 22 pounds in less than 5 years. In view of the preceding discussion regarding the potential demand for copper by the principal copper-consuming industries, it seems quite probable that this level of consumption will be attained in 10 years.[15] With the population estimated at 139 million[16] and per-capita consumption at 22 pounds in 1947, the total domestic consumption in that year would be around 1,500,000 short tons. A check of the reasonableness of this figure was made by plotting on graph paper the curves of copper consumption by individual copper-using industries for 1919-36, smoothing these curves, and extending them freehand to 1947, taking into consideration various factors discussed above that are likely to influence the consumption of copper by each use in making the projections. By aggregating the values read from the extended curves, a total of 1,400,000 short tons was obtained. The estimate of 1,500,000 tons, therefore, although subject to the limitations of any such projection, may be taken as a rough approximation of total copper consumption in the United States in 1947. Possibly it may err on the conservative side.

Since secondary copper from old scrap is likely to compose about a third of the total consumption, the domestic mine production may be expected to be around 1,000,000 tons of copper in 1947,[17] which is practically equal to the mine production in 1929.

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[14]The trend line was fitted by the method of least squares to the per-capita consumption of copper for the period 1919-30 as estimated by the American Bureau of Metal Statistics and published in its yearbooks. The equation, with the origin at 1919, is $\log y = 0.99445 + 0.02086x$, in which y represents per-capita consumption of copper and x represents a time unit of 1 year.

[15]This estimate of 22 pounds represents the trend of consumption rather than an estimate of the actual per-capita consumption in 1947, which may be above or below the trend value, depending on business conditions at that date. The per-capita consumption will probably exceed this estimate if 1947 is a year of business prosperity, and vice versa if that year happens to be one of business recession.

[16]Interpolated figure based on estimates of future population assuming medium fertility and mortality and no net migration of foreign-born persons. Prepared by Thompson and Whelpton of the Scripps Foundation for Research in Population Problems, and published in National Resources Committee, *Population Statistics*, Pt.1, "National Data" (Oct. 1937), p. 9.

[17]Parsons and St. Clair estimate the minimum annual consumption of primary copper in 1947 at about 900,000 tons. This figure is not actually given by them. but is read from their line of estimation projected from 1936 to 1954. See A. B. Parsons and S. St. Clair, "Outlook for World Consumption of Metals and Fuels," *Mining and Metallurgy*, Vol. 18, No. 364 (Apr. 1937), p. 193.

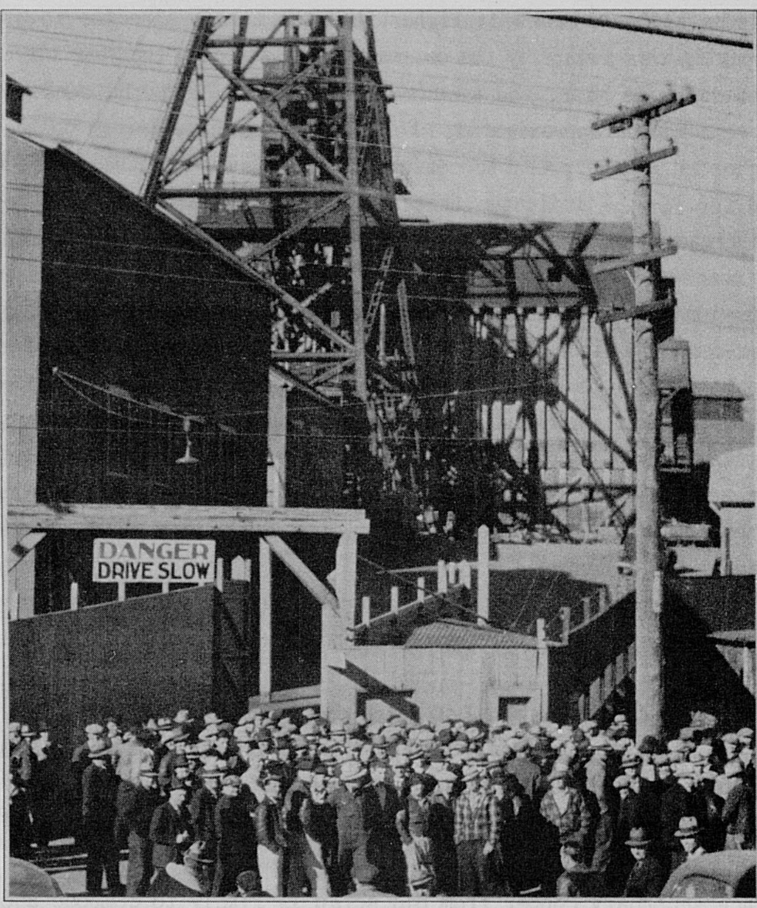

FIGURE 49.– MINERS "RUSTLING" JOBS AT A COPPER MINE IN MONTANA
The men at the mine gate are waiting to be interviewed by the company official who does the hiring at this operation.

EMPLOYMENT OPPORTUNITIES[18]

With mine production of copper estimated at 1,000,000 tons and output per man at 38 pounds per hour in 1947, copper mining will require about 52,600,000 man-hours in that year. This is equivalent to 21,900 men employed an average of 300 days per year and 8 hours per day. On the basis of a 6-hour instead of the prevailing 8-hour shift the number of men employed would be about 29,200. This estimate of employment represents a decline of 45 percent from the 1929 level of man-hours of employment,

[18]It should be noted that the estimates of employment, consumption, and production are based on trends and hence the figures for a specific future year, such as 1947, may be above or below the actual figure in that year. However, the fact that it is not possible to make a highly accurate forecast with reference to a specified time period in the future does not detract from the value of the trend estimates. From the viewpoint of this study, the significance of the estimated figures is that they indicate the probable magnitude and the general direction of future production and employment in the copper-mining industry.

compared with a drop of 37 percent between 1916 and 1929. The production in 1916 and 1929 and the estimated output in 1947, it is interesting to observe, are practically the same.

Over a longer period, the forecasting of employment trends becomes more difficult and questionable, for shifts in employment centers are likely to be effected by possible future discoveries of new deposits. Disregarding the possibility of discoveries, however, it seems rather certain that areas having large deposits of disseminated ores, which comprise approximately 85 percent of the total estimated reserves in the United States, will play a more important role in providing employment opportunities, even though the high output per worker places a definite limitation on the amount of employment that can be provided by mines in such areas. Arizona will become even more important than it is now as a source of employment in copper mining. Utah, Nevada, and New Mexico will also play a more prominent part. The importance of Montana and Michigan as sources of employment will, on the other hand, continue to diminish.

APPENDIX

Table A-1.- PRODUCTION, EMPLOYMENT, AND OUTPUT PER WORKER AT COPPER MINES, 1880-1936[a]

Year	Production		Employment[b]			Output of copper ore and tailings per -		Output of copper recoverable per -	
	Copper ore and tailings,[c] sold or treated (short tons)	Copper recoverable (pounds)	Men	Man-shifts	Man-hours	Man-hour	Man-shift	Man-hour	Man-shift
1880	1,007,490	56,115,454	6,039	1,811,700[d]	n.a.	n.a.	0.556	n.a.	30.974
1889	3,322,742	220,569,438	9,750[e]	2,925,000[f]	n.a.	n.a.	1.136	n.a.	75.408
1902	11,780,064[g]	625,004,529	26,344[h]	7,903,200[i]	n.a.	n.a.	1.491	n.a.	79.082
1911	29,988,235	1,095,131,104	44,693	13,774,325	110,194,600	0.272	2.177	9.938	79.505
1912	35,656,414	1,215,956,054	51,776	15,831,250	126,650,000	0.282	2.252	9.601	76.807
1913	36,336,682	1,213,247,830	56,139	17,369,266	138,954,128	0.262	2.092	8.731	69.850
1914	35,175,541	1,127,258,546	44,686	12,845,058	102,760,464	0.342	2.738	10.970	87.758
1915	43,404,182	1,453,912,379	47,174	14,211,400	113,691,200	0.382	3.054	12.788	102.306
1916	57,863,365	1,969,403,226	61,228	18,932,142	151,457,136	0.382	3.056	13.003	104.024
1917	59,213,237	1,866,079,144	61,275	19,085,563	152,684,504	0.388	3.103	12.222	97.774
1918	63,004,076	1,884,054,874	59,447	19,104,178	152,833,424	0.412	3.298	12.328	98.620
1919	37,037,281	1,191,292,206	39,327	11,856,491	94,851,928	0.390	3.124	12.559	100.476
1920	38,143,870	1,201,686,812	35,254	11,182,119	89,456,952	0.426	3.411	13.433	107.465
1921	13,396,382	455,707,733	18,300	4,461,296	35,690,368	0.375	3.003	12.768	102.147
1922	26,893,247	947,299,105	25,739	7,504,983	60,039,864	0.448	3.583	15.778	126.223
1923	45,544,558	1,449,780,217	32,477	10,306,466	82,451,728	0.552	4.419	17.583	140.667
1924	49,272,382	1,585,020,296	32,477	10,227,645	81,821,160	0.602	4.818	19.372	154.974
1925	53,195,376	1,650,291,482	33,266	10,420,803	83,366,424	0.638	5.105	19.796	158.365
1926	57,280,775	1,690,042,707	32,723	10,511,974	84,095,792	0.681	5.449	20.097	160.773
1927	56,794,178	1,615,927,676	30,724	9,625,317	77,002,536	0.738	5.900	20.985	167.883
1928	62,097,903	1,774,119,686	30,561	9,900,647	79,205,176	0.784	6.272	22.399	179.192
1929	68,421,853	1,961,560,104	37,147	11,983,712	95,869,696	0.714	5.710	20.461	163.686
1930	47,381,509	1,385,168,759	27,692	8,250,237	66,001,896	0.718	5.743	20.987	167.894

1931	34,248,203	1,042,531,163	19,687	5,075,862	41,019,314	0.835	6.747	25.416	205.390
1932	12,320,194	464,856,952	9,555	2,290,669	18,608,421	0.662	5.378	24.981	202.935
1933	8,387,612	368,224,716	6,976	1,689,788	13,471,547	0.623	4.964	27.334	217.912
1934	11,723,638	457,646,425	8,084	1,840,798	14,726,617	0.796	6.369	31.076	248.613
1935	19,112,054	737,777,584	10,188	2,787,083	22,293,255	0.857	6.857	33.094	264.713
1936	38,514,245	1,203,201,887	14,102	4,354,689	34,900,287	1.104	8.844	34.475	276.300

[a]Data for ore production, copper production, and number of men employed for 1880, 1889, and 1902 from *Special Reports of the Census Office*, "Mines and Quarries: 1902" (U. S. Dept. Com. and Labor, Bur. Census, 1905), pp. 469, 504; ore production (including tailings) and copper production (including copper from copper ores, tailings, and mine-water precipitates) for 1911-36 from issues of *Mineral Resources of the United States* (U. S. Dept. Int., Geol. Survey and U. S. Dept. Com., Bur. Mines) and *Minerals Yearbook* (U. S. Depts. Com. and Int., Bur. Mines) and from reports of individual companies to the Bureau of Mines; number of men employed, man-shifts, and man-hours for 1911-36 (except for man-hours from 1911-30 which were computed by assuming an 8-hour shift) from *Metal-Mine Accidents in the United States* (U. S. Depts. Com. and Int., Bur. Mines).

It should be noted that the data for census years 1880, 1889, and 1902 are not strictly comparable. The figures for ore and copper production for 1889 exclude States other than Michigan, Montana, Arizona, and New Mexico, whereas those for 1880 and 1902 cover all States. (Copper-production figures for the entire United States for 1889 are reported by the census and also by the U. S. Geological Survey, but corresponding data for ore production and employment are not available. The reported mine output of copper in other States was negligible, amounting to about 1 percent of the total production.)

The employment data for 1889 and 1902 represent employment at mines and mills, whereas those for 1880 probably include the workers at some of the smelters which were located near the mills and whose employment could not be segregated from mill labor. This appears to be the major difference between the statistics for 1880 and those for later years.

The employment figures for 1911-36 represent only those workers employed at mines and related surface operations such as carpenter and machine shops; they exclude mills and ore-dressing plants. The Bureau of Mines has been compiling figures on employment at mills since 1916, but its compilations on mill labor from 1916 to 1932 are not comparable with its mine-labor statistics which are used in this report. Whereas its mine-employment data cover all mines regardless of size, its mill data embrace a select group of mills which employed not fewer than 25 men and from whom reliable reports could be obtained. The number of mills included and the coverage varied from year to year. Beginning in 1933 the Bureau's figures for employment at mills represent complete coverage. Under the circumstance it was deemed inadvisable to include mill employment from 1911 to 1936. It should be pointed out, however, that the inclusion of mill labor in the census years 1880, 1889, and 1902 and the exclusion of it from 1911 on do not have a significant effect on the comparability of the employment series. As reference to table A-4 will show, before the turn of the century most of the copper ore mined was of such a high grade that it went directly from the mine to the smelter. It was not until after 1902, with the exploitation of the porphyry deposits, that milling began to play an increasingly significant role in the copper industry. It is estimated that since 1916 copper mills and ore-dressing plants have been employing a fifth to a third as many workers as the mines. If the output per worker were computed on the basis of mine and mill labor, it would have been decreased by a proportion ranging from a sixth to a fourth.

[b]Employment statistics are not strictly comparable with those for production. They do not include workers who produced incidentally some copper ores at noncopper mines (the principal metal contents of whose ores were other than copper). On the other hand, they include workers at copper mines who incidentally produced some ores whose principal metal contents were other than copper. Statistical difficulties due to these factors are unimportant and can be ignored.

[c]Ores of which the major metal is copper are included even though produced in mines engaged primarily in the extraction of other metals. However, such mines produced a small proportion of the total copper-ore output.

[d]Estimated on basis of 300-day working year, as the average number of days is not reported by the census.

[e]Includes foremen.

[f]Computed by multiplying the average number of men employed by 300, the average number of working days in the year. The average number of working days was computed from data in the *Eleventh Census of the United States: 1890*, "Report on Mineral Industries" (U. S. Dept. Int., Census Office, 1892), p. 156; it represents the weighted average of the average number of men and boys employed above and below ground and the average number of days they were employed.

[g]Quantity of ore mined.

[h]Includes 337 foremen working underground in addition to 26,007 wage earners.

[i]Computed by multiplying the average number of men by 300, the average number of working days assumed in the census. See *Special Reports of the Census Office*, "Mines and Quarries: 1902," pp. 90-1 for detailed statement.

n.a. Data not available.

Table A-2.- PRODUCTION OF COPPER ORE AND COPPER, BY MINING METHOD, 1880-1936

Year	Total[a]		Open-cut[b]		Caving[c]		Mines with natural supports[d]		Mines with artificial supports[e]	
	Ore and tailings, sold or treated (short tons)	Copper recoverable (pounds)	Ore and tailings, sold or treated (short tons)	Copper recoverable (pounds)	Ore and tailings, sold or treated (short tons)	Copper recoverable (pounds)	Ore and tailings, sold or treated (short tons)	Copper recoverable (pounds)	Ore and tailings, sold or treated (short tons)	Copper recoverable (pounds)
1880	1,007,490	60,480,000	n.a.	n.a.	n.a.	n.a.	938,960	50,736,960	68,530	9,743,040
1881	n.a.	71,680,000	n.a.	n.a.	n.a.	n.a.	n.a.	54,073,120	n.a.	17,606,880
1882	n.a.	90,646,232	n.a.	n.a.	n.a.	n.a.	n.a.	57,809,460	n.a.	32,836,772
1883	n.a.	115,526,053	n.a.	n.a.	n.a.	n.a.	n.a.	61,303,266	n.a.	54,222,787
1884	n.a.	144,946,653	n.a.	n.a.	n.a.	n.a.	n.a.	70,276,035	n.a.	74,670,618
1885	n.a.	165,875,483	n.a.	n.a.	n.a.	n.a.	n.a.	72,657,298	n.a.	93,218,185
1886	n.a.	157,763,043	n.a.	n.a.	n.a.	n.a.	n.a.	81,348,670	n.a.	76,414,373
1887	n.a.	181,477,331	n.a.	n.a.	n.a.	n.a.	n.a.	77,628,697	n.a.	103,848,634
1888	n.a.	226,361,466	n.a.	n.a.	n.a.	n.a.	n.a.	88,092,055	n.a.	138,269,411
1889	3,322,742	226,775,962	n.a.	n.a.	n.a.	n.a.	2,433,733	88,483,670	889,009	138,292,292
1890	n.a.	259,763,092	n.a.	n.a.	n.a.	n.a.	n.a.	101,520,867	n.a.	158,242,225
1891	n.a.	284,121,764	n.a.	n.a.	n.a.	n.a.	n.a.	117,766,939	n.a.	166,354,825
1892	n.a.	344,998,679	n.a.	n.a.	n.a.	n.a.	n.a.	126,405,404	n.a.	218,593,275
1893	n.a.	329,354,398	n.a.	n.a.	n.a.	n.a.	n.a.	112,881,127	n.a.	216,473,271
1894	n.a.	354,188,374	n.a.	n.a.	n.a.	n.a.	n.a.	114,428,870	n.a.	239,759,504
1895	n.a.	380,613,404	n.a.	n.a.	n.a.	n.a.	n.a.	130,974,995	n.a.	249,638,409
1896	n.a.	460,061,430	n.a.	n.a.	n.a.	n.a.	n.a.	144,214,306	n.a.	315,847,124
1897	n.a.	494,078,274	n.a.	n.a.	n.a.	n.a.	n.a.	157,453.108	n.a.	336,625,166
1898	n.a.	526,512,987	n.a.	n.a.	n.a.	n.a.	n.a.	176,684,257	n.a.	349,828,730
1899	n.a.	568,666,921	n.a.	n.a.	n.a.	n.a.	n.a.	173,732,235	n.a.	394,934,686
1900	n.a.	606,117,166	n.a.	n.a.	n.a.	n.a.	n.a.	175,602,475	n.a.	430,514,691
1901	n.a.	602,072,519	n.a.	n.a.	n.a.	n.a.	n.a.	192,481,034	n.a.	409,591,485
1902	11,780,064	659,508,644	n.a.	n.a.	n.a.	n.a.	5,878,331	184,594,082	5,901,733	474,914,562
1903	n.a.	698,044,517	n.a.	n.a.	n.a.	n.a.	n.a.	186,445,855	n.a.	511,598,662
1904	n.a.	812,537,267	n.a.	n.a.	n.a.	n.a.	n.a.	210,703,884	n.a.	601,833,383
1905	n.a.	888,784,267	n.a.	n.a.	n.a.	n.a.	n.a.	218,323,856	n.a.	670,460,411
1906	18,000,000	916,971,387	n.a.	n.a.	54,451	1,590,280	n.a.	244,804,992	n.a.	670,576,115
1907	20,039,484	847,151,015	640,383	15,216,772	1,241,174	49,412,251	9,122,037	239,517,854	9,035,890	543,004,138
1908	22,290,886	942,849,264	2,473,294	61,123,748	1,190,590	58,823,512	9,928,688	277,286,285	8,698,314	545,615,719
1909	27,932,618	1,109,922,340	4,931,078	117,004,003	1,233,938	54,754,564	11,090,903	307,872,285	10,676,699	630,291,488

1910	28,497,238	1,073,436,846	6,780,121	152,979,030	1,698,041	61,710,764	10,430,022	274,590,904	9,589,054	584,156,148
1911	29,988,235	1,095,131,104	7,396,823	164,277,362	3,069,170	94,016,541	10,385,046	264,479,886	9,137,196	572,357,315
1912	35,656,414	1,215,956,054	9,041,215	177,509,362	5,105,320	151,708,571	10,832,190	326,570,450	10,677,689	560,167,671
1913	36,336,682	1,213,247,830	12,353,453	224,858,013	6,240,272	166,630,247	7,412,396	294,459,315	10,330,561	527,300,255
1914	35,175,541	1,127,258,546	10,910,112	218,339,016	5,966,540	165,464,860	9,351,009	280,990,970	8,947,880	462,463,700
1915	43,404,182	1,453,912,379	13,865,882	278,703,126	6,849,434	181,216,564	12,357,117	346,562,405	10,331,749	647,430,284
1916	57,863,365	1,969,403,226	17,438,505	339,112,435	14,057,132	348,379,230	12,855,363	345,230,791	13,512,365	936,680,770
1917	59,213,237	1,866,079,144	20,059,859	359,275,077	13,298,640	327,500,952	12,213,040	340,830,316	13,641,698	838,472,799
1918	63,004,076	1,884,054,874	20,822,032	366,897,693	15,657,751	375,484,887	11,628,247	299,765,218	14,896,046	841,907,076
1919	37,037,281	1,191,292,206	10,351,224	211,824,818	10,006,075	257,050,476	8,521,374	231,115,455	8,158,608	491,301,457
1920	38,143,870	1,201,686,812	11,340,071	226,253,315	10,693,009	242,939,997	8,032,474	228,178,192	8,078,316	504,315,308
1921	13,396,382	455,707,733	3,028,104	65,983,473	3,758,421	94,775,165	3,796,021	133,734,376	2,813,836	161,214,719
1922	26,893,247	947,299,105	7,809,343	181,952,133	7,648,916	196,129,893	5,526,640	197,713,310	5,908,348	371,503,769
1923	45,544,558	1,449,780,217	19,319,330	413,761,783	11,682,562	285,376,480	6,661,290	242,994,163	7,881,376	507,647,791
1924	49,272,382	1,585,020,296	22,389,783	469,772,877	12,813,025	302,467,317	6,903,866	249,464,847	7,165,708	563,315,255
1925	53,195,376	1,650,291,482	23,430,259	474,966,934	13,632,635	305,193,829	8,880,248	268,100,854	7,252,234	602,029,865
1926	57,280,775	1,690,042,707	25,953,658	546,762,300	15,197,416	309,415,626	9,312,337	272,775,914	6,817,364	561,088,867
1927	56,794,178	1,615,927,676	24,856,799	527,200,263	15,311,769	319,622,903	10,035,332	268,215,976	6,590,278	500,888,534
1928	62,097,903	1,774,119,686	29,364,499	634,625,700	16,190,664	332,903,698	9,652,087	262,437,295	6,890,653	544,152,993
1929	68,421,853	1,961,560,104	31,498,868	682,356,603	18,652,457	352,986,218	9,970,691	278,280,908	8,299,837	647,936,375
1930	47,381,509	1,385,168,759	17,817,809	381,724,370	14,798,423	282,558,733	8,661,772	249,228,674	6,103,505	471,656,982
1931	34,248,203	1,042,531,163	14,441,663	285,557,534	10,964,606	210,984,575	4,422,274	163,575,965	4,419,660	382,413,089
1932	12,320,194	464,856,952	5,706,787	113,188,720	3,678,694	77,087,497	1,197,385	63,010,557	1,737,328	211,570,178
1933	8,387,612	368,224,716	5,578,473	110,349,669	375,149	12,326,457	851,581	47,243,920	1,582,409	198,304,670
1934	11,723,638	457,646,425	8,097,472	159,549,245	647,203	27,273,069	1,072,409	51,893,514	1,906,554	218,930,597
1935	19,112,054	737,777,584	12,978,778	299,333,801	1,451,634	47,217,851	1,900,447	81,209,177	2,781,195	310,016,755
1936	38,514,245	1,203,201,887	24,068,571	526,596,388	5,919,136	122,634,628	4,187,450	143,285,244	4,339,088	410,685,627

[a]Data for ore production for 1880, 1889, and 1902 from *Special Reports of the Census Office*, "Mines and Quarries: 1902" (U. S. Dept. Com. and Labor, Bur. Census, 1905), p. 469; ore production for 1906-36 and copper production for all years from issues of *Mineral Resources of the United States* (U. S. Dept. Int., Geol. Survey and U. S. Dept. Com., Bur. Mines) and *Minerals Yearbook* (U. S. Depts. Com. and Int., Bur. Mines) and from reports of individual companies to the Bureau of Mines. Copper-production figures for 1880-1905 represent smelter output (for difference between smelter output and mine production see *Mineral Resources of the United States, 1930* [U. S. Dept. Com., Bur. Mines, 1933], pt. I, pp. 704-5); those for 1906-36, mine production (production of copper from ores, tailings, and mine-water precipitates).

[b]Compiled from data reported annually by individual companies to the Bureau of Mines and from published annual reports of the individual companies to the stockholders. Includes the production of the Utah Copper Co., 1907-36; Nevada Consolidated Copper Co., Chino mine, 1910-34 (idle in 1935 and 1936); Nevada Consolidated Copper Co., Copper Flat pits, 1910-36; New Cornelia mine, 1916-32 and 1934-36 (idle in 1933); United Verde mine, 1920, 1922-31, and 1935-36 (no open-cut production in 1921 and 1932-34); and Sacramento Hill mine, 1917 and 1919-29 (no open-cut production in 1918; open-cut operations discontinued at the beginning of 1930). Figures for ore mined rather than for ore sold or treated were used for United Verde, Copper Flat, and Sacramento Hill. The differences between "ore mined" and "ore sold or treated" are negligible.

[c]Compiled from the reports of individual companies to the Bureau of Mines and annual reports of the companies to the stockholders. Includes the following mines: Ohio Copper, Longfellow group, Morenci, Miami, Ray, Inspiration, Consolidated Copper Co.'s mine, Burro Mountain group, and Ruth.

[d]Includes mines employing predominantly open-stope and shrinkage methods of mining. For the period 1880-1916 the figures include the output from Michigan, Tennessee, California, North Carolina, Idaho, and Alaska, except for the exclusion of production of the Copper Range Co. in Michigan for the period 1902-16. (Data for the Baltic, Champion, and Trimountain mines of the Copper Range Co. are from B. S. Butler, W. S. Burbank, and Others, *The Copper Deposits of Michigan* [U. S. Dept. Int., Geol. Survey, Professional Paper No. 144, 1929], pp. 79, 82, 97.) Ore-production data for 1880, 1889, and 1902 are from *Tenth Census of the United States: 1880*, Vol. XV, "Report on the Mining Industries" (U. S. Dept. Int., Census Office, 1886), p. 798; *Eleventh Census of the United States: 1890*, "Report on Mineral Industries" (U. S. Dept. Int., Census Office, 1892), p. 155; and *Special Reports of the Census Office*, "Mines and Quarries: 1902," pp. 480, 483; for 1907-16 from issues of *Mineral Resources of the United States*; and for 1917-36 from reports of individual companies to the Bureau of Mines and classified according to their predominating method of mining. Copper-production data for 1880-1916 are from issues of *Mineral Resources of the United States* and for 1917-36 from reports of individual companies to the Bureau of Mines. Mines were classified according to their predominating method of mining. Figures for 1880-1905 represent smelter output; those for 1906-36, mine output production.

[e]Includes mines employing predominantly cut-and-fill and square-set methods of mining and, except for the production from caving mines, represents mainly the production from the underground mines of Arizona, Montana, Nevada, New Mexico, and Utah. Figures were obtained by subtracting the production of open-cut mines, caving mines, and mines with natural supports from the total-production figures.

n.a. Data not available.

Table A-3.- PRODUCTION OF COPPER, BY PRINCIPAL DISTRICT, 1880-1936[a]

(Thousands of pounds)

Year	Total	Lake Superior	Butte[b]	Bingham[c]	Central (including Santa Rita)	Morenci-Metcalf[d]	Bisbee (Warren)[d]	Globe-Miami	Yavapai County (mostly Jerome)[d e]	Ely (Robinson)[f]	Ray (Mineral Creek)[d]	Ajo[d]	Others[g]
1880	60,480	49,737	1,212	50	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	9,481
1881	71,680	53,573	1,450	45	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	16,612
1882	90,646	56,983	9,058	50	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	24,555
1883	115,526	59,702	24,664	50	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	31,110
1884	144,947	69,353	43,093	95	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	32,406
1885	165,875	72,148	67,798	100	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	25,829
1886	157,763	80,918	57,612	240	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	18,993
1887	181,477	76,029	78,700	500	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	26,248
1888	226,361	86,472	97,898	650	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	41,341
1889	226,776	88,176	98,222	251	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	40,127
1890	259,763	101,410	112,981	175	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	45,197
1891	284,122	114,223	112,063	595	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	57,241
1892	344,999	123,198	163,206	497	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	58,098
1893	329,354	112,605	155,209	205	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	61,335
1894	354,188	114,309	183,073	275	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	56,531
1895	380,613	129,331	190,172	611	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	60,499
1896	460,061	143,524	221,918	500	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	94,119
1897	494,078	145,282	232,728	1,419	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	114,649
1898	526,513	158,492	207,435	2,284	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	158,302
1899	568,667	147,400	225,144	4,145	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	191,978
1900	606,117	145,461	270,754	6,197	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	183,705
1901	602,073	156,289	230,624	14,422	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	200,738
1902	659,509	170,609	289,349	14,760	7,001	52,433	37,930	9,107	19,407	n.a.	n.a.	30	58,883
1903	698,045	192,401	272,729	17,280	5,959	51,032	62,802	7,740	23,772	n.a.	n.a.	n.a.	64,330
1904	812,537	208,309	290,033	30,629	3,264	59,535	90,346	14,678	30,454	n.a.	n.a.	n.a.	85,289
1905	888,784	230,288	304,308	39,220	3,179	53,898	107,928	24,992	33,418	n.a.	6	n.a.	91,547
1906	916,971	224,572	289,780	39,424	3,075	59,177	129,163	26,082	38,839	n.a.	n.a.	n.a.	106,859
1907	847,151	217,767	218,837	45,432	4,995	62,580	109,332	35,102	33,016	12	620	39	119,419
1908	956,841	223,287	250,151	71,156	3,995	77,146	126,409	37,321	36,183	14,797	14	n.a.	116,382
1909	1,126,521	234,137	311,324	92,560	3,349	71,825	139,337	39,451	36,695	57,482	637	143	139,581

1910	1,088,237	222,683	284,265	113,725	2,680	72,878	140,235	31,323	38,664	63,914	n.a.	98	117,772
1911	1,114,764	219,840	272,271	129,996	1,413	70,926	128,767	49,010	33,168	67,034	15,722	100	126,517
1912	1,249,095	218,138	308,223	116,622	29,513	76,848	138,703	61,412	31,566	67,374	34,674	100	165,922
1913	1,235,570	135,853	285,682	144,920	51,696	69,467	158,006	71,250	35,335	71,917	54,148	n.a.	157,296
1914	1,148,431	164,344	232,189	141,925	55,223	65,559	151,613	72,895	32,448	51,090	57,004	n.a.	124,141
1915	1,488,072	265,283	266,006	176,593	66,581	51,096	164,667	102,914	50,267	64,662	60,339	20	219,644
1916	2,005,875	273,693	349,538	223,620	74,229	75,977	193,696	220,039	103,000	93,045	74,984	1,000	323,054
1917	1,895,434	255,710	270,593	225,412	82,200	53,072	167,599	169,417	145,984	93,627	88,583	20,202	323,035
1918	1,910,023	226,794	321,107	211,195	77,547	71,869	160,456	201,806	133,781	92,420	86,099	49,950	276,999
1919	1,212,334	178,826	169,053	116,697	40,359	44,868	122,311	171,106	72,333	47,871	48,511	39,509	160,890
1920	1,224,550	154,695	176,404	110,670	44,400	35,748	118,448	176,215	107,084	48,348	50,062	40,104	162,372
1921	466,191	86,370	47,958	28,011	9,138	11,422	37,249	76,488	26,568	10,283	10,110	20,198	102,396
1922	964,584	121,712	165,374	91,762	28,408	14,177	79,591	170,035	74,794	22,615	29,115	26,613	140,388
1923	1,477,740	138,304	223,043	216,037	54,375	28,736	123,884	198,426	140,425	65,928	63,704	38,368	186,510
1924	1,606,165	135,663	247,365	236,181	66,824	41,049	140,630	190,452	143,348	73,279	69,590	63,884	197,900
1925	1,678,118	155,157	266,637	227,612	68,812	59,464	145,433	175,272	154,017	78,627	74,691	69,262	203,134
1926	1,725,276	175,382	252,991	248,975	75,219	63,970	140,668	175,386	155,368	95,009	69,925	82,312	190,071
1927	1,649,959	177,538	221,312	249,919	66,971	59,760	138,595	171,167	141,642	107,441	62,860	72,933	179,821
1928	1,809,797	178,443	247,067	287,685	80,239	54,054	161,492	160,430	165,297	141,969	68,661	77,995	186,465
1929	1,995,110	186,402	296,316	311,892	87,446	57,095	186,130	191,596	208,172	130,755	66,287	71,000	202,019
1930	1,410,147	169,381	195,472	175,070	57,243	43,144	127,900	158,120	117,690	105,385	36,118	50,474	174,150
1931	1,057,749	118,059	184,362	147,706	56,317	38,344	95,328	126,444	44,576	71,334	24,438	41,200	109,641
1932	476,221	54,396	84,608	62,469	26,513	23,861	47,405	28,448	35,809	30,885	14,404	10,000	57,423
1933	381,285	46,853	65,239	71,636	25,142	7	55,796	259	33,258	28,189	2,752	n.a.	52,154
1934	474,810	48,216	62,856	83,586	21,790	11	71,111	14,322	26,398	40,934	n.a.	n.a.	105,586
1935	760,980	64,109	153,929	126,121	3,095	2	64,563	37,359	76,172	65,630	2	67,120	102,878
1936	1,229,031	95,968	218,008	248,906	4,427	12	79,684	111,336	100,654	115,161	14	96,040	158,821

[a]Except as noted, data are from issues of *Mineral Resources of the United States* (U. S. Dept. Int., Geol. Survey and U. S. Dept. Com., Bur. Mines) and *Minerals Yearbook* (U. S. Depts. Com. and Int., Bur. Mines) and from reports of individual companies to the Bureau of Mines. All figures for 1880-1901, those for the total and Lake Superior for 1902-5, and those for Butte for 1902-3 represent smelter output. All other figures represent mine production of copper from all sources (including old slags, smelter cleanings, and precipitates) except scrap.

[b]Figures for 1904-36 represent production from Silver Bow County.

[c]Data for 1880-1909 from B. S. Butler, G. F. Loughlin, and Others, *The Ore Deposits of Utah* (U. S. Dept. Int., Geol. Survey, Professional Paper No. 111, 1920), pp. 345-6.

[d]Data for Morenci-Metcalf, 1910-34; Bisbee or Warren, 1902-9; Yavapai (mostly Jerome), 1908-23; Ray (Mineral Creek), 1911-33; and Ajo, 1909-33 are from M. J. Elsing and R. E. S. Heineman, "Arizona Metal Production" (Ariz. Bur. Mines, Econ. Series No. 19, Bull. No. 140), *University of Arizona Bulletin*, Vol. VII, No. 2 (Feb. 15, 1936), pp. 48, 69, 73, 78, 86.

[e]Data for 1902-23 represent mine production from Jerome district; those for 1924-36, Yavapai County.

[f]Data for 1907-9 from F. C. Lincoln, *Mining Districts and Mineral Resources of Nevada* (Reno, Nevada: Nevada Newsletter Publishing Co., 1923), p. 247.

[g]Obtained by subtracting from the total the sum of the figures for the specified districts.

n.a. Data not available.

Table A-4.- PRODUCTION AND YIELD OF DIRECT-SMELTING AND MILLING COPPER ORES, 1880-1936[a]

Year	Total[b]		Smelting ores		Milling ores	
	Short tons[c]	Yield of copper, percent	Short tons	Yield of copper, percent	Short tons	Yield of copper, percent
1880	1,007,490	3.00	n.a.	n.a.	n.a.	n.a.
1889	3,322,742[d]	3.32[d]	n.a.	n.a.	n.a.	n.a.
1902	11,464,868	2.73	4,905,471	4.19	6,559,397	1.68
1906	18,000,000	2.50	3,278,000	n.a.	14,722,000	n.a.
1907	20,253,000	2.11	3,957,500	4.05	16,295,500	1.41
1908	22,290,886[e]	2.07	4,386,689	4.27	17,760,522	1.49
1909	27,932,618	1.98	5,267,707	4.15	22,664,911	1.47
1910	28,497,238	1.88	5,001,394	4.14	23,495,844	1.40
1911	29,988,235	1.82	4,355,650	4.66	25,632,585	1.34
1912	35,656,414	1.71	5,014,018	4.45	30,642,396	1.26
1913	36,336,682	1.67	5,290,310	4.22	31,046,372	1.22
1914	35,175,541[f]	1.60	4,597,398	4.23	30,478,327	1.20
1915	43,404,182	1.66	5,433,907	4.60	37,970,275	1.26
1916	57,863,365	1.70	6,928,010	4.72	50,935,355	1.28
1917	58,482,694	1.60	7,438,068	4.39	51,044,626	1.19
1918	62,289,069	1.51	6,223,674	4.51	53,937,796	1.18
1919	36,121,622	1.65	3,465,880	4.64	30,769,966	1.35
1920	36,765,370	1.63	3,200,509	4.89	31,348,340	1.34
1921	13,396,382	1.70	1,235,602	4.77	11,022,544	1.44
1922	26,893,247	1.74	2,278,047	5.36	23,258,970	1.43
1923	45,519,317	1.58	3,496,728	5.12	40,209,754	1.29
1924	49,178,315	1.59	3,554,915	5.08	44,427,264	1.33
1925	53,103,014	1.54	3,876,733	4.90	48,186,769	1.28
1926	57,181,894	1.46	3,767,947	4.75	52,083,784	1.24
1927	56,725,460	1.41	3,407,610	4.67	49,179,035	1.23
1928	62,097,132	1.41	3,766,368	4.44	54,214,485	1.24
1929	68,421,853[g]	1.41	4,235,192	4.60	59,727,536[g]	1.22
1930	47,381,509[g]	1.43	2,983,912	4.57	41,327,237[g]	1.23
1931	34,050,961[g]	1.50	1,519,915	5.38	30,056,857[g]	1.33
1932	12,320,194[g]	1.83	758,623	6.98	10,964,749[g]	1.51
1933	8,387,612[g]	2.11	872,033	6.30	7,475,988[g]	1.63
1934	11,723,638[g]	1.92	977,096	6.21	10,681,967[g]	1.53
1935	19,112,054[g]	1.89	1,612,200[g]	5.42	17,065,419[g] [h]	1.57
1936	38,514,245[g]	1.54	2,388,635[g]	5.05	36,116,692[g]	1.31

[a]Data for 1880, 1889, and 1902 from *Special Reports of the Census Office*, "Mines and Quarries: 1902" (U. S. Dept. Com. and Labor, Bur. Census, 1905), pp. 469, 479; yield of copper ore is the computed percent that copper production is of ore production. Data for 1906-24 from *Mineral Resources of the United States, 1919* (U. S. Dept. Int., Geol. Survey, 1922), pt. I, pp. 556-7; for 1917-24 from same, *1925* (U. S. Dept. Com., Bur. Mines, 1928), pt. I, p. 369; for 1925-32 from *Minerals Yearbook, 1936* (U. S. Dept. Int., Bur. Mines, 1936), p. 118; and for 1933-36 from same, *1939* (1939), p. 102.

[b]These totals will not in all cases agree with those in table A-1 because the latter includes data obtained from reports of individual companies to the Bureau of Mines.

[c]Includes ores that were leached and copper-bearing materials not otherwise reported.

[d]Represents production from Michigan, Montana, Arizona, and New Mexico only; see table A-1, ftn. a.

[e]Includes 143,675 tons of slag smelted.

[f]Includes 99,816 tons of slag smelted and ore leached.

[g]Includes tailings, etc.

[h]Excludes small quantities from California, data for which the Bureau of Mines is not at liberty to publish.

n.a. Data not available.

Table A-5.- PRODUCTION OF COPPER, BY PRINCIPAL PRODUCING STATE, 1880-1936[a]

(Pounds)

Year	Total	Michigan	Montana	Arizona	Utah	New Mexico	Nevada	Others[b]
1880	60,480,000	49,736,960	1,212,000	2,000,000	86,000	4,000[c]	135,000[c]	7,306,040
1881	71,680,000	53,573,120	1,450,000	8,000,000	386,000	500,000	n.a.	7,770,880
1882	90,646,232	56,982,765	9,058,284	17,984,415	605,880	869,498	350,000	4,795,390
1883	115,526,053	59,702,404	24,664,346	23,874,963	341,885	823,511	288,077	5,830,867
1884	144,946,653	69,353,202	43,093,054	26,734,345	265,526	59,450	100,000	5,341,076
1885	165,875,483	72,147,889	67,797,864	22,706,366	126,199	79,839	8,871	3,008,455
1886	157,763,043	80,918,460	57,611,621	15,657,035	500,000	558,385	50,000	2,467,542
1887	181,477,331	76,028,697	78,699,677	17,720,462	2,500,000	283,664	n.a.	6,244,831
1888	226,361,466	86,472,034	97,897,968	31,797,300	2,131,047	1,631,271	50,000	6,381,846
1889	226,775,962	88,175,675	98,222,444	31,586,185	65,467	3,686,137	26,420	5,013,634
1890	259,763,092	101,410,277	112,980,896	34,796,689	1,006,636	850,034	n.a.	8,718,560
1891	284,121,764	114,222,709	112,063,320	39,873,279	1,562,098	1,233,197	n.a.	15,167,161
1892	344,998,679	123,198,460	163,206,128	38,436,099	2,209,428	1,188,796	n.a.	16,759,768
1893	329,354,398	112,605,078	155,209,133	43,902,824	1,135,330	280,742	20,000	16,201,291
1894	354,188,374	114,308,870	183,072,756	44,514,894	1,147,570	31,884	n.a.	11,112,400
1895	380,613,404	129,330,749	190,172,150	47,953,553	2,184,708	143,719	n.a.	10,828,525
1896	460,061,430	143,524,069	221,918,179	72,934,927	3,502,012	2,701,664	n.a.	15,480,579
1897	494,078,274	145,282,059	232,728,479	81,530,735	3,919,010	701,892	n.a.	29,916,099
1898	526,512,987	158,491,703	207,434,550	111,158,246	3,750,000	1,592,371	437,396	43,648,721
1899	568,666,921	147,400,338	225,143,875	133,054,860	9,584,746	3,935,441	556,775	48,990,886
1900	606,117,166	145,461,498	270,753,636	118,317,764	18,354,726	4,169,400	407,535	48,652,607
1901	602,072,519	156,289,481	230,623,925	130,778,611	20,116,979	9,629,884	593,608	54,040,031
1902	659,508,644	170,609,228	289,349,483	119,944,944	23,939,901	6,614,961	164,301	48,885,826
1903	698,044,517	192,400,577	272,729,056	147,648,271	38,302,602	7,300,832	150,000	39,513,179
1904	812,537,267	208,309,130	298,414,804	191,602,958	47,062,889	5,368,666	n.a.	61,778,820
1905	888,784,267	230,287,992	314,750,620	226,854,461	54,083,506	5,334,192	413,292	57,060,204
1906	916,971,387	224,572,310	290,700,975	266,831,864	56,593,576	7,028,670	1,625,985	69,618,007
1907	847,151,015	217,767,232	220,108,792	254,879,489	64,256,884	10,990,015	1,782,571	77,366,032
1908	956,840,578	223,286,700	251,667,795	285,858,133	86,848,812	6,112,630	15,598,788	87,467,720
1909	1,126,521,126	234,136,529	312,058,011	303,899,461	108,947,811	5,393,146	57,976,477	104,109,691

See footnotes at end of table.

Table A-5.- PRODUCTION OF COPPER, BY PRINCIPAL PRODUCING STATE, 1880-1936[a] - *Continued*

(Pounds)

Year	Total	Michigan	Montana	Arizona	Utah	New Mexico	Nevada	Others[b]
1910	1,088,237,432	222,683,461	284,808,553	297,491,151	127,597,072	4,614,386	64,359,398	86,683,411
1911	1,114,764,197	219,840,201	272,847,705	306,141,538	146,960,827	4,057,040	67,377,518	97,539,368
1912	1,249,094,891	218,138,408	309,738,873	365,038,649	137,307,485	34,030,964	86,477,494	98,363,018
1913	1,235,569,727	135,853,409	287,828,699	407,923,402	161,445,962	56,308,706	90,693,751	95,515,798
1914	1,148,431,437	164,344,058	233,229,640	393,017,400	152,034,002	59,307,925	60,986,450	85,511,962
1915	1,488,071,528	265,283,378	267,231,014	459,972,295	187,671,188	76,788,366	68,636,370	162,488,917
1916	2,005,875,312	273,692,525	352,928,373	721,833,169	240,275,222	92,747,289	105,116,813	219,281,921
1917	1,895,434,349	255,710,128	274,462,574	712,166,891	246,674,153	105,568,000	122,794,704	178,057,899
1918	1,910,022,841	226,794,139	323,174,850	764,855,874	227,169,630	98,264,563	116,316,441	153,447,344
1919	1,212,334,041	178,826,486	169,981,288	538,100,844	124,061,807	51,150,541	52,331,175	97,881,900
1920	1,224,550,151	154,695,073	177,059,260	558,256,302	116,931,238	54,400,691	50,559,763	112,647,824
1921	466,190,853	86,370,028	48,098,730	185,034,194	30,891,403	14,267,338	10,961,491	90,567,669
1922	964,583,894	121,712,365	165,754,442	400,043,128	97,193,850	31,937,207	23,133,091	124,809,811
1923	1,477,739,709	138,304,080	224,125,264	618,928,602	222,393,572	61,356,802	67,204,282	145,427,107
1924	1,606,165,018	135,662,605	249,152,062	677,752,013	242,138,165	74,691,436	73,805,323	152,963,414
1925	1,678,117,747	155,157,488	268,910,847	713,355,129	236,486,540	76,427,825	79,300,224	148,479,694
1926	1,725,275,931	175,381,565	255,372,862	723,296,051	257,464,482	81,642,379	101,827,937	130,290,655
1927	1,649,959,370	177,537,775	223,492,693	682,190,547	256,933,278	74,251,863	120,259,276	115,293,938
1928	1,809,796,907	178,442,704	248,262,027	732,276,803	293,235,039	89,854,646	158,876,883	108,848,805
1929	1,995,110,398	186,402,218	297,725,973	830,628,411	318,282,523	97,717,262	140,138,809	124,215,202
1930	1,410,147,374	169,381,413	196,187,523	576,190,607	180,526,423	65,150,000	109,203,512	113,507,896
1931	1,057,749,350	118,059,491	184,555,735	401,344,909	151,236,505	61,503,100	72,634,497	68,415,113
1932	476,221,076	54,396,108	84,847,349	182,491,825	64,964,111	28,419,000	31,487,606	29,615,077
1933	381,285,194	46,853,130	65,476,375	114,041,781	73,583,130	26,947,000	28,489,610	25,894,168
1934	474,810,458	48,215,859	63,265,000	178,082,213	86,024,925	23,630,000	41,611,119	33,981,342
1935	760,979,802	64,108,689	154,957,470	278,029,289	129,515,217	4,505,000	74,266,000	55,598,137
1936	1,229,030,719	95,968,019	219,088,000	422,550,000	252,434,000	6,332,000	141,392,000	91,266,700

[a]Except as noted, data are from issues of *Mineral Resources of the United States* (U. S. Dept. Int., Geol. Survey and U. S. Dept. Com., Bur. Mines) and *Minerals Yearbook* (U. S. Depts. Com. and Int., Bur. Mines). Data for 1880-1905 represent smelter output; those for 1906-36, mine production of copper from all sources (including old slags, smelter cleanings, and precipitates) except scrap.

[b]Obtained by subtracting from the total the sum of the figures for the specified States.

[c]Data for Nevada and New Mexico for 1880 are from *Tenth Census of the United States: 1880*, Vol. XV, "Report on the Mining Industries" (U. S. Dept. Int., Census Office, 1886), p. 800.

n.a. Data not available.

Table A-6.- PRODUCTION OF COPPER ORE, BY PRINCIPAL PRODUCING STATE, 1880-1936[a]

Table A-6.- PRODUCTION OF COPPER ORE, BY PRINCIPAL PRODUCING STATE, 1880-1936[a]

(Short tons)

Year	Total[b]	Michigan	Montana	Arizona	Utah	New Mexico	Nevada	Other[c]
1880	1,007,490	938,960	523	7,650	n.a.	13	357	59,987
1889	3,322,742	2,433,733	698,837	155,586	n.a.	34,586	n.a.	n.a.
1902	11,780,064	6,247,352	3,428,860	1,210,301	246,012	46,993	n.a.	600,546
1906	18,000,000	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
1907	20,039,484	9,892,214	3,719,600	3,191,125	1,793,084	164,849	11,100	1,267,512
1908	22,290,886	10,531,271	3,745,576	3,172,465	2,976,433	93,359	434,887	1,336,895
1909	27,932,618	11,429,394	4,965,001	3,741,713	4,216,226	81,144	1,670,457	1,828,683
1910	28,497,238	10,869,561	4,517,709	3,644,495	5,417,558	55,220	2,375,490	1,617,205
1911	29,988,235	10,978,827	4,332,479	4,274,209	6,121,099	69,305	2,778,325	1,433,991
1912	35,656,414	11,411,941	5,094,432	6,542,154	6,670,845	1,195,683	3,276,169	1,465,190
1913	36,336,682	7,016,370	5,335,851	7,553,935	9,070,740	2,006,014	3,698,255	1,655,517
1914	35,175,541	9,269,413	4,346,034	7,508,020	7,578,220	2,005,024	2,882,121	1,586,709
1915	43,404,182	12,334,700	4,803,398	9,045,224	9,436,450	2,552,609	3,247,140	1,984,661
1916	57,863,365	12,364,114	6,197,978	16,491,971	12,676,152	3,452,910	4,146,956	2,533,284
1917	59,213,237	13,131,266	4,844,537	15,245,297	14,121,671	4,188,027	4,945,482	2,736,957
1918	63,004,076	12,036,372	6,412,941	18,647,039	13,512,274	4,516,407	4,994,804	2,884,239
1919	37,037,281	8,605,912	3,081,451	13,347,182	5,884,118	2,027,107	2,363,189	1,728,322
1920	38,143,870	7,317,467	4,002,491	14,516,580	5,778,373	2,090,711	2,616,237	1,822,011
1921	13,396,382	3,610,648	928,461	5,181,177	1,357,857	533,231	401,153	1,383,855
1922	26,893,247	4,952,264	3,250,213	10,431,729	4,509,380	1,477,166	629,408	1,643,087
1923	45,544,558	5,363,160	4,392,291	16,331,974	11,519,722	2,946,800	2,833,908	2,156,703
1924	49,272,382	5,372,572	3,278,662	19,460,836	12,357,366	2,832,901	3,505,301	2,464,744
1925	53,195,376	7,036,751	3,130,087	20,628,410	12,575,261	3,172,291	3,923,158	2,729,418
1926	57,280,775	7,641,682	2,825,081	22,030,481	13,949,866	3,644,936	4,888,681	2,300,048
1927	56,794,178	8,046,334	2,929,557	21,674,826	13,879,173	3,124,447	4,885,052	2,254,789
1928	62,097,903	7,361,658	3,147,671	22,583,844	16,631,757	3,733,442	6,244,047	2,395,484
1929	68,421,853	7,598,180	3,802,976	25,669,975	17,895,332	4,114,187	6,572,404	2,768,799
1930	47,381,509	6,659,036	2,238,997	19,703,349	9,696,962	2,636,107	4,015,426	2,431,632
1931	34,248,203	3,570,748	1,878,757	13,606,755	8,212,141	2,642,654	2,936,899	1,400,249
1932	12,320,194	1,142,775	668,679	4,343,070	3,196,677	1,184,528	1,357,464	427,001
1933	8,387,612	697,158	491,893	888,508	3,524,073	1,100,707	1,197,498	487,775
1934	11,723,638	700,055	458,587	2,845,604	4,092,303	1,000,972	1,819,913	806,204
1935	19,112,054	1,376,803	1,259,892	6,011,755	6,530,569	3,275	2,904,641	1,025,119
1936	38,514,245	3,225,600	2,429,529	12,829,873	13,774,589	31,056	4,668,590	1,555,008

[a]Data for 1880 are from *Tenth Census of the United States: 1880*, Vol. XV, "Report on the Mining Industries" (U. S. Dept. Int., Census Office, 1886), pp. 798-800; for 1889 from *Eleventh Census of the United States: 1890*, "Report on Mineral Industries" (U. S. Dept. Int., Census Office, 1892), p. 155 (figures represent production for Michigan, Montana, Arizona, and New Mexico); for 1902 from *Special Reports of the Census Office*, "Mines and Quarries: 1902" (U. S. Dept. Com. and Labor, Bur. Census, 1905), pp. 469, 480, 483, 485 (figures represent ore mined except for Utah, the figure for which represents short tons of ore sold or smelted); and for 1906-36 from issues of *Mineral Resources of the United States* (U. S. Dept. Int., Geol. Survey and U. S. Dept. Com., Bur. Mines) and *Minerals Yearbook* (U. S. Depts. Com. and Int., Bur. Mines) (figures represent short tons of copper ore, tailings, etc., sold or treated).

[b]Same as table A-2.

[c]Obtained by subtracting from the total the sum of the figures for the specified States.

n.a. Data not available.

Table A-7.- PRODUCTION OF ORE, COPPER, AND COPPER EQUIVALENT, YIELD OF ORE, EMPLOYMENT, AND PRODUCTIVITY IN THE COPPER-MINING INDUSTRY, BY MINING METHOD, 1917-36[a]

Year and mining method	Production										Yield[b] (pounds of copper per ton) of -	
	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Metals recovered									
			Copper from ore (pounds)	Copper from ore and tailings[c] (pounds)	Gold[d] (fine ounces)	Silver[d] (fine ounces)	Lead[d] (pounds)	Zinc[d] (pounds)	Copper equivalent of accessory metals[e] (pounds)	Copper plus copper equivalent of accessory metals[f] (pounds)	Ore	Ore and tailings
1917, all mines	51,486,691	52,234,496	1,573,524,050	1,596,685,905	227,166.24	15,304,540	11,351,855	2,265,853	106,545,564	1,703,231,469	30.56	30.57
Open-cut mines[g]	21,106,608	21,106,608	385,938,976	385,938,976	92,197.80	741,061	1,459,079	0	16,847,976	402,786,952	18.29	18.29
Underground mines	30,380,083	31,127,888	1,187,585,074	1,210,746,929	134,968.44	14,563,479	9,892,776	2,265,853	89,697,588	1,300,444,517	39.09	38.90
Block-caving	10,607,354	10,607,354	244,754,531	244,754,531	5,934.52	262,249	65,309	0	2,052,409	246,806,940	23.07	23.07
Open-stope	8,949,057	9,679,600	178,714,069	187,789,526	883.09	122,921	0	0	684,425	188,473,951	19.97	19.40
Square-set	5,754,026	5,771,288	445,537,712	459,624,110	53,287.93	9,885,494	8,988,073	1,688,966	56,345,464	515,969,574	77.43	79.64
Cut-and-fill	3,013,471	3,013,471	197,125,566	197,125,566	60,957.09	2,934,928	839,394	576,887	22,470,954	219,596,520	65.41	65.41
Shrinkage	2,056,175	2,056,175	121,453,196	121,453,196	13,905.81	1,357,887	0	0	8,144,336	129,597,532	59.07	59.07
1918, all mines	54,200,536	55,470,912	1,593,109,933	1,613,278,593	221,802.57	16,111,621	5,199,271	1,678,272	107,099,497	1,720,378,090	29.39	29.08
Open-cut mines[g]	22,133,143	22,133,143	392,896,417	392,896,417	85,720.03	666,993	876,821	0	15,397,525	408,293,942	17.75	17.75
Underground mines	32,067,393	33,337,769	1,200,213,516	1,220,382,176	136,082.54	15,444,628	4,322,450	1,678,272	91,701,972	1,312,084,148	37.43	36.61
Block-caving	12,180,309	12,180,309	279,643,266	279,643,266	7,338.85	290,275	178,451	652,885	2,678,908	282,322,174	22.96	22.96
Open-stope	8,190,149	8,905,156	159,921,033	169,166,421	884.20	119,347	0	0	668,292	169,834,713	19.53	19.00
Square-set	6,628,334	7,183,703	475,292,687	486,215,959	55,375.66	11,246,505	4,068,187	892,887	60,815,858	547,031,817	71.71	67.68
Cut-and-fill	2,716,340	2,716,340	178,912,167	178,912,167	62,531.37	2,782,530	52,964	132,500	21,546,511	200,458,678	65.87	65.87
Shrinkage	2,352,261	2,352,261	106,444,363	106,444,363	9,952.46	1,005,971	22,848	0	5,992,403	112,436,766	45.25	45.25
1919, all mines	32,093,931	33,250,376	1,050,176,442	1,068,213,557	147,913.59	10,842,104	4,554,709	2,952,279	72,979,988	1,141,193,545	32.72	32.13
Open-cut mines[g]	11,081,070	11,081,070	230,623,058	230,623,058	46,325.83	345,763	0	0	8,089,906	238,712,964	20.81	20.81
Underground mines	21,012,861	22,169,306	819,553,384	837,590,499	101,587.76	10,496,341	4,554,709	2,952,279	64,890,082	902,480,581	39.00	37.78
Block-caving	8,012,260	8,012,260	196,358,921	196,358,921	2,795.35	195,378	53,643	2,952,279	2,487,097	198,846,018	24.51	24.51
Open-stope	5,818,735	6,734,394	139,304,501	148,387,453	933.72	128,050	0	0	714,922	149,102,375	23.94	22.03
Square-set	3,498,029	3,738,815	275,989,449	284,943,612	38,334.26	7,302,293	4,486,600	0	40,229,402	325,173,014	78.90	76.21
Cut-and-fill	2,022,606	2,022,606	133,414,152	133,414,152	48,481.60	2,086,416	0	0	16,326,622	149,740,774	65.96	65.96
Shrinkage	1,661,231	1,661,231	74,486,361	74,486,361	11,042.83	784,204	14,466	0	5,132,039	79,618,400	44.84	44.84

Year and mining method	Employment: Average number of men employed	Employment: Average number of days worked	Employment: Number of man-shifts	Employment: Number of man-hours	Output per man-shift[h]: Ore sold or treated (short tons)	Output per man-shift[h]: Ore and tailings, sold or treated (short tons)	Output per man-shift[h]: Copper from ore (pounds)	Output per man-shift[h]: Copper from ore and tailings (pounds)	Output per man-shift[h]: Copper plus copper equivalent of accessory metals (pounds)	Output per man-hour[i]: Ore sold or treated (short tons)	Output per man-hour[i]: Ore and tailings, sold or treated (short tons)	Output per man-hour[i]: Copper from ore (pounds)	Output per man-hour[i]: Copper from ore and tailings (pounds)	Output per man-hour[i]: Copper plus copper equivalent of accessory metals (pounds)
1917, all mines	43,857	302	13,241,371	105,930,968	3.888	3.945	118.834	120.583	128.630	0.486	0.493	14.854	15.073	16.079
Open-cut mines[g]	4,066	364	1,478,585	11,828,680	14.275	14,275	261.019	261.019	272.414	1.784	1.784	32.627	32.627	34.052
Underground mines	39,791	296	11,762,786	94,102,288	2.583	2.646	100.961	102.930	110.556	0.323	0.331	12.620	12.866	13.819
Block-caving	5,271	320	1,687,802	13,502,416	6.285	6.285	145.014	145.014	146.230	0.786	0.786	18.127	18.127	18.279
Open-stope	9,285	309	2,871,438	22,971,504	3.117	3.371	62.239	65.399	65.637	0.390	0.421	7.780	8.175	8.205
Square-set	17,772	267	4,739,851	37,918,808	1.214	1.218	93.998	96.970	108.858	0.152	0.152	11.750	12.121	13.607
Cut-and-fill	5,250	327	1,714,950	13,719,600	1.757	1.757	114.945	114.945	128.048	0.220	0.220	14.368	14.368	16.006
Shrinkage	2,213	338	748,745	5,989,960	2.746	2.746	162.209	162.209	173.086	0.343	0.343	20.276	20.276	21.636
1918, all mines	44,478	313	13,913,563	111,308,504	3.896	3.987	114.501	115.950	123.648	0.487	0.498	14.313	14.494	15.456
Open-cut mines[g]	4,125	363	1,496,252	11,970,016	14.792	14.792	262.587	262.587	272.878	1.849	1.849	32.823	32.823	34.110
Underground mines	40,353	308	12,417,311	99,338,488	2.582	2.685	96.656	98.281	105.666	0.323	0.336	12.082	12.285	13.208
Block-caving	5,167	343	1,770,630	14,165,040	6.879	6.879	157.934	157.934	159.447	0.860	0.860	19.742	19.742	19.931
Open-stope	9,907	309	3,060,036	24,480,288	2.676	2.910	52.261	55.282	55.501	0.335	0.364	6.533	6.910	6.938
Square-set	17,877	287	5,123,832	40,990,656	1.294	1.402	92.761	94.893	106.762	0.162	0.175	11.595	11.862	13.345
Cut-and-fill	5,143	334	1,716,357	13,730,856	1.583	1.583	104.239	104.239	116.793	0.198	0.198	13.030	13.030	14.599
Shrinkage	2,259	330	746,456	5,971,648	3.151	3.151	142.600	142.600	150.627	0.394	0.394	17.825	17.825	18.828
1919, all mines	31,727	286	9,066,489	72,531,912	3.540	3.667	115.831	117.820	125.869	0.442	0.458	14.479	14.727	15.734
Open-cut mines[g]	2,702	354	956,917	7,655,336	11,580	11.580	241.006	241.006	249.460	1.447	1.447	30.126	30.126	31.183
Underground mines	29,025	279	8,109,572	64,876,576	2.591	2.734	101.060	103.284	111.286	0.324	0.342	12.633	12.911	13.911
Block-caving	3,202	348	1,114,238	8,913,904	7.191	7.191	176.227	176.227	178.459	0.899	0.899	22.028	22.028	22.307
Open-stope	7,380	294	2,169,924	17,359,392	2.682	3.104	64.198	68.384	68.713	0.335	0.388	8.025	8.548	8.589
Square-set	12,030	251	3,016,758	24,134,064	1.160	1.239	91.485	94.454	107.789	0.145	0.155	11.436	11.807	13.474
Cut-and-fill	4,623	272	1,257,284	10,058,272	1.609	1.609	106.113	106.113	119.099	0.201	0.201	13.264	13.264	14.887
Shrinkage	1,790	308	551,368	4,410,944	3.013	3.013	135.094	135.094	144.402	0.377	0.377	16.887	16.887	18.050

See footnotes at end of table.

Table A-7.- PRODUCTION OF ORE, COPPER, AND COPPER EQUIVALENT, YIELD OF ORE, EMPLOYMENT, AND PRODUCTIVITY IN THE COPPER-MINING INDUSTRY, BY MINING METHOD, 1917-36[a] - *Continued*

Year and mining method	Production										Yield[b] (pounds of copper per ton) of -	
			Metals recovered									
	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings[c] (pounds)	Gold[d] (fine ounces)	Silver[d] (fine ounces)	Lead[d] (pounds)	Zinc[d] (pounds)	Copper equivalent of accessory metals[e] (pounds)	Copper plus copper equivalent of accessory metals[f] (pounds)	Ore	Ore and tailings
1920, all mines	32,874,914	35,697,289	1,060,609,150	1,099,270,592	156,911.48	11,529,528	9,993,829	2,573,845	79,113,511	1,178,384,103	32.26	30.79
Open-cut mines[g]	11,879,461	11,879,461	236,667,022	236,667,022	53,588.18	349,521	0	0	9,128,209	245,795,231	19.92	19.92
Underground mines	20,995,453	23,817,828	823,942,128	862,603,570	103,323.30	11,180,007	9,993,829	2,573,845	69,985,302	932,588,872	39.24	36.22
Block-caving	8,617,047	8,617,047	184,207,521	184,207,521	1,602.54	138,927	70,057	1,314,837	1,410,571	185,618,092	21.38	21.38
Open-stope	4,908,089	6,286,589	117,886,410	132,024,650	698.76	466,948	0	0	2,226,520	134,251,170	24.02	21.00
Square-set	3,850,696	5,294,571	289,279,115	313,802,317	42,710.99	7,539,749	9,810,940	1,259,008	44,279,642	358,081,959	75.12	59.27
Cut-and-fill	1,905,698	1,905,698	138,389,626	138,389,626	48,351.30	2,074,941	46,057	0	16,271,978	154,661,604	72.62	72.62
Shrinkage	1,713,923	1,713,923	94,179,456	94,179,456	9,959.71	959,442	66,775	0	5,796,591	99,976,047	54.95	54.95
1921, all mines	11,747,262	12,513,552	416,107,107	428,471,546	44,206.72	5,326,705	8,067,251	6,038,083	35,716,476	464,188,022	35.42	34.24
Open-cut mines[g]	3,011,450	3,011,450	64,129,336	64,129,336	11,519.46	84,559	0	0	2,005,182	66,134,518	21.30	21.30
Underground mines	8,735,812	9,502,102	351,977,771	364,342,210	32,687.26	5,242,146	8,067,251	6,038,083	33,711,294	398,053,504	40.29	38.34
Block-caving	3,198,116	3,198,116	78,858,575	78,858,575	808.00	83,640	0	0	494,831	79,353,406	24.66	24.66
Open-stope	2,510,014	2,877,014	56,720,310	62,022,046	255.17	337,654	0	0	1,574,847	63,596,893	22.60	21.56
Square-set	865,872	1,265,162	78,568,431	85,631,134	12,320.09	3,282,372	8,067,251	6,038,083	21,915,117	107,546,251	90.74	67.68
Cut-and-fill	1,242,884	1,242,884	66,121,054	66,121,054	15,541.88	777,105	0	0	5,727,286	71,848,340	53.20	53.20
Shrinkage	918,926	918,926	71,709,401	71,709,401	3,762.12	761,375	0	0	3,999,213	75,708,614	78.04	78.04
1922, all mines	24,001,151	26,406,279	876,917,816	908,286,650	115,401.55	10,791,563	12,417,135	6,990,907	72,526,766	980,813,416	36.54	34.40
Open-cut mines[g]	7,746,579	7,746,579	161,810,473	161,810,473	34,878.42	316,732	69,426	0	6,372,028	168,182,501	20.89	20.89
Underground mines	16,254,572	18,659,700	715,107,343	746,476,177	80,523.13	10,474,831	12,347,709	6,990,907	66,154,738	812,630,915	43.99	40.00
Block-caving	7,362,612	7,362,612	179,671,069	179,671,069	2,198.51	178,354	185,857	315,916	1,313,321	180,984,390	24.40	24.40
Open-stope	2,811,828	4,190,078	85,140,426	96,502,926	208.98	364,139	0	0	1,689,066	98,191,992	30.28	23.03
Square-set	3,168,058	4,194,936	245,588,549	265,594,883	36,996.00	7,635,421	11,889,728	6,674,991	46,807,075	312,401,958	77.52	63.31
Cut-and-fill	1,576,754	1,576,754	103,605,731	103,605,731	33,612.58	1,366,967	272,124	0	11,051,144	114,656,875	65.71	65.71
Shrinkage	1,335,320	1,335,320	101,101,568	101,101,568	7,507.06	929,950	0	0	5,294,132	106,395,700	75.71	75.71

Year and mining method	Employment				Output per man-shift[h]					Output per man-hour[i]				
	Average number of men employed	Average number of days worked	Number of man-shifts	Number of man-hours	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)
1920, all mines	28,255	296	8,374,808	66,998,464	3.925	4.262	126.643	131.259	140.706	0.491	0.533	15.830	16.407	17.588
Open-cut mines[g]	2,579	362	934,231	7,473,848	12.716	12.716	253.328	253.328	263.099	1.589	1.589	31.666	31.666	32.887
Underground mines	25,676	290	7,440,577	59,524,616	2.822	3.201	110.736	115.932	125.338	0.353	0.400	13.842	14.492	15.667
Block-caving	3,221	349	1,124,947	8,999,576	7.660	7.660	163.748	163.748	165.002	0.957	0.957	20.468	20.468	20.625
Open-stope	5,350	303	1,621,736	12,973,888	3.026	3.876	72.691	81.409	82.782	0.378	0.485	9.086	10.176	10.348
Square-set	11,832	252	2,982,933	23,863,464	1.291	1.775	96.978	105.199	120.044	0.161	0.222	12.122	13.150	15.005
Cut-and-fill	3,733	323	1,205,025	9,640,200	1.581	1.581	114.844	114.844	128.347	0.198	0.198	14.355	14.355	16.043
Shrinkage	1,540	329	505,936	4,047,488	3.388	3.388	186.149	186.149	197.606	0.423	0.423	23.269	23.269	24.701
1921, all mines	15,392	217	3,337,450	26,699,600	3.520	3.749	124.678	128.383	139.085	0.440	0.469	15.585	16.048	17.386
Open-cut mines[g]	1,357	135	183,726	1,469,808	16.391	16.391	349.049	349.049	359.963	2.049	2.049	43.631	43.631	44.995
Underground mines	14,035	225	3,153,724	25,229,792	2.770	3.013	111.607	115.528	126.217	0.346	0.377	13.951	14.441	15.777
Block-caving	1,395	233	325,320	2,602,560	9.831	9.831	242.403	242.403	243.924	1.229	1.229	30.300	30.300	30.491
Open-stope	3,383	212	718,874	5,750,992	3.492	4.002	78.902	86.277	88.467	0.436	0.500	9.863	10.785	11.058
Square-set	5,462	170	926,260	7,410,080	0.935	1.366	84.823	92.448	116.108	0.117	0.171	10.603	11.556	14.514
Cut-and-fill	2,890	313	905,061	7,240,488	1.373	1.373	73.057	73.057	79.385	0.172	0.172	9.132	9.132	9.923
Shrinkage	905	307	278,209	2,225,672	3.303	3.303	257.754	257.754	272.129	0.413	0.413	32.219	32.219	34.016
1922, all mines	23,447	279	6,548,531	52,388,248	3.665	4.032	133.911	138.701	149.776	0.458	0.504	16.739	17.338	18.722
Open-cut mines[g]	1,482	245	363,487	2,907,896	21.312	21.312	445.162	445.162	462.692	2.664	2.664	55.645	55.645	57.836
Underground mines	21,965	282	6,185,044	49,480,352	2.628	3.017	115.619	120.691	131.386	0.329	0.377	14.452	15.086	16.423
Block-caving	3,071	323	991,007	7,928,056	7.429	7.429	181.302	181.302	182.627	0.929	0.929	22.663	22.663	22.828
Open-stope	4,960	263	1,302,880	10,423,040	2.158	3.216	65.348	74.069	75.365	0.270	0.402	8.168	9.259	9.421
Square-set	9,443	261	2,465,938	19,727,504	1.285	1.701	99.592	107.705	126.687	0.161	0.213	12.449	13.463	15.836
Cut-and-fill	3,207	310	993,945	7,951,560	1.586	1.586	104.237	104.237	115.355	0.198	0.198	13.030	13.030	14.419
Shrinkage	1,284	336	431,274	3,450,192	3.096	3.096	234.425	234.425	246.701	0.387	0.387	29.303	29.303	30.838

See footnotes at end of table.

Table A-7.- PRODUCTION OF ORE, COPPER, AND COPPER EQUIVALENT, YIELD OF ORE, EMPLOYMENT, AND PRODUCTIVITY IN THE COPPER-MINING INDUSTRY, BY MINING METHOD, 1917-36[a] - *Continued*

Year and mining method	Production										Yield[b] (pounds of copper per ton) of -	
			Metals recovered									
	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings[c] (pounds)	Gold[d] (fine ounces)	Silver[d] (fine ounces)	Lead[d] (pounds)	Zinc[d] (pounds)	Copper equivalent of accessory metals[e] (pounds)	Copper plus copper equivalent of accessory metals[f] (pounds)	Ore	Ore and tailings
1923, all mines	41,743,897	44,752,669	1,353,386,332	1,387,539,501	273,904.82	14,887,194	11,124,930	7,123,173	113,086,199	1,500,625,700	32.42	31.00
Open-cut mines[g]	18,433,838	18,433,838	351,645,795	351,645,795	104,827.19	806,788	282,237	0	18,515,103	370,160,898	19.08	19.08
Underground mines	23,310,059	26,318,831	1,001,740,537	1,035,893,706	169,077.63	14,080,406	10,842,693	7,123,173	94,571,096	1,130,464,802	42.97	39.36
Block-caving	11,028,847	11,028,847	252,342,847	252,342,847	5,009.02	249,968	99,990	334,952	2,012,766	254,355,613	22.88	22.88
Open-stope	3,283,303	5,026,403	105,228,118	122,129,318	320.00	292,928	0	0	1,380,110	123,509,428	32.05	24.30
Square-set	4,962,615	6,228,287	358,605,874	375,857,843	58,313.83	9,511,718	10,231,154	6,788,221	57,826,475	433,684,318	72.26	60.35
Cut-and-fill	2,416,679	2,416,679	166,180,785	166,180,785	96,911.52	3,110,292	511,549	0	27,980,583	194,161,368	68.76	68.76
Shrinkage	1,618,615	1,618,615	119,382,913	119,382,913	8,523.26	915,500	0	0	5,371,162	124,754,075	73.76	73.76
1924, all mines	46,577,894	48,424,163	1,491,003,012	1,518,067,116	296,989.49	14,684,650	7,723,549	9,929,631	115,355,357	1,633,422,473	32.01	31.35
Open-cut mines[g]	21,298,354	21,298,354	415,399,308	415,399,308	123,936.98	1,021,839	158,540	0	22,139,422	437,538,730	19.50	19.50
Underground mines	25,279,540	27,125,809	1,075,603,704	1,102,667,808	173,052.51	13,662,811	7,565,009	9,929,631	93,215,935	1,195,883,743	42.55	40.65
Block-caving	11,958,098	11,958,098	274,248,825	274,248,825	4,819.64	287,771	60,365	19,025	2,017,889	276,266,714	22.93	22.93
Open-stope	3,345,969	5,032,969	100,690,257	119,317,257	328.95	217,869	0	0	1,039,263	120,356,520	30.09	23.71
Square-set	5,806,393	5,965,662	399,892,162	408,329,266	71,829.57	8,912,406	7,284,599	9,910,606	57,226,323	465,555,589	68.87	68.45
Cut-and-fill	2,472,688	2,472,688	187,216,191	187,216,191	84,020.05	3,285,242	220,045	0	26,864,142	214,080,333	75.71	75.71
Shrinkage	1,696,392	1,696,392	113,556,269	113,556,269	12,054.30	959,523	0	0	6,068,318	119,624,587	66.94	66.94
1925, all mines	49,602,217	52,394,576	1,557,979,177	1,592,498,146	316,905.24	15,678,043	13,917,542	10,121,430	124,909,382	1,717,407,528	31.41	30.39
Open-cut mines[g]	22,738,583	22,738,583	424,526,891	424,526,891	132,293.71	1,126,489	153,905	0	23,789,849	448,316,740	18.67	18.67
Underground mines	26,863,634	29,655,993	1,133,452,286	1,167,971,255	184,611.53	14,551,554	13,763,637	10,121,430	101,119,533	1,269,090,788	42.19	39.38
Block-caving	12,617,276	12,617,276	279,132,878	279,132,878	6,103.66	347,393	101,351	207,226	2,559,940	281,692,818	22.12	22.12
Open-stope	4,077,284	6,808,889	114,263,838	141,220,838	399.19	244,001	0	0	1,168,245	142,389,083	28.02	20.74
Square-set	5,578,574	5,639,328	423,937,725	431,499,694	81,172.17	9,326,277	11,295,705	3,201,404	59,125,157	490,624,851	75.99	76.52
Cut-and-fill	2,614,851	2,614,851	199,270,610	199,270,610	84,624.01	3,560,853	2,366,581	6,712,800	31,644,222	230,914,832	76.21	76.21
Shrinkage	1,975,649	1,975,649	116,847,235	116,847,235	12,312.50	1,073,030	0	0	6,621,969	123,469,204	59.14	59.14

Year and mining method	Employment				Output per man-shift[h]					Output per man-hour[i]				
	Average number of men employed	Average number of days worked	Number of man-shifts	Number of man-hours	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)
1923, all mines	29,337	315	9,226,683	73,813,464	4.524	4.850	146.682	150.383	162.640	0.566	0.606	18.335	18.798	20.330
Open-cut mines[g]	3,283	364	1,193,947	9,551,576	15.439	15.439	294.524	294.524	310.031	1.930	1.930	36.815	36.815	38.754
Underground mines	26,054	308	8,032.736	64,261,888	2.902	3.276	124.707	128.959	140.732	0.363	0.410	15.588	16.120	17.592
Block-caving	4,407	356	1,568,464	12,547,712	7.032	7.032	160.885	160.885	162.169	0.879	0.879	20.111	20.111	20.271
Open-stope	4,126	313	1,290,246	10,321,968	2.545	3.896	81.557	94.656	95.725	0.318	0.487	10.195	11.832	11.966
Square-set	11,947	281	3,362,936	26,903,488	1.476	1.852	106.635	111.765	128.960	0.184	0.232	13.329	13.971	16.120
Cut-and-fill	3,974	316	1,257,079	10,056,632	1.922	1.922	132.196	132.196	154.454	0.240	0.240	16.524	16.524	19.307
Shrinkage	1,600	346	554,011	4,432,088	2.922	2.922	215.488	215.488	225.183	0.365	0.365	26.936	26.936	28.148
1924, all mines	29,527	316	9,343,359	74,746,872	4.985	5.183	159.579	162.476	174.822	0.623	0.648	19.947	20.309	21.853
Open-cut mines[g]	3,986	351	1,399,284	11,194,272	15.221	15.221	296.866	296.866	312.688	1.903	1.903	37.108	37.108	39.086
Underground mines	25,541	311	7,944,075	63,552,600	3.182	3.415	135.397	138.804	150.538	0.398	0.427	16.925	17.350	18.817
Block-caving	4,648	344	1,599,137	12,793,096	7.478	7.478	171.498	171.498	172.760	0.935	0.935	21.437	21.437	21.595
Open-stope	4,458	279	1,243,704	9,949,632	2.690	4.047	80.960	95.937	96.773	0.336	0.506	10.120	11.992	12.097
Square-set	10,584	300	3,174,256	25,394,048	1.829	1.879	125.980	128.638	146.666	0.229	0.235	15.747	16.080	18.333
Cut-and-fill	4,330	324	1,401,668	11,213,344	1.764	1.764	133.567	133.567	152.733	0.221	0.221	16.696	16.696	19.092
Shrinkage	1,521	345	525,310	4,202,480	3.229	3.229	216.170	216.170	227.722	0.404	0.404	27.021	27.021	28.465
1925, all mines	31,787	305	9,709,811	77,678,488	5.108	5.396	160.454	164.009	176.873	0.639	0.675	20.057	20.501	22.109
Open-cut mines[g]	4,384	365	1,600,005	12,800,040	14.212	14.212	265.328	265.328	280.197	1.776	1.776	33.166	33.166	35.025
Underground mines	27,403	296	8,109,806	64,878,448	3.312	3.657	139.763	144.020	156.488	0.414	0.457	17.470	18.002	19.561
Block-caving	4,219	350	1,476,977	11,815,816	8.543	8.543	188.989	188.989	190.723	1.068	1.068	23.624	23.624	23.840
Open-stope	4,798	284	1,363,738	10,909,904	2.990	4.993	83.787	103.554	104.411	0.374	0.624	10.473	12.944	13.051
Square-set	12,195	277	3,380,976	27,047,808	1.650	1.668	125.389	127.626	145.113	0.206	0.208	15.674	15.953	18.139
Cut-and-fill	4,570	291	1,328,247	10,625,976	1.969	1.969	150.025	150.025	173.849	0.246	0.246	18.753	18.753	21.731
Shrinkage	1,621	345	559,868	4,478,944	3.529	3.529	208.705	208.705	220.533	0.441	0.441	26.088	26.088	27.567

See footnotes at end of table.

Table A-7.- PRODUCTION OF ORE, COPPER, AND COPPER EQUIVALENT, YIELD OF ORE, EMPLOYMENT, AND PRODUCTIVITY IN THE COPPER-MINING INDUSTRY, BY MINING METHOD, 1917-36[a] - *Continued*

Year and mining method	Production										Yield[b] (pounds of copper per ton) of -	
	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Metals recovered									
			Copper from ore (pounds)	Copper from ore and tailings[c] (pounds)	Gold[d] (fine ounces)	Silver[d] (fine ounces)	Lead[d] (pounds)	Zinc[d] (pounds)	Copper equivalent of accessory metals[e] (pounds)	Copper plus copper equivalent of accessory metals[f] (pounds)	Ore	Ore and tailings
1926, all mines	53,908,697	57,018,844	1,608,504,802	1,650,701,343	353,130.14	15,480,610	12,408,675	8,301,426	127,849,219	1,778,550,562	29.84	28.95
Open-cut mines[g]	25,264,255	25,264,255	479,163,193	479,163,193	142,906.41	1,221,045	673,321	489,178	26,089,636	505,252,829	18.97	18.97
Underground mines	28,644,442	31,754,589	1,129,341,609	1,171,538,150	210,223.73	14,259,565	11,735,354	7,812,248	101,759,583	1,273,297,733	39.43	36.89
Block-caving	14,028,422	14,028,422	282,572,679	282,572,679	9,095.74	344,946	125,437	51,900	2,915,525	285,488,204	20.14	20.14
Open-stope	4,457,942	7,214,119	132,154,761	163,142,837	643.49	220,633	0	0	1,096,088	164,238,925	29.64	22.61
Square-set	5,379,355	5,733,325	403,932,082	415,140,547	91,021.99	8,730,453	9,514,435	1,871,458	56,642,723	471,783,270	75.09	72.41
Cut-and-fill	2,681,824	2,681,824	201,167,218	201,167,218	88,725.04	3,908,423	2,095,482	5,888,890	33,380,300	234,547,518	75.01	75.01
Shrinkage	2,096,899	2,096,899	109,514,869	109,514,869	20,737.47	1,055,110	0	0	7,724,947	117,239,816	52.23	52.23
1927, all mines	53,427,937	56,475,825	1,551,370,797	1,585,688,167	350,273.63	14,026,725	14,938,257	2,683,855	119,443,947	1,705,132,114	29.04	28.08
Open-cut mines[g]	24,062,939	24,062,939	460,795,448	460,795,448	150,117.02	1,315,231	4,333,085	1,111,341	29,052,245	489,847,693	19.15	19.15
Underground mines	29,364,998	32,412,886	1,090,575,349	1,124,892,719	200,156.61	12,711,494	10,605,172	1,572,514	90,391,702	1,215,284,421	37.14	34.71
Block-caving	14,306,063	14,306,063	287,387,767	287,387,767	17,702.62	310,669	16,904	9,604	3,914,899	291,302,666	20.09	20.09
Open-stope	4,676,630	7,695,630	136,956,706	165,673,706	581.07	189,762	0	0	946,607	166,620,313	29.29	21.53
Square-set	5,462,651	5,491,539	371,107,563	376,707,933	71,415.59	7,588,053	10,238,342	695,886	48,458,289	425,166,222	67.94	68.60
Cut-and-fill	2,584,907	2,584,907	193,100,935	193,100,935	86,288.40	3,790,123	349,926	867,024	29,877,308	222,978,243	74.70	74.70
Shrinkage	2,334,747	2,334,747	102,022,378	102,022,378	24,168.93	832,887	0	0	7,194,599	109,216,977	43.70	43.70
1928, all mines	59,081,963	61,737,797	1,711,742,268	1,742,748,465	393,266.76	13,855,639	14,249,593	2,549,647	124,416,738	1,867,165,203	28.97	28.23
Open-cut mines[g]	28,615,549	28,615,549	551,833,019	551,833,019	185,346.29	1,653,463	8,352,911	1,972,195	37,287,660	589,120,679	19.28	19.28
Underground mines	30,466,414	33,122,248	1,159,909,249	1,190,915,446	207,920.47	12,202,176	5,896,682	577,452	87,129,078	1,278,044,524	38.07	35.96
Block-caving	14,930,700	14,930,700	285,892,660	285,892,660	21,761.66	308,456	20,483	0	4,472,950	290,365,610	19.15	19.15
Open-stope	4,604,248	7,233,248	140,056,662	169,952,662	784.29	123,719	0	0	674,170	170,626,832	30.42	23.50
Square-set	5,650,124	5,676,958	417,360,968	418,471,165	71,045.66	6,869,204	5,467,274	577,452	43,427,083	461,898,248	73.87	73.71
Cut-and-fill	2,908,765	2,908,765	225,444,353	225,444,353	88,952.52	4,129,926	408,925	0	31,473,158	256,917,511	77.51	77.51
Shrinkage	2,372,577	2,372,577	91,154,606	91,154,606	25,376.34	770,871	0	0	7,081,717	98,236,323	38.42	38.42

Year and mining method	Employment				Output per man-shift[h]					Output per man-hour[i]				
	Average number of men employed	Average number of days worked	Number of man-shifts	Number of man-hours	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)
1926, all mines	30,636	325	9,968,867	80,422,426	5.408	5.720	161.353	165.586	178.411	0.670	0.709	20.001	20.525	22.115
Open-cut mines[g]	4,414	365	1,611,153	12,889,224	15.681	15.681	297.404	297.404	313.597	1.960	1.960	37.175	37.175	39.200
Underground mines	26,222	319	8,357,714	67,533,202	3.427	3.799	135.126	140.174	152.350	0.424	0.470	16.723	17.348	18.854
Block-caving	4,532	344	1,559,880	12,479,040	8.993	8.993	181.150	181.150	183.019	1.124	1.124	22.644	22.644	22.877
Open-stope	5,316	313	1,665,818	13,879,304	2.676	4.331	79.333	97.936	98.594	0.321	0.520	9.522	11.754	11.833
Square-set	10,702	309	3,304,900	26,439,200	1.628	1.735	122.222	125.614	142.753	0.203	0.217	15.278	15.702	17.844
Cut-and-fill	3,970	314	1,247,072	10,058,726	2.150	2.150	161.312	161.312	188.079	0.267	0.267	19.999	19.999	23.318
Shrinkage	1,702	341	580,044	4,676,932	3.615	3.615	188.804	188.804	202.122	0.448	0.448	23.416	23.416	25.068
1927, all mines	29,242	321	9,389,029	75,814,219	5.690	6.015	165.232	168.887	181.609	0.705	0.745	20.463	20.915	22.491
Open-cut mines[g]	3,787	355	1,344,769	10,758,152	17.894	17.894	342.658	342.658	364.262	2.237	2.237	42.832	42.832	45.533
Underground mines	25,455	316	8,044,260	65,056,067	3.650	4.029	135.572	139.838	151.075	0.451	0.498	16.764	17.291	18.681
Block-caving	4,327	349	1,508,458	12,067,664	9.484	9.484	190.518	190.518	193.113	1.185	1.185	23.815	23.815	24.139
Open-stope	5,483	297	1,627,952	13,613,693	2.873	4.727	84.128	101.768	102.350	0.344	0.565	10.060	12.170	12.239
Square-set	10,539	308	3,246,712	25,973,696	1.683	1.691	114.303	116.028	130.953	0.210	0.211	14.288	14.503	16.369
Cut-and-fill	3,500	315	1,103,252	8,901,036	2.343	2.343	175.029	175.029	202.110	0.290	0.290	21.694	21.694	25.051
Shrinkage	1,606	347	557,886	4,499,978	4.185	4.185	182.873	182.873	195.769	0.519	0.519	22.672	22.672	24.271
1928, all mines	30,522	314	9,569,144	77,245,104	6.174	6.452	178.881	182.122	195.124	0.765	0.799	22.160	22.561	24.172
Open-cut mines[g]	4,167	361	1,505,502	12,044,016	19.007	19.007	366.544	366.544	391.312	2.376	2.376	45.818	45.818	48.914
Underground mines	26,355	306	8,063,642	65,201,088	3.778	4.108	143.844	147.690	158.495	0.467	0.508	17.790	18.265	19.602
Block-caving	4,081	357	1,457,539	11,660,312	10.244	10.244	196.148	196.148	199.216	1.280	1.280	24.518	24.518	24.902
Open-stope	5,039	310	1,562,495	13,088,224	2.947	4.629	89.637	108.770	109.202	0.352	0.553	10.701	12.985	13.037
Square-set	11,728	271	3,173,081	25,384,648	1.781	1.789	131.532	131.882	145.568	0.223	0.224	16.441	16.485	18.196
Cut-and-fill	3,816	345	1,314,856	10,583,293	2.212	2.212	171.459	171.459	195.396	0.275	0.275	21.302	21.302	24.276
Shrinkage	1,691	329	555,671	4,484,611	4.270	4.270	164.044	164.044	176.789	0.529	0.529	20.326	20.326	21.905

See footnotes at end of table.

Table A-7.- PRODUCTION OF ORE, COPPER, AND COPPER EQUIVALENT, YIELD OF ORE, EMPLOYMENT, AND PRODUCTIVITY IN THE COPPER-MINING INDUSTRY, BY MINING METHOD, 1917-36[a] - *Continued*

Year and mining method	Production										Yield[b] (pounds of copper per ton) of -	
	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Metals recovered									
			Copper from ore (pounds)	Copper from ore and tailings[c] (pounds)	Gold[d] (fine ounces)	Silver[d] (fine ounces)	Lead[d] (pounds)	Zinc[d] (pounds)	Copper equivalent of accessory metals[e] (pounds)	Copper plus copper equivalent of accessory metals[f] (pounds)	Ore	Ore and tailings
1929, all mines	65,197,640	67,670,562	1,873,967,159	1,916,037,768	417,835.96	16,207,532	12,853,292	2,084,269	137,919,765	2,053,957,533	28.74	28.31
Open-cut mines[g]	30,640,936	30,640,936	561,029,232	561,029,232	190,939.09	1,749,267	10,775,999	2,084,269	39,396,379	600,425,611	18.31	18.31
Underground mines	34,556,704	37,029,626	1,312,937,927	1,355,008,536	226,896.87	14,458,265	2,077,293	0	98,523,386	1,453,531,922	37.99	36.59
Block-caving	17,217,016	17,217,016	314,160,938	314,160,938	17,835.30	288,068	35,394	0	3,833,104	317,994,042	18.25	18.25
Open-stope	5,123,135	7,567,135	152,287,817	185,798,817	1,456.04	152,995	0	0	902,061	186,700,878	29.73	24.55
Square-set	6,740,491	6,769,413	495,613,110	504,172,719	76,226.19	8,539,687	2,040,899	0	50,348,785	554,521,504	73.53	74.48
Cut-and-fill	3,122,020	3,122,020	260,079,021	260,079,021	102,888.75	4,574,121	0	0	35,315,453	295,394,474	83.30	83.30
Shrinkage	2,354,042	2,354,042	90,797,041	90,797,041	28,490.59	903,394	1,000	0	8,123,983	98,921,024	38.57	38.57
1930, all mines	45,108,280	46,683,016	1,312,517,806	1,342,846,987	291,160.66	10,975,032	3,940,680	2,472,101	93,322,387	1,436,169,374	29.10	28.77
Open-cut mines[g]	17,320,623	17,320,623	339,331,261	339,331,261	116,345.51	986,669	3,628,349	322,101	22,244,760	361,576,021	19.59	19.59
Underground mines	27,787,657	29,362,393	973,186,545	1,003,515,726	174,815.15	9,988,363	312,331	2,150,000	71,077,627	1,074,593,353	35.02	34.18
Block-caving	14,039,957	14,039,957	247,047,435	247,047,435	19,196.60	218,336	0	0	3,694,414	250,741,849	17.60	17.60
Open-stope	5,174,375	6,719,375	158,504,884	176,346,884	2,381.20	162,150	0	2,150,000	1,936,802	178,283,686	30.63	26.24
Square-set	4,511,007	4,540,743	328,053,958	340,541,139	48,843.24	5,366,487	206,665	0	31,399,156	371,940,295	72.72	75.00
Cut-and-fill	2,127,490	2,127,490	167,187,904	167,187,904	74,570.59	3,395,711	96,080	0	25,995,922	193,183,826	78.58	78.58
Shrinkage	1,934,828	1,934,828	72,392,364	72,392,364	29,823.52	845,679	9,586	0	8,051,333	80,443,697	37.42	37.42
1931, all mines	33,863,385	33,881,998	1,017,938,820	1,027,330,781	205,044.68	7,897,080	510,384	2,097,833	65,844,346	1,093,175,127	30.06	30.32
Open-cut mines[g]	14,690,625	14,690,625	292,256,872	292,256,872	95,576.47	824,001	7,142	5,833	17,199,692	309,456,564	19.89	19.89
Underground mines	19,172,760	19,191,373	725,681,948	735,073,909	109,468.21	7,073,079	503,242	2,092,000	48,644,654	783,718,563	37.85	38.30
Block-caving	10,331,477	10,331,477	191,561,390	191,561,390	12,457.20	159,510	0	0	2,478,653	194,040,043	18.54	18.54
Open-stope	3,466,441	3,466,441	121,401,584	121,401,584	1,083.08	78,788	0	2,092,000	1,351,042	122,752,626	35.02	35.02
Square-set	3,281,357	3,299,970	275,496,111	284,888,072	38,230.82	4,503,184	466,547	0	26,062,301	310,950,373	83.96	86.33
Cut-and-fill	1,137,652	1,137,652	95,048,482	95,048,482	34,016.85	1,734,058	36,695	0	12,699,446	107,747,928	83.55	83.55
Shrinkage	955,833	955,833	42,174,381	42,174,381	23,680.26	597,539	0	0	6,053,212	48,227,593	44.12	44.12

Year and mining method	Employment				Output per man-shift[h]					Output per man-hour[i]				
	Average number of men employed	Average number of days worked	Number of man-shifts	Number of man-hours	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)
1929, all mines	35,499	318	11,286,433	91,035,980	5.777	5.996	166.037	169.765	181.985	0.716	0.743	20.585	21.047	22.562
Open-cut mines[g]	4,790	357	1,712,206	13,697,648	17.896	17.896	327.665	327.665	350.674	2.237	2.237	40.958	40.958	43.834
Underground mines	30,709	312	9,574,227	77,338,332	3.609	3.868	137.133	141.527	151.817	0.447	0.479	16.977	17.521	18.794
Block-caving	4,955	340	1,685,005	13,480,040	10.218	10.218	186.445	186.445	188.720	1.277	1.277	23.306	23.306	23.590
Open-stope	5,847	315	1,839,112	15,326,579	2.786	4.115	82.805	101.026	101.517	0.334	0.494	9.936	12.123	12.182
Square-set	13,540	288	3,893,712	31,149,696	1.731	1.739	127.286	129.484	142.415	0.216	0.217	15.911	16.185	17.802
Cut-and-fill	4,310	345	1,485,433	11,968,727	2.102	2.102	175.086	175.086	198.861	0.261	0.261	21.730	21.730	24.681
Shrinkage	2,057	326	670,965	5,413,290	3.508	3.508	135.323	135.323	147.431	0.435	0.435	16.773	16.773	18.274
1930, all mines	25,663	300	7,706,440	62,323,121	5.853	6.058	170.314	174.250	186.360	0.724	0.749	21.060	21.547	23.044
Open-cut mines[g]	3,046	350	1,065,248	8,521,984	16.260	16.260	318.547	318.547	339.429	2.032	2.032	39.818	39.818	42.429
Underground mines	22,617	294	6,641,192	53,801,137	4.184	4.421	146.538	151.105	161.807	0.516	0.546	18.089	18.652	19.973
Block-caving	3,406	317	1,078,615	8,628,920	13.017	13.017	229.041	229.041	232.466	1.627	1.627	28.630	28.630	29.058
Open-stope	5,968	296	1,763,901	14,674,301	2.933	3.809	89.860	99.975	101.074	0.353	0.458	10.802	12.017	12.149
Square-set	7,924	280	2,219,061	17,752,488	2.033	2.046	147.835	153.462	167.612	0.254	0.256	18.479	19.183	20.951
Cut-and-fill	3,529	299	1,055,033	8,509,892	2.017	2.017	158.467	158.467	183.107	0.250	0.250	19.646	19.646	22.701
Shrinkage	1,790	293	524,582	4,235,536	3.688	3.688	138.000	138.000	153.348	0.457	0.457	17.092	17.092	18.993
1931, all mines	18,764	262	4,922,135	39,779,776	6.880	6.884	206.808	208.716	222.094	0.851	0.852	25.589	25.825	27.481
Open-cut mines[g]	2,335	312	729,407	5,835,256	20.141	20.141	400.677	400.677	424.258	2.518	2.518	50.085	50.085	53.032
Underground mines	16,429	255	4,192,728	33,944,520	4.573	4.577	173.081	175.321	186.923	0.565	0.565	21.378	21.655	23.088
Block-caving	2,057	293	602,947	4,823,576	17.135	17.135	317.709	317.709	321.819	2.142	2.142	39.714	39.714	40.227
Open-stope	5,194	209	1,083,473	9,018,806	3.199	3.199	112.049	112.049	113.296	0.384	0.384	13.461	13.461	13.611
Square-set	5,727	287	1,646,217	13,169,736	1.993	2.005	167.351	173.056	188.888	0.249	0.251	20.919	21.632	23.611
Cut-and-fill	2,521	246	620,089	4,986,914	1.835	1.835	153.282	153.282	173.762	0.228	0.228	19.060	19.060	21.606
Shrinkage	930	258	240,002	1,945,488	3.983	3.983	175.725	175.725	200.947	0.491	0.491	21.678	21.678	24.789

See footnotes at end of table.

Table A-7.- PRODUCTION OF ORE, COPPER, AND COPPER EQUIVALENT, YIELD OF ORE, EMPLOYMENT, AND PRODUCTIVITY IN THE COPPER-MINING INDUSTRY, BY MINING METHOD, 1917-36[a] - *Continued*

Year and mining method	Production										Yield[b] (pounds of copper per ton) of -	
	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Metals recovered									
			Copper from ore (pounds)	Copper from ore and tailings[c] (pounds)	Gold[d] (fine ounces)	Silver[d] (fine ounces)	Lead[d] (pounds)	Zinc[d] (pounds)	Copper equivalent of accessory metals[e] (pounds)	Copper plus copper equivalent of accessory metals[f] (pounds)	Ore	Ore and tailings
1932, all mines	12,162,502	12,176,845	445,803,526	455,470,167	91,063.63	4,034,749	861,767	786,000	31,808,784	487,278,951	36.65	37.40
Open-cut mines[g]	5,963,153	5,963,153	122,161,249	122,161,249	43,393.68	352,523	0	0	7,708,421	129,869,670	20.49	20.49
Underground mines	6,199,349	6,213,692	323,642,277	333,308,918	47,669.95	3,682,226	861,767	786,000	24,100,363	357,409,281	52.21	53.64
Block-caving	3,422,328	3,422,328	68,114,968	68,114,968	2,465.59	46,985	0	0	560,842	68,675,810	19.90	19.90
Open-stope	1,039,829	1,039,829	51,608,627	51,608,627	317.38	102,083	0	786,000	825,373	52,434,000	49.63	49.63
Square-set	879,831	894,174	110,877,341	120,543,982	13,476.05	1,921,887	23,964	0	10,662,832	131,206,814	126.02	134.81
Cut-and-fill	699,805	699,805	81,639,411	81,639,411	27,327.69	1,493,290	837,803	0	10,939,426	92,578,837	116.66	116.66
Shrinkage	157,556	157,556	11,401,930	11,401,930	4,083.24	117,981	0	0	1,111,890	12,513,820	72.37	72.37
1933, all mines	8,243,327	8,243,327	348,589,323	357,745,833	100,682.53	4,204,040	215,277	693,000	33,671,298	391,417,131	42.29	43.40
Open-cut mines[g]	5,821,488	5,821,488	119,608,509	119,608,509	48,008.03	399,664	0	0	8,572,115	128,180,624	20.55	20.55
Underground mines	2,421,839	2,421,839	228,980,814	238,137,324	52,674.50	3,804,376	215,277	693,000	25,099,183	263,236,507	94.55	98.33
Block-caving	132,134	132,134	3,067,617	3,067,617	42.63	1,853	0	0	14,440	3,082,057	23.22	23.22
Open-stope	849,464	849,464	47,190,320	47,190,320	335.64	51,269	0	693,000	559,013	47,749,333	55.55	55.55
Square-set	719,953	719,953	88,943,995	98,100,505	14,135.74	1,945,215	1,020	0	10,853,958	108,954,463	123.54	136.26
Cut-and-fill	718,171	718,171	89,725,282	89,725,282	35,864.87	1,801,543	213,737	0	13,328,308	103,053,590	124.94	124.94
Shrinkage	2,117	2,117	53,600	53,600	2,295.62	4,496	520	0	343,464	397,064	25.32	25.32
1934, all mines	11,502,188	11,502,188	439,079,547	444,246,852	141,840.53	5,786,300	125,558	2,246,000	47,262,464	491,509,316	38.17	38.62
Open-cut mines[g]	8,388,549	8,388,549	172,097,969	172,097,969	71,917.54	584,455	0	0	12,776,329	184,874,298	20.52	20.52
Underground mines	3,113,639	3,113,639	266,981,578	272,148,883	69,922.99	5,201,845	125,558	2,246,000	34,486,135	306,635,018	85.75	87.41
Block-caving	356,110	356,110	14,719,045	14,719,045	16.40	4,512	0	0	22,871	14,741,916	41.33	41.33
Open-stope	1,069,391	1,069,391	51,842,800	51,842,800	655.00	70,648	0	2,246,000	1,315,558	53,158,358	48.48	48.48
Square-set	667,898	667,898	83,638,463	88,805,768	10,648.52	2,068,046	95,621	0	10,956,275	99,762,043	125.23	132.96
Cut-and-fill	1,017,222	1,017,222	116,730,556	116,730,556	56,368.62	3,052,490	29,937	0	21,849,214	138,579,770	114.75	114.75
Shrinkage	3,018	3,018	50,714	50,714	2,234.45	6,149	0	0	342,217	392,931	16.80	16.80

Year and mining method	Employment				Output per man-shift[h]					Output per man-hour[i]				
	Average number of men employed	Average number of days worked	Number of man-shifts	Number of man-hours	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)
1932, all mines	9,086	228	2,072,046	16,894,743	5.870	5.877	215.151	219.817	235.168	0.720	0.721	26.387	26.959	28.842
Open-cut mines[g]	1,653	207	341,465	2,731,720	17.463	17.463	357.756	357.756	380.331	2.183	2.183	44.720	44.720	47,541
Underground mines	7,433	233	1,730,581	14,163,023	3.582	3.591	187.014	192.599	206.526	0.438	0.439	22.851	23.534	25.235
Block-caving	1,104	161	177,257	1,418,056	19.307	19.307	384.272	384.272	387.436	2.413	2.413	48.034	48.034	48.430
Open-stope	2,152	268	577,592	4,913,981	1.800	1.800	89.351	89.351	90.780	0.212	0.212	10.502	10.502	10.670
Square-set	2,350	235	552,159	4,417,272	1.593	1.619	200.807	218.314	237.625	0.199	0.202	25.101	27.289	29.703
Cut-and-fill	1,501	252	378,949	3,048,631	1.847	1.847	215.436	215.436	244.304	0.230	0.230	26.779	26.779	30.367
Shrinkage	326	137	44,624	365,083	3.531	3.531	255.511	255.511	280.428	0.432	0.432	31.231	31.231	34.277
1933, all mines	6,652	212	1,410,414	11,283,312	5.845	5.845	247.154	253.646	277.519	0.731	0.731	30.894	31.706	34.690
Open-cut mines[g]	1,271	221	281,485	2,251,880	20.681	20.681	424.920	424.920	455.373	2.585	2.585	53.115	53.115	56.922
Underground mines	5,381	210	1,128,929	9,031,432	2.145	2.145	202.830	210.941	233.174	0.268	0.268	25.354	26.368	29.147
Block-caving	510	31	15,957	127,656	8.281	8.281	192.243	192.243	193.148	1.035	1.035	24.030	24.030	24.143
Open-stope	1,004	286	287,493	2,299,944	2.955	2.955	164.144	164.144	166.089	0.369	0.369	20.518	20.518	20.761
Square-set	2,429	186	451,779	3,614,232	1.594	1.594	196.875	217.143	241.168	0.199	0.199	24.609	27.143	30.146
Cut-and-fill	1,421	260	369,025	2,952,200	1.946	1.946	243.141	243.141	279.259	0.243	0.243	30.393	30.393	34.907
Shrinkage	17	275	4,675	37,400	0.453	0.453	11.465	11.465	84.933	0.057	0.057	1.433	1.433	10.617
1934, all mines	7,749	215	1,668,104	13,344,832	6.895	6.895	263.221	266.318	294.651	0.862	0.862	32.903	33.290	36.831
Open-cut mines[g]	1,769	215	380,711	3,045,688	22.034	22.034	452.044	452.044	485.603	2.754	2.754	56.505	56.505	60.700
Underground mines	5,980	215	1,287,393	10,299,144	2.419	2.419	207.382	211.395	238.183	0.302	0.302	25.923	26.424	29.773
Block-caving	104	167	17,317	138,536	20.564	20.564	849.977	849.977	851.297	2.571	2.571	106.247	106.247	106.412
Open-stope	1,144	273	311,943	2,495,544	3.428	3.428	166.193	166.193	170.410	0.429	0.429	20.774	20.774	21.301
Square-set	2,926	165	483,384	3,867,072	1.382	1.382	173.027	183.717	206.383	0.173	0.173	21.628	22.965	25.798
Cut-and-fill	1,794	263	471,149	3,769,192	2.159	2.159	247.757	247.757	294.132	0.270	0.270	30.970	30.970	36.766
Shrinkage	12	300	3,600	28,800	0.838	0.838	14.087	14.087	109.148	0.105	0.105	1.761	1.761	13.643

See footnotes at end of table.

Table A-7.- PRODUCTION OF ORE, COPPER, AND COPPER EQUIVALENT, YIELD OF ORE, EMPLOYMENT, AND PRODUCTIVITY IN THE COPPER-MINING INDUSTRY, BY MINING METHOD, 1917-36[a] - *Continued*

Year and mining method	Production										Yield[b] (pounds of copper per ton) of -	
	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Metals recovered									
			Copper from ore (pounds)	Copper from ore and tailings[c] (pounds)	Gold[d] (fine ounces)	Silver[d] (fine ounces)	Lead[d] (pounds)	Zinc[d] (pounds)	Copper equivalent of accessory metals[e] (pounds)	Copper plus copper equivalent of accessory metals[f] (pounds)	Ore	Ore and tailings
1935, all mines	18,245,802	18,868,802	692,535,832	711,313,757	218,814.12	9,858,104	481,906	2,600,000	76,910,168	788,223,925	37.96	37.70
Open-cut mines[g]	12,568,312	12,568,312	250,868,968	250,868,968	119,930.43	946,510	9,459	0	21,180,994	272,049,962	19.96	19.96
Underground mines	5,677,490	6,300,490	441,666,864	460,444,789	98,883.69	8,911,594	472,447	2,600,000	55,729,174	516,173,963	77.79	73.08
Block-caving	1,168,629	1,168,629	38,051,027	38,051,027	288.10	53,071	0	0	282,398	38,333,425	32.56	32.56
Open-stope	1,139,970	1,762,970	55,146,380	64,264,380	603.00	54,057	0	2,600,000	1,374,709	65,639,089	48.38	36.45
Square-set	1,451,062	1,451,062	164,200,985	173,860,910	12,077.73	4,697,207	437,896	0	23,259,238	197,120,148	113.16	119.82
Cut-and-fill	1,780,352	1,780,352	167,323,675	167,323,675	81,587.88	3,917,037	31,879	0	29,336,477	196,660,152	93.98	93.98
Shrinkage	137,477	137,477	16,944,797	16,944,797	4,326.98	190,222	2,672	0	1,476,352	18,421,149	123.26	123.26
1936, all mines	36,551,791	38,098,791	1,123,032,954	1,151,785,142	365,020.63	13,542,079	611,704	2,890,000	114,420,927	1,266,206,069	30.72	30.23
Open-cut mines[g]	23,265,345	23,265,345	452,477,985	452,477,985	209,573.89	1,669,909	0	0	37,079,805	489,557,790	19.45	19.45
Underground mines	13,286,446	14,833,446	670,554,969	699,307,157	155,446.74	11,872,170	611,704	2,890,000	77,341,122	776,648,279	50.47	47.14
Block-caving	5,532,120	5,532,120	111,797,031	111,797,031	730.60	64,002	0	0	394,441	112,191,472	20.21	20.21
Open-stope	2,039,716	3,586,716	77,694,800	96,861,800	525.00	53,248	0	2,890,000	1,476,448	98,338,248	38.09	27.01
Square-set	2,574,779	2,574,779	224,817,025	234,402,213	14,397.54	5,843,813	59,404	0	28,680,146	263,082,359	87.32	91.04
Cut-and-fill	2,539,097	2,539,097	209,822,669	209,822,669	124,715.29	5,290,495	552,300	0	41,841,252	251,663,921	82.64	82.64
Shrinkage	600,734	600,734	46,423,444	46,423,444	15,078.31	620,612	0	0	4,948,835	51,372,279	77.28	77.28

Year and mining method	Employment				Output per man-shift[h]					Output per man-hour[i]				
	Average number of men employed	Average number of days worked	Number of man-shifts	Number of man-hours	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)
1935, all mines	8,845	271	2,394,755	19,158,040	7.619	7.879	289.189	297.030	329.146	0.952	0.985	36.149	37.129	41.143
Open-cut mines[g]	1,625	280	454,935	3,639,480	27.627	27.627	551.439	551.439	597.997	3.453	3.453	68.930	68.930	74.750
Underground mines	7,220	269	1,939,820	15,518,560	2.927	3.248	227.684	237.365	266.094	0.366	0.406	28.461	29.671	33.262
Block-caving	551	144	79.500	636,000	14.700	14.700	478.629	478.629	482.181	1.837	1.837	59.829	59.829	60.273
Open-stope	1,219	264	322,085	2,576,678	3.539	5.474	171.217	199.526	203.794	0.442	0.684	21.402	24.941	25.474
Square-set	2,850	301	856,938	6,855,502	1.693	1.693	191.614	202.886	230.028	0.212	0.212	23.952	25.361	28.754
Cut-and-fill	2,275	278	633,253	5,066,028	2.811	2.811	264.229	264.229	310.555	0.351	0.351	33.029	33.029	38.819
Shrinkage	325	148	48,044	384,352	2.861	2.861	352.693	352.693	383.422	0.358	0.358	44.087	44.087	47.928
1936, all mines	12,341	311	3,836,618	30,694,017	9.527	9.930	292.714	300.208	330.032	1.191	1.241	36.588	37.525	41.253
Open-cut mines[g]	2,197	337	739,797	5,917,845	31.448	31.448	611.625	611.625	661.746	3.931	3.931	76.460	76.460	82.726
Underground mines	10,144	305	3,096,821	24,776,172	4.290	4.790	216.530	225.815	250.789	0.536	0.599	27.065	28.225	31.347
Block-caving	762	318	241,978	1,935,824	22.862	22.862	462.013	462.013	463.643	2.858	2.858	57.752	57.752	57.955
Open-stope	1,545	297	458,609	3,668,867	4.448	7.821	169.414	211.208	214.427	0.556	0.978	21.177	26.401	26.803
Square-set	4,724	299	1,412,727	11,301,819	1.823	1.823	159.137	165.922	186.223	0.228	0.228	19.892	20.740	23.278
Cut-and-fill	2,680	312	836,225	6,689,806	3.036	3.036	250.917	250.917	300.952	0.380	0.380	31.365	31.365	37.619
Shrinkage	433	340	147,282	1,179,856	4.079	4.079	315.201	315.201	348.802	0.509	0.509	39.347	39.347	43.541

[a]The mines covered in this table are those of principal domestic mine operators producing copper principally and whose output of copper exceeded 2,000,000 pounds in 1929. Copper producers not covered in this table are small copper mines and some large mines that produced in 1929 a considerable quantity of copper which, in that year and in most of the other years, was not the principal metal produced; in 1929 there were, in the latter group, seven mines that produced more than 2,000,000 pounds of copper each as an accessory metal. The total copper output from these mines was 23,280,000 pounds. It should be noted, however, that this 1929 production was exceptional, for the copper output of these mines is usually relatively smaller than in 1928-29, when copper prices reached a peak.

The mines are classified according to their mining method which predominated over the entire 1917-36 period. For example, a mine which produced more of its copper by the square-set method than by any other single method during the period 1917-36 was considered a square-set mine. This general rule was adopted because it was not possible to segregate the employment involved in mining the ore or copper by each of several methods utilized, even though it may have been possible in some cases to determine the quantity of copper produced by each method.

The basic data were compiled from reports of individual companies to the Bureau of Mines.

[b]Copper production divided by ore production.

[c]Includes mine-water precipitates.

[d]Recovered from copper ores, tailings, and mine-water precipitates.

[e]The four accessory metals were converted to copper equivalents by multiplying the quantity of each metal by a constant price for that metal, aggregating the values thus obtained, and dividing the sum by a constant price for copper to obtain the number of pounds of copper equivalent. The constant prices used in this calculation are the arithmetic averages of the yearly prices of the respective metals from 1890 to 1932, inclusive (gold, $20.67 per fine ounce; silver, $0.67 per fine ounce; lead, $0.051 per pound; zinc, $0.059 per pound; and copper, $0.147 per pound).

[f]Includes the copper and copper equivalent of the accessory metals recovered from copper ores, tailings, and mine-water precipitates.

[g]Includes data for the Utah Copper Co., 1917-36; Nevada Consolidated Copper Co., Chino mine, 1917-34 (idle in 1935-36); Nevada Consolidated Copper Co., Copper Flat pits and Ruth mine, 1917-36; and New Cornelia mine, 1917-32 and 1934-36 (idle in 1933). Since employment data were available only in the combined form for the open-cut mine at Copper Flat and the underground block-caving mine at Ruth, it was necessary to include data for the latter with data for the open-cut mines. Insofar as the Ruth mine is less productive than the Copper Flat pits, its inclusion tends to understate the level of the over-all output per worker for the open-cut mines. Data for the other two open-cut mines - Sacramento Hill and United Verde - are not included because the open-cut production at each mine was generally less than half of the respective company's total production. Furthermore, because the employment data were available only in combined form, for open-cut and underground operation, it was not possible to separate with a reasonable degree of accuracy the workers engaged in mining by each method.

[h]Appropriate measure of production divided by number of man-shifts.

[i]Appropriate measure of production divided by number of man-hours.

Table A-8.- PRODUCTION OF ORE, COPPER, AND COPPER EQUIVALENT, YIELD OF ORE, EMPLOYMENT, AND PRODUCTIVITY IN THE COPPER-MINING INDUSTRY, BY STATE, 1917-36[a]

Year and State	Production										Yield[b] (pounds of copper per ton) of -	
	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Metals recovered									
			Copper from ore (pounds)	Copper from ore and tailings[c] (pounds)	Gold[d] (fine ounces)	Silver[d] (fine ounces)	Lead[d] (pounds)	Zinc[d] (pounds)	Copper equivalent of accessory metals[e] (pounds)	Copper plus copper equivalent of accessory metals[f] (pounds)	Ore	Ore and tailings
1917, all mines	51,486,691	52,234,496	1,573,524,050	1,596,685,905	227,166.24	15,304,540	11,351,855	2,265,853	106,545,564	1,703,231,469	30.562	30.568
Arizona	13,385,140	13,385,140	605,983,331	605,983,331	91,129.67	4,686,407	9,719,667	576,887	37,777,431	643,760,762	45.273	45.273
Michigan	10,557,888	11,288,431	216,354,054	225,429,511	0	0	0	0	0	225,429,511	20.492	19.970
Alaska, North Carolina, and Tennessee	1,128,106	1,128,106	94,245,178	94,425,178	257.05	1,032,626	0	0	4,742,671	98,987,849	83.543	83.543
California, Idaho, and Montana	4,650,912	4,668,174	257,013,619	271,100,017	29,808.15	8,569,178	107,800	1,688,966	43,963,473	315,063,490	55.261	58.074
New Mexico, Nevada, and Utah	21,764,645	21,764,645	399,927,868	399,927,868	105,971.37	1,016,329	1,524,388	0	20,061,989	419,989,857	18.375	18.375
1918, all mines	54,200,536	55,470,912	1,593,109,933	1,613,278,593	221,802.57	16,111,621	5,199,271	1,678,272	107,099,497	1,720,378,090	29.393	29.083
Arizona	16,134,976	16,134,976	640,124,052	640,124,052	90,566.28	4,491,151	3,708,151	132,500	34,544,281	674,668,333	39.673	39.673
Michigan	9,634,045	10,349,052	192,728,637	201,974,025	0	0	0	0	0	201,974,025	20.005	19.516
Alaska, North Carolina, and Tennessee	1,189,240	1,189,240	75,794,437	75,794,437	235.39	778,399	0	0	3,580,904	79,375,341	63.734	63.734
California, Idaho, and Montana	5,631,102	6,186,471	303,948,787	314,872,059	34,039.94	10,023,551	413,000	892,887	50,973,660	365,845,719	53.977	50.897
New Mexico, Nevada, and Utah	21,611,173	21,611,173	380,514,020	380,514,020	96,960.96	818,520	1,078,120	652,885	18,000,652	398,514,672	17.607	17.607
1919, all mines	32,093,931	33,250,376	1,050,176,442	1,068,213,557	147,913.59	10,842,104	4,554,709	2,952,279	72,979,988	1,141,193,545	32.722	32.126
Arizona	11,700,574	11,700,574	468,784,792	468,784,792	72,191.92	3,471,867	4,486,600	0	27,531,799	496,316,591	40.065	40.065
Michigan	6,806,645	7,722,304	164,389,036	173,471,988	0	0	0	0	0	173,471,988	24.151	22.464
Alaska, North Carolina, and Tennessee	1,015,338	1,015,338	58,985,007	58,985,007	273.92	531,280	0	0	2,459,997	61,445,004	58.094	58.094
California, Idaho, and Montana	2,706,455	2,947,241	155,986,784	164,940,947	19,612.49	6,320,113	0	0	31,563,714	196,504,661	57.635	55.965
New Mexico, Nevada, and Utah	9,864,919	9,864,919	202,030,823	202,030,823	55,835.26	518,844	68,109	2,952,279	11,424,478	213,455,301	20.480	20.480

Year and State	Employment: Average number of men employed	Employment: Average number of days worked	Employment: Number of man-shifts	Employment: Number of man-hours	Output per man-shift[g]: Ore sold or treated (short tons)	Output per man-shift[g]: Ore and tailings, sold or treated (short tons)	Output per man-shift[g]: Copper from ore (pounds)	Output per man-shift[g]: Copper from ore and tailings (pounds)	Output per man-shift[g]: Copper plus copper equivalent of accessory metals (pounds)	Output per man-hour[h]: Ore sold or treated (short tons)	Output per man-hour[h]: Ore and tailings, sold or treated (short tons)	Output per man-hour[h]: Copper from ore (pounds)	Output per man-hour[h]: Copper from ore and tailings (pounds)	Output per man-hour[h]: Copper plus copper equivalent of accessory metals (pounds)
1917, all mines	43,857	302	13,241,371	105,930,968	3.888	3.945	118.834	120.583	128.630	0.486	0.493	14.854	15.073	16.079
Arizona	11,909	325	3,867,555	30,940,440	3.461	3.461	156.684	156.684	166.452	0.433	0.433	19.585	19.585	20.806
Michigan	11,886	310	3,680,210	29,441,680	2.869	3.067	58.789	61.255	61.255	0.359	0.383	7.349	7.657	7.657
Alaska, North Carolina, and Tennessee	1,051	324	340,054	2,720,432	3.317	3.317	277.148	277.148	291.094	0.415	0.415	34.643	34.643	36.387
California, Idaho, and Montana	14,017	253	3,540,317	28,322,536	1.314	1.319	72.596	76.575	88.993	0.164	0.165	9.075	9.572	11.124
New Mexico, Nevada, and Utah	4,994	363	1,813,235	14,505,880	12.003	12.003	220.560	220.560	231.625	1.500	1.500	27.570	27.570	28.953
1918, all mines	44,478	313	13,913,563	111,308,504	3.896	3.987	114.501	115.950	123.648	0.487	0.498	14.313	14.494	15.456
Arizona	11,989	343	4,110,275	32,882,200	3.926	3.926	155.738	155.738	164.142	0.491	0.491	19.467	19.467	20.518
Michigan	12,372	309	3,823,260	30,586,080	2.520	2.707	50.410	52.828	52.828	0.315	0.338	6.301	6.603	6.603
Alaska, North Carolina, and Tennessee	944	329	311,010	2,488,080	3.824	3.824	243.704	243.704	255.218	0.478	0.478	30.463	30.463	31.902
California, Idaho, and Montana	14,010	273	3,818,073	30,544,584	1.475	1.620	79.608	82.469	95.819	0.184	0.203	9.951	10.309	11.977
New Mexico, Nevada, and Utah	5,163	359	1,850,945	14,807,560	11.676	11.676	205.578	205.578	215.303	1.459	1.459	25.697	25.697	26.913
1919, all mines	31,727	286	9,066,489	72,531,912	3.540	3.667	115.831	117.820	125.869	0.442	0.458	14.479	14.727	15.734
Arizona	9,667	312	3,017,530	24,140,240	3.878	3.878	155.354	155.354	164.478	0.485	0.485	19.419	19.419	20.560
Michigan	9,245	296	2,736,365	21,890,920	2.487	2.822	60.076	63.395	63.395	0.311	0.353	7.509	7.924	7.924
Alaska, North Carolina, and Tennessee	934	309	288,269	2,306,152	3.522	3.522	204.618	204.618	213.152	0.440	0.440	25.577	25.577	26.644
California, Idaho, and Montana	8,581	223	1,915,988	15,327,904	1.413	1.538	81.413	86.087	102.560	0.177	0.192	10.177	10.761	12.820
New Mexico, Nevada, and Utah	3,300	336	1,108,337	8,866,696	8.901	8.901	182.283	182.283	192.591	1.113	1.113	22.785	22.785	24.074

See footnotes at end of table.

Table A-8.- PRODUCTION OF ORE, COPPER, AND COPPER EQUIVALENT, YIELD OF ORE, EMPLOYMENT, AND PRODUCTIVITY IN THE COPPER-MINING INDUSTRY, BY STATE, 1917-36[a] - *Continued*

Year and State	Production										Yield[b] (pounds of copper per ton) of -	
	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Metals recovered									
			Copper from ore (pounds)	Copper from ore and tailings[c] (pounds)	Gold[d] (fine ounces)	Silver[d] (fine ounces)	Lead[d] (pounds)	Zinc[d] (pounds)	Copper equivalent of accessory metals[e] (pounds)	Copper plus copper equivalent of accessory metals[f] (pounds)	Ore	Ore and tailings
1920, all mines	32,874,914	35,697,289	1,060,609,150	1,099,270,592	156,911.48	11,529,528	9,993,829	2,573,845	79,113,511	1,178,384,103	32.262	30.794
Arizona	13,171,566	13,171,566	489,388,499	489,388,499	74,286.07	3,587,099	9,222,405	0	29,994,504	519,383,003	37.155	37.155
Michigan	5,461,486	6,839,986	132,392,620	146,530,860	0	405,766	0	0	1,849,409	148,380,269	24.241	21.423
Alaska, North Carolina, and Tennessee	1,336,782	1,336,782	85,567,584	85,567,584	282.02	750,389	0	0	3,459,796	89,027,380	64.010	64.010
California, Idaho, and Montana	2,689,190	4,133,065	152,966,219	177,489,421	21,531.06	6,285,903	634,592	1,259,008	32,403,045	209,892,466	56.882	42.944
New Mexico, Nevada, and Utah	10,215,890	10,215,890	200,294,228	200,294,228	60,812.33	500,371	136,832	1,314,837	11,406,757	211,700,985	19.606	19.606
1921, all mines	11,747,262	12,513,552	416,107,107	428,471,546	44,206.72	5,326,705	8,067,251	6,038,083	35,716,476	464,188,022	35.422	34.241
Arizona	4,791,566	4,791,566	171,634,741	171,634,741	21,706.92	1,386,006	4,692,571	0	10,997,464	182,632,205	35.820	35.820
Michigan	2,930,605	3,297,605	76,705,748	82,007,484	0	297,895	0	0	1,357,753	83,365,237	26.174	24.869
Alaska, North Carolina, and Tennessee	1,039,317	1,039,317	71,058,466	71,058,466	238.17	633,993	0	0	2,923,118	73,981,584	68.370	68.370
California, Idaho, and Montana	891,457	1,290,747	52,201,980	59,264,683	9,645.17	2,891,158	3,374,680	6,038,083	18,127,871	77,392,554	58.558	45.915
New Mexico, Nevada, and Utah	2,094,317	2,094,317	44,506,172	44,506,172	12,616.46	117,653	0	0	2,310,270	46,816,442	21.251	21.251
1922, all mines	24,001,151	26,406,279	876,917,816	908,286,650	115,401.55	10,791,563	12,417,135	6,990,907	72,526,766	980,813,416	36.536	34.397
Arizona	10,292,440	10,292,440	384,482,812	384,482,812	57,943.62	2,945,969	8,374,511	210,624	24,564,767	409,047,579	37.356	37.356
Michigan	3,301,029	4,679,279	106,608,361	117,970,861	0	362,440	0	0	1,651,937	119,622,798	32.295	25.211
Alaska, North Carolina, and Tennessee	1,137,031	1,137,031	83,039,581	83,039,581	208.98	677,786	0	0	3,118,614	86,158,195	73.032	73.032
California, Idaho, and Montana	2,805,924	3,832,802	164,186,234	184,192,568	22,330.06	6,476,792	3,787,341	6,464,367	36,577,585	220,770,153	58.514	48.057
New Mexico, Nevada, and Utah	6,464,727	6,464,727	138,600,828	138,600,828	34,918.89	326,576	255,283	315,916	6,613,863	145,214,691	21.440	21.440

Year and State	Employment				Output per man-shift[g]					Output per man-hour[h]				
	Average number of men employed	Average number of days worked	Number of man-shifts	Number of man-hours	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)
1920, all mines	28,255	296	8,374,808	66,998,464	3.925	4.262	126.643	131.259	140.706	0.491	0.533	15.830	16.407	17.588
Arizona	9,576	338	3,236,423	25,891,384	4.070	4.070	151.213	151.213	160.481	0.509	0.509	18.902	18.902	20.060
Michigan	6,534	303	1,980,244	15,841,952	2.758	3.454	66.857	73.996	74.930	0.345	0.432	8.357	9.250	9.366
Alaska, North Carolina, and Tennessee	1,099	335	368,549	2,948,392	3.627	3.627	232.174	232.174	241.562	0.453	0.453	29.022	29.022	30.195
California, Idaho, and Montana	8,307	220	1,831,229	14,649,832	1.469	2.257	83.532	96.924	114.618	0.184	0.282	10.441	12.115	14.327
New Mexico, Nevada, and Utah	2,739	350	958,363	7,666,904	10.660	10.660	208.996	208.996	220.899	1.332	1.332	26.125	26.125	27.612
1921, all mines	15,392	217	3,337,450	26,699,600	3.520	3.749	124.678	128.383	139.085	0.440	0.469	15.585	16.048	17.386
Arizona	3,756	272	1,022,061	8,176,488	4.688	4.688	167.930	167.930	178.690	0.586	0.586	20.991	20.991	22.336
Michigan	4,910	240	1,176,214	9,409,712	2.492	2.804	65.214	69.722	70.876	0.311	0.350	8.152	8.715	8.859
Alaska, North Carolina, and Tennessee	896	315	281,920	2,255,360	3.687	3.687	252.052	252.052	262,420	0.461	0.461	31.506	31.506	32.803
California, Idaho, and Montana	4,464	151	672,375	5,379,000	1.326	1.920	77.638	88.142	115.103	0.166	0.240	9.705	11.018	14.388
New Mexico, Nevada, and Utah	1,366	135	184,880	1,479,040	11.328	11.328	240.730	240.730	253.226	1.416	1.416	30.091	30.091	31.653
1922, all mines	23,447	279	6,548,531	52,388,248	3.665	4.032	133.911	138.701	149.776	0.458	0.504	16.739	17.338	18.722
Arizona	6,541	319	2,088,653	16,709,224	4.928	4.928	184.082	184.082	195.843	0.616	0.616	23.010	23.010	24.480
Michigan	6,480	272	1,761,437	14,091,496	1.874	2.657	60.524	66.974	67.912	0.234	0.332	7.565	8.372	8.489
Alaska, North Carolina, and Tennessee	886	326	288,805	2,310,440	3.937	3.937	287.528	287.528	298.327	0.492	0.492	35.941	35.941	37.291
California, Idaho, and Montana	7,893	255	2,009,143	16,073,144	1.397	1.908	81.720	91.677	109.883	0.175	0.238	10.215	11.460	13.735
New Mexico, Nevada, and Utah	1,647	243	400,493	3,203,944	16.142	16.142	346.076	346.076	362.590	2.018	2.018	43.259	43.259	45.324

See footnotes at end of table.

Table A-8.- PRODUCTION OF ORE, COPPER, AND COPPER EQUIVALENT, YIELD OF ORE, EMPLOYMENT, AND PRODUCTIVITY IN THE COPPER-MINING INDUSTRY, BY STATE, 1917-36[a] - *Continued*

Year and State	Production										Yield[b] (pounds of copper per ton) of -	
	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Metals recovered									
			Copper from ore (pounds)	Copper from ore and tailings[c] (pounds)	Gold[d] (fine ounces)	Silver[d] (fine ounces)	Lead[d] (pounds)	Zinc[d] (pounds)	Copper equivalent of accessory metals[e] (pounds)	Copper plus copper equivalent of accessory metals[f] (pounds)	Ore	Ore and tailings
1923, all mines	41,743,897	44,752,669	1,353,386,332	1,387,539,501	273,904.82	14,887,194	11,124,930	7,123,173	113,086,199	1,500,625,700	32.421	31.005
Arizona	16,285,698	16,265,698	587,780,784	587,780,784	138,207.43	5,139,632	8,442,252	519,103	45,996,482	633,777,266	36.092	36.092
Michigan	3,408,970	5,152,070	118,095,317	134,996,517	0	218,420	0	0	995,520	135,992,037	34.643	26.202
Alaska, North Carolina, and Tennessee	1,503,322	1,503,322	101,594,912	101,594,912	320.00	700,045	0	0	3,235,677	104,830,589	67.580	67.580
California, Idaho, and Montana	3,673,678	4,939,350	220,875,308	238,127,277	28,968.16	8,013,415	2,300,451	6,269,118	43,911,299	282,038,576	60.124	48.210
New Mexico, Nevada, and Utah	16,872,229	16;872,229	325,040,011	325,040,011	106,409.23	815,682	382,227	334,952	18,947,221	343,987,232	19.265	19.265
1924, all mines	46,577,894	48,424,163	1,491,003,012	1,518,067,116	296,989.49	14,684,650	7,723,549	9,929,631	115,355,357	1,633,422,473	32.011	31.349
Arizona	19,425,415	19,425,415	663,023,254	663,023,254	146,437.66	5,402,720	5,461,412	0	47,110,346	710,133,600	34.132	34.132
Michigan	3,561,921	5,248,921	115,566,797	134,193,797	0	123,090	0	0	561,022	134,754,819	32.445	25.566
Alaska, North Carolina, and Tennessee	1,446,868	1,446,868	92,279,675	92,279,675	328.95	667,258	0	0	3,087,498	95,367,173	63.779	63.779
California, Idaho, and Montana	3,724,632	3,883,901	249,729,078	258,166,182	36,811.69	7,624,610	2,043,232	9,910,606	44,614,401	302,780,583	67.048	66.471
New Mexico, Nevada, and Utah	18,419,058	18,419,058	370,404,208	370,404,208	113,411.19	866,972	218,905	19,025	19,982,090	390,386,298	20.110	20.110
1925, all mines	49,602,217	52,394,576	1,557,979,177	1,592,498,146	316,905.24	15,678,043	13,917,542	10,121,430	124,909,382	1,717,407,528	31.409	30.394
Arizona	20,578,989	20,578,989	696,200,118	696,200,118	157,323.72	5,848,097	13,313,415	6,712,800	56,089,426	752,289,544	33.831	33.831
Michigan	4,261,483	6,993,088	127,626,404	154,583,404	0	139,499	0	0	635,812	155,219,216	29.949	22.105
Alaska, North Carolina, and Tennessee	1,565,509	1,565,509	92,219,398	92,219,398	358.99	717,036	0	0	3,318,602	95,538,000	58.907	58.907
California, Idaho, and Montana	3,626,092	3,686,846	270,513,996	278,075,965	40,962.80	8,046,612	348,871	3,201,404	43,840,859	321,916,824	74.602	75.424
New Mexico, Nevada, and Utah	19,570,144	19,570,144	371,419,261	371,419,261	118,259.73	926,799	255,256	207,226	21,024,683	392,443,944	18.979	18.979

Year and State	Employment				Output per man-shift[g]					Output per man-hour[h]				
	Average number of men employed	Average number of days worked	Number of man-shifts	Number of man-hours	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)
1923, all mines	29,337	315	9,226,683	73,813,464	4.524	4.850	146.682	150.383	162.640	0.566	0.606	18.335	18.798	20.330
Arizona	10,099	340	3,428,983	27,431,864	4.749	4.749	171.415	171.415	184.830	0.594	0.594	21.427	21.427	23.104
Michigan	5,243	312	1,636,644	13,093,152	2.083	3.148	72.157	82.484	83.092	0.260	0.393	9.020	10.310	10.387
Alaska, North Carolina, and Tennessee	1,087	335	364,574	2,916,592	4.124	4.124	278.667	278.667	287.543	0.515	0.515	34.833	34.833	35.943
California, Idaho, and Montana	9,369	268	2,510,720	20,085,760	1.463	1.967	87.973	94.844	112.334	0.183	0.246	10.997	11.856	14.042
New Mexico, Nevada, and Utah	3,539	363	1,285,762	10,286,096	13.122	13.122	252.800	252.800	267.536	1.640	1.640	31.600	31.600	33.442
1924, all mines	29,527	316	9,343,359	74,746,872	4.985	5.183	159.579	162.476	174.822	0.623	0.648	19.947	20.309	21.853
Arizona	10,987	333	3,654,216	29,233,728	5.316	5.316	181.441	181.441	194.333	0.664	0.664	22.680	22.680	24.292
Michigan	5,639	286	1,612,791	12,902,328	2.209	3.255	71.656	83.206	83.554	0.276	0.407	8.957	10.401	10.444
Alaska, North Carolina, and Tennessee	998	331	330,561	2,644,488	4.377	4.377	279.161	279.161	288.501	0.547	0.547	34.895	34.895	36.063
California, Idaho, and Montana	7,777	296	2,298,189	18,385,512	1.621	1.690	108.663	112.335	131.747	0.203	0.211	13.583	14.042	16.468
New Mexico, Nevada, and Utah	4,126	351	1,447,602	11,580,816	12.724	12.724	255.874	255.874	269.678	1.590	1.590	31.984	31.984	33.710
1925, all mines	31,787	305	9,709,811	77,678,488	5.108	5.396	160.454	164.009	176.873	0.639	0.675	20.057	20.501	22.109
Arizona	10,701	319	3,418,278	27,346,224	6.020	6.020	203.670	203.670	220.079	0.753	0.753	25.459	25.459	27.510
Michigan	5,894	292	1,718,637	13,749,096	2.480	4.069	74.260	89.945	90.315	0.310	0.509	9.283	11.243	11.289
Alaska, North Carolina, and Tennessee	1,011	333	336,963	2,695,704	4.646	4.646	273.678	273.678	283.527	0.581	0.581	34.210	34.210	35.441
California, Idaho, and Montana	9,620	267	2,572,019	20,576,152	1.410	1.433	105.176	108.116	125.161	0.176	0.179	13.147	13.514	15.645
New Mexico, Nevada, and Utah	4,561	365	1,663,914	13,311,312	11.762	11.762	223.220	223.220	235.856	1.470	1.470	27.903	27.903	29.482

See footnotes at end of table.

Table A-8.- PRODUCTION OF ORE, COPPER, AND COPPER EQUIVALENT, YIELD OF ORE, EMPLOYMENT, AND PRODUCTIVITY IN THE COPPER-MINING INDUSTRY, BY STATE, 1917-36[a] - *Continued*

Year and State	Production										Yield[b] (pounds of copper per ton) of -	
	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Metals recovered									
			Copper from ore (pounds)	Copper from ore and tailings[c] (pounds)	Gold[d] (fine ounces)	Silver[d] (fine ounces)	Lead[d] (pounds)	Zinc[d] (pounds)	Copper equivalent of accessory metals[e] (pounds)	Copper plus copper equivalent of accessory metals[f] (pounds)	Ore	Ore and tailings
1926, all mines	53,908,697	57,018,844	1,608,504,802	1,650,701,343	353,130.14	15,480,610	12,408,675	8,301,426	127,849,219	1,778,550,562	29.838	28.950
Arizona	21,955,110	21,955,110	708,187,130	708,187,130	177,467.49	6,038,532	11,454,869	5,888,890	58,814,369	767,001,499	32.256	32.256
Michigan	4,885,505	7,641,682	144,393,489	175,381,565	0	105,242	0	0	479,674	175,861,239	29.555	22.951
Alaska, North Carolina, and Tennessee	1,285,747	1,285,747	86,072,249	86,072,249	417.09	695,688	0	0	3,229,471	89,301,720	66.943	66.943
California, Idaho, and Montana	3,389,464	3,743,434	251,941,108	263,149,573	36,253.67	7,518,269	155,048	1,871,458	40,169,572	303,319,145	74.331	70.296
New Mexico, Nevada, and Utah	22,392,871	22,392,871	417,910,826	417,910,826	138,991.89	1,122,879	798,758	541,078	25,156,133	443,066,959	18.663	18.663
1927, all mines	53,427,937	56,475,825	1,551,370,797	1,585,688,167	350,273.63	14,026,725	14,938,257	2,683,855	119,443,947	1,705,132,114	29.037	28.077
Arizona	21,670,371	21,670,371	672,290,680	672,290,680	162,146.72	5,943,694	10,511,017	867,024	53,884,789	726,175,469	31.023	31.023
Michigan	5,027,334	8,046,334	148,820,775	177,537,775	0	97,193	0	0	442,989	177,980,764	29.602	22.064
Alaska, North Carolina, and Tennessee	1,312,392	1,312,392	74,874,109	74,874,109	426.47	503,654	0	0	2,355,533	77,229,642	57.052	57.052
California, Idaho, and Montana	3,593,458	3,622,346	229,726,787	235,327,157	36,074.95	6,256,056	77,251	695,886	33,892,679	269,219,836	63.929	64.965
New Mexico, Nevada, and Utah	21,824,382	21,824,382	425,658,446	425,658,446	151,625.49	1,226,128	4,349,989	1,120,945	28,867,957	454,526,403	19.504	19.504
1928, all mines	59,081,963	61,737,797	1,711,742,268	1,742,748,465	393,266.76	13,855,639	14,249,593	2,549,647	124,416,738	1,867,165,203	28.972	28.228
Arizona	22,557,613	22,557,613	723,538,641	723,538,641	161,943.79	5,924,935	5,833,275	0	51,799,875	775,338,516	32.075	32.075
Michigan	4,732,658	7,361,658	148,546,704	178,442,704	0	17,153	0	0	78,180	178,520,884	31.388	24.239
Alaska, North Carolina, and Tennessee	1,411,852	1,411,852	65,387,984	65,387,984	611.99	426,698	0	0	2,030,867	67,418,851	46.314	46.314
California, Idaho, and Montana	3,855,894	3,882,728	256,710,821	257,821,018	39,643.28	5,889,370	42,924	577,452	32,663,696	290,484,714	66.576	66.402
New Mexico, Nevada, and Utah	26,523,946	26,523,946	517,558,118	517,558,118	191,067.70	1,597.483	8,373,394	1,972,195	37,844,120	555,402,238	19.513	19.513

Year and State	Employment				Output per man-shift[g]					Output per man-hour[h]				
	Average number of men employed	Average number of days worked	Number of man-shifts	Number of man-hours	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)
1926, all mines	30,636	325	9,968,867	80,422,426	5.408	5.720	161.353	165.586	178.411	0.670	0.709	20.001	20.525	22.115
Arizona	10,614	336	3,563,899	28,511,192	6.160	6.160	198.711	198.711	215.214	0.770	0.770	24.839	24.839	26.902
Michigan	6,311	313	1,976,650	16,472,170	2.472	3.866	73.050	88.727	88.969	0.297	0.464	8.766	10.647	10.676
Alaska, North Carolina, and Tennessee	1,037	333	345,563	2,777,024	3.721	3.721	249.078	249.078	258.424	0.463	0.463	30.994	30.994	32.157
California, Idaho, and Montana	7,944	303	2,409,726	19,277,808	1.407	1.553	104.552	109.203	125.873	0.176	0.194	13.069	13.650	15.734
New Mexico, Nevada, and Utah	4,730	354	1,673,029	13,384,232	13.385	13.385	249.793	249.793	264.829	1.673	1.673	31.224	31.224	33.104
1927, all mines	29,242	321	9,389,029	75,814,219	5.690	6.015	165.232	168.887	181.609	0.705	0.745	20.463	20.915	22.491
Arizona	9,927	335	3,324,553	26,596,424	6.518	6.518	202.220	202.220	218.428	0.815	0.815	25.277	25.277	27.304
Michigan	6,370	298	1,897,735	15,847,367	2.649	4.240	78.420	93.552	93.786	0.317	0.508	9.391	11.203	11.231
Alaska, North Carolina, and Tennessee	938	340	318,921	2,587,868	4.115	4.115	234.773	234.773	242.159	0.507	0.507	28.933	28.933	29.843
California, Idaho, and Montana	7,823	304	2,375,864	19,006,912	1.512	1.525	96.692	99.049	113.314	0.189	0.191	12.086	12.381	14.164
New Mexico, Nevada, and Utah	4,184	352	1,471,956	11,775,648	14.827	14.827	289.179	289.179	308.791	1.853	1.853	36.147	36.147	38.599
1928, all mines	30,522	314	9,569,144	77,245,104	6.174	6.452	178.881	182.122	195.124	0.765	0.799	22.160	22.561	24.172
Arizona	9,776	348	3,400,193	27,201,544	6.634	6.634	212.793	212.793	228.028	0.829	0.829	26.599	26.599	28.503
Michigan	5,852	309	1,806,248	15,094,486	2.620	4.076	82.240	98.792	98.835	0.314	0.488	9.841	11.822	11.827
Alaska, North Carolina, and Tennessee	1,015	337	342,383	2,786,514	4.124	4.124	190.979	190.979	196.911	0.507	0.507	23.466	23.466	24.195
California, Idaho, and Montana	9,147	257	2,347,487	18,779,896	1.643	1.654	109.356	109.829	123.743	0.205	0.207	13.669	13.729	15.468
New Mexico, Nevada, and Utah	4,732	354	1,672,833	13,382,664	15.856	15.856	309.390	309.390	332.013	1.982	1.982	38.674	38.674	41.502

See footnotes at end of table.

Table A-8.- PRODUCTION OF ORE, COPPER, AND COPPER EQUIVALENT, YIELD OF ORE, EMPLOYMENT, AND PRODUCTIVITY IN THE COPPER-MINING INDUSTRY, BY STATE, 1917-36[a] - *Continued*

Year and State	Production										Yield[b] (pounds of copper per ton) of -	
			Metals recovered									
	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings[c] (pounds)	Gold[d] (fine ounces)	Silver[d] (fine ounces)	Lead[d] (pounds)	Zinc[d] (pounds)	Copper equivalent of accessory metals[e] (pounds)	Copper plus copper equivalent of accessory metals[f] (pounds)	Ore	Ore and tailings
1929, all mines	65,197,640	67,670,562	1,873,967,159	1,916,037,768	417,835.96	16,207,532	12,853,292	2,084,269	137,919,765	2,053,957,533	28.743	28.314
Arizona	25,588,400	25,588,400	819,340,720	819,340,720	178,770.64	6,480,990	2,040,899	0	55,384,614	874,725,334	32.020	32.020
Michigan	5,154,180	7,598,180	152,891,218	186,402,218	0	17,101	0	0	77,943	186,480,161	29.664	24.532
Alaska, North Carolina, and Tennessee	1,488,623	1,488,623	69,475,186	69,475,186	745.02	438,771	0	0	2,104,600	71,579,786	46.671	46.671
California, Idaho, and Montana	4,594,377	4,623,299	309,214,784	317,774,393	42,028.41	7,569,286	1,000	0	40,409,523	358,183,916	67.303	68.733
New Mexico, Nevada, and Utah	28,372,060	28,372,060	523,045,251	523,045,251	196,291.89	1,701,384	10,811,393	2,084,269	39,943,085	562,988,336	18.435	18.435
1930, all mines	45,108,280	46,683,016	1,312,517,806	1,342,846,987	291,160.66	10,975,032	3,940,680	2,472,101	93,322,387	1,436,169,374	29.097	28.765
Arizona	19,582,530	19,582,530	567,575,110	567,575,110	129,824.81	4,765,207	302,745	0	40,078,963	607,654,073	28.984	28.984
Michigan	5,114,036	6,659,036	151,539,413	169,381,413	0	0	0	0	0	169,381,413	29.632	25.436
Alaska, North Carolina, and Tennessee	1,410,548	1,410,548	66,328,773	66,328,773	1,218.47	379,587	0	2,150,000	2,764,347	69,093,120	47.023	47.023
California, Idaho, and Montana	2,832,920	2,862,656	198,004,563	210,491,744	30,019.46	4,814,990	0	0	26,166,979	236,658,723	69.894	73.530
New Mexico, Nevada, and Utah	16,168,246	16,168,246	329,069,947	329,069,947	130,097.90	1,015,248	3,637,935	322,101	24,312,098	353,382,045	20.353	20.353
1931, all mines	33,863,385	33,881,998	1,017,938,820	1,027,330,781	205,044.68	7,897,080	510,384	2,097,833	65,844,346	1,093,175,127	30.060	30.321
Arizona	13,569,396	13,569,396	397,370,313	397,370,313	80,725.76	2,880,976	503,242	0	24,656,604	422,026,917	29.284	29.284
Michigan	3,570,748	3,570,748	118,059,491	118,059,491	0	0	0	0	0	118,059,491	33.063	33.063
Alaska, North Carolina, and Tennessee	664,306	664,306	44,752,089	44,752,089	684.40	255,162	0	2,092,000	2,098,864	46,850,953	67.367	67.367
California, Idaho, and Montana	2,272,136	2,290,749	186,458,163	195,850,124	23,549.82	3,985,296	0	0	21,475,668	217,325,792	82.063	85.496
New Mexico, Nevada, and Utah	13,786,799	13,786,799	271,298,764	271,298,764	100,084.70	775,646	7,142	5,833	17,613,210	288,911,974	19.678	19.678

Year and State	Employment				Output per man-shift[g]					Output per man-hour[h]				
	Average number of men employed	Average number of days worked	Number of man-shifts	Number of man-hours	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)
1929, all mines	35,499	318	11,286,433	91,035,980	5.777	5.996	166.037	169.765	181.985	0.716	0.743	20.585	21.047	22.562
Arizona	11,246	339	3,817,708	30,541,664	6.703	6.703	214.616	214.616	229.123	0.838	0.838	26.827	26.827	28.640
Michigan	6,660	314	2,089,988	17,444,486	2.466	3.636	73.154	89.188	89.225	0.295	0.436	8.764	10.685	10.690
Alaska, North Carolina, and Tennessee	1,032	329	339,570	2,736,494	4.384	4.384	204.598	204.598	210.795	0.544	0.544	25.388	25.388	26.157
California, Idaho, and Montana	10,972	281	3,084,510	24,676,080	1.489	1.499	100.248	103.023	116.123	0.186	0.187	12.531	12.878	14.515
New Mexico, Nevada, and Utah	5,589	350	1,954,657	15,637,256	14.515	14.515	267.589	267.589	288.024	1.814	1.814	33.449	33.449	36.003
1930, all mines	25,663	300	7,706,440	62,323,121	5.853	6.058	170.314	174.250	186.360	0.724	0.749	21.060	21.547	23.044
Arizona	8,594	300	2,577,304	20,618,432	7.598	7.598	220.220	220.220	235.771	0.950	0.950	27.528	27.528	29.471
Michigan	6,600	291	1,918,461	15,966,463	2.666	3.471	78.990	88.290	88.290	0.320	0.417	9.491	10.609	10.609
Alaska, North Carolina, and Tennessee	981	318	312,378	2,551,850	4.516	4.516	212.335	212.335	221.184	0.553	0.553	25.992	25.992	27.076
California, Idaho, and Montana	5,847	280	1,636,120	13,088,960	1.731	1.750	121.021	128.653	144.646	0.216	0.219	15.128	16.082	18.081
New Mexico, Nevada, and Utah	3,641	347	1,262,177	10,097,416	12.810	12.810	260.716	260.716	279.978	1.601	1.601	32.590	32.590	34.997
1931, all mines	18,764	262	4,922,135	39,779,776	6.880	6.884	206.808	208.716	222.094	0.851	0.852	25.589	25.825	27.481
Arizona	5,494	265	1,455,263	11,642,104	9.324	9.324	273.057	273.057	290.000	1.166	1.166	34.132	34.132	36.250
Michigan	5,814	208	1,211,928	10,098,120	2.946	2.946	97.415	97.415	97.415	0.354	0.354	11.691	11.691	11.691
Alaska, North Carolina, and Tennessee	668	242	161,925	1,295,400	4.103	4.103	276.375	276.375	289.337	0.513	0.513	34.547	34.547	36.167
California, Idaho, and Montana	4,216	301	1,269,460	10,155,680	1.790	1.805	146.880	154.278	171.195	0.224	0.226	18.360	19.285	21.399
New Mexico, Nevada, and Utah	2,572	320	823,559	6,588,472	16.741	16.741	329.422	329.422	350.809	2.093	2.093	41.178	41.178	43.851

See footnotes at end of table.

Table A-8.- PRODUCTION OF ORE, COPPER, AND COPPER EQUIVALENT, YIELD OF ORE, EMPLOYMENT, AND PRODUCTIVITY IN THE COPPER-MINING INDUSTRY, BY STATE, 1917-36[a] - *Continued*

Year and State	Production										Yield[b] (pounds of copper per ton) of -	
	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Metals recovered									
			Copper from ore (pounds)	Copper from ore and tailings[c] (pounds)	Gold[d] (fine ounces)	Silver[d] (fine ounces)	Lead[d] (pounds)	Zinc[d] (pounds)	Copper equivalent of accessory metals[e] (pounds)	Copper plus copper equivalent of accessory metals[f] (pounds)	Ore	Ore and tailings
1932, all mines	12,162,502	12,176,845	445,803,526	455,470,167	91,063.63	4,034,749	861,767	786,000	31,808,784	487,278,951	36.654	37.405
Arizona	4,343,944	4,343,944	181,815,279	181,815,279	42,905.27	1,939,389	861,767	0	15,171,379	196,986,658	41.855	41.855
Michigan	1,142,775	1,142,775	54,396,108	54,396,108	0	71,408	0	0	325,465	54,721,573	47.600	47.600
Alaska, North Carolina, and Tennessee	298,206	298,206	19,269,527	19,269,527	290.98	109,155	0	786,000	853,894	20,123,421	64.618	64.618
California, Idaho, and Montana	672,235	686,578	75,975,257	85,641,898	4,321.71	1,587,952	0	0	7,845,290	93,487,188	113.019	124.737
New Mexico, Nevada, and Utah	5,705,342	5,705,342	114,347,355	114,347,355	43,545.67	326,845	0	0	7,612,756	121,960,111	20.042	20.042
1933, all mines	8,243,327	8,243,327	348,589,323	357,745,833	100,682.53	4,204,040	215,277	693,000	33,671,298	391,417,131	42.287	43.398
Arizona	890,136	890,136	113,759,721	113,759,721	48,469.27	2,142,650	214,757	0	16,655,700	130,415,421	127.800	127.800
Michigan	697,158	697,158	46,853,130	46,853,130	0	0	0	0	0	46,853,130	67.206	67.206
Alaska, North Carolina, and Tennessee	356,246	356,246	12,504,320	12,504,320	335.64	51,269	0	693,000	559,013	13,063,333	35.100	35.100
California, Idaho, and Montana	476,182	476,182	55,746,197	64,902,707	1,573.97	1,605,961	520	0	7,541,186	72,443,893	117.069	136.298
New Mexico, Nevada, and Utah	5,823,605	5,823,605	119,725,955	119,725,955	50,303.65	404,160	0	0	8,915,399	128,641,354	20.559	20.559
1934, all mines	11,502,188	11,502,188	439,079,547	444,246,852	141,840.53	5,786,300	125,558	2,246,000	47,262,464	491,509,316	38.174	38.623
Arizona	2,820,287	2,820,287	177,206,935	177,206,935	76,612.91	3,463,735	125,558	0	26,603,366	203,810,301	62.833	62.833
Michigan	700,055	700,055	48,215,859	48,215,859	0	0	0	0	0	48,215,859	68.874	68.874
Alaska, North Carolina, and Tennessee	610,511	610,511	17,556,800	17,556,800	655.00	70,648	0	2,246,000	1,315,558	18,872,358	28.758	28.758
California, Idaho, and Montana	457,213	457,213	56,635,000	61,802,305	1,359.00	1,793,184	0	0	8,364,107	70,166,412	123.870	135.172
New Mexico, Nevada, and Utah	6,914,122	6,914,122	139,464,953	139,464,953	63,213.62	458,733	0	0	10,979,433	150,444,386	20.171	20.171

Year and State	Employment				Output per man-shift[g]					Output per man-hour[h]				
	Average number of men employed	Average number of days worked	Number of man-shifts	Number of man-hours	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)
1932, all mines	9,086	228	2,072,046	16,894,743	5.870	5.877	215.151	219.817	235.168	0.720	0.721	26.387	26.959	28.842
Arizona	2,678	198	529,302	4,234,416	8.207	8.207	343.500	343.500	372.163	1.026	1.026	42.938	42.938	46.520
Michigan	2,491	279	694,811	5,876,863	1.645	1.645	78.289	78.289	78.757	0.194	0.194	9.256	9.256	9.311
Alaska, North Carolina, and Tennessee	242	219	53,049	424,392	5.621	5.621	363.240	363.240	379.336	0.703	0.703	45.405	45.405	47.417
California, Idaho, and Montana	2,092	214	446,994	3,575,952	1.504	1.536	169.969	191.595	209.146	0.188	0.192	21.246	23.949	26.143
New Mexico, Nevada, and Utah	1,583	220	347,890	2,783,120	16.400	16.400	328.688	328.688	350.571	2.050	2.050	41.086	41.086	43.821
1933, all mines	6,652	212	1,410,414	11,283,312	5.845	5.845	247.154	253.646	277.519	0.731	0.731	30.894	31.706	34.690
Arizona	1,738	203	353,235	2,825,880	2.520	2.520	322.051	322.051	369.203	0.315	0.315	40.256	40.256	46.150
Michigan	1,235	284	350,588	2,804,704	1.989	1.989	133.642	133.642	133.642	0.249	0.249	16.705	16.705	16.705
Alaska, North Carolina, and Tennessee	257	246	63,245	505,960	5.633	5.633	197.712	197.712	206.551	0.704	0.704	24.714	24.714	25.819
California, Idaho, and Montana	2,121	168	356,438	2,851,504	1.336	1.336	156.398	182.087	203.244	0.167	0.167	19.550	22.761	25.406
New Mexico, Nevada, and Utah	1,301	221	286,908	2,295,264	20.298	20.298	417.297	417.297	448.371	2.537	2.537	52.162	52.162	56.046
1934, all mines	7,749	215	1,668,104	13,344,832	6.895	6.895	263.221	266.318	294.651	0.862	0.862	32.903	33.290	36.831
Arizona	2,059	235	482,887	3,863,096	5.840	5.840	366.974	366.974	422.066	0.730	0.730	45.872	45.872	52.758
Michigan	1,423	270	383,701	3,069,608	1.824	1.824	125.660	125.660	125.660	0.228	0.228	15.707	15.707	15.707
Alaska, North Carolina, and Tennessee	330	270	89,091	712,728	6.853	6.853	197.066	197.066	211.832	0.857	0.857	24.633	24.633	26.479
California, Idaho, and Montana	2,561	148	380,210	3,041,680	1.203	1.203	148.957	162.548	184.546	0.150	0.150	18.620	20.318	23.068
New Mexico, Nevada, and Utah	1,376	241	332,215	2,657,720	20.812	20.812	419.803	419.803	452.852	2.602	2.602	52.475	52.475	56.607

See footnotes at end of table.

Table A-8.- PRODUCTION OF ORE, COPPER, AND COPPER EQUIVALENT, YIELD OF ORE, EMPLOYMENT, AND PRODUCTIVITY IN THE COPPER-MINING INDUSTRY, BY STATE, 1917-36[a] - *Continued*

Year and State	Production										Yield[b] (pounds of copper per ton) of -	
	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Metals recovered									
			Copper from ore (pounds)	Copper from ore and tailings[c] (pounds)	Gold[d] (fine ounces)	Silver[d] (fine ounces)	Lead[d] (pounds)	Zinc[d] (pounds)	Copper equivalent of accessory metals[e] (pounds)	Copper plus copper equivalent of accessory metals[f] (pounds)	Ore	Ore and tailings
1935, all mines	18,245,802	18,868,802	692,535,832	711,313,757	218,814.12	9,858,104	481,906	2,600,000	76,910,168	788,223,925	37.956	37.698
Arizona	6,011,483	6,011,483	275,925,592	275,925,592	113,252.38	4,565,835	469,775	0	36,897,923	312,823,515	45.900	45.900
Michigan	753,803	1,376,803	54,990,689	64,108,689	0	4,219	0	0	19,229	64,127,918	72.951	46.563
Alaska, North Carolina, and Tennessee	711,266	711,266	32,130,587	32,130,587	603.00	188,517	0	2,600,000	1,987,554	34,118,141	45.174	45.174
California, Idaho, and Montana	1,348,483	1,348,483	144,895,282	154,555,207	5,007.43	4,389,027	0	0	20,708,515	175,263,722	107.451	114.614
New Mexico, Nevada, and Utah	9,420,767	9,420,767	184,593,682	184,593,682	99,951.31	710,506	12,131	0	17,296,947	201,890,629	19.594	19.594
1936, all mines	36,551,791	38,098,791	1,123,032,954	1,151,785,142	365,020.63	13,542,079	611,704	2,890,000	114,420,927	1,266,206,069	30.724	30.232
Arizona	12,798,617	12,798,617	417,151,910	417,151,910	165,218.83	5,990,755	611,704	0	50,748,816	467,900,726	32.594	32.594
Michigan	1,678,600	3,225,600	76,801,019	95,968,019	0	0	0	0	0	95,968,019	45.753	29.752
Alaska, North Carolina, and Tennessee	825,063	825,063	56,079,800	56,079,800	525.00	376,286	0	2,890,000	2,948,799	59,028,599	67.970	67.970
California, Idaho, and Montana	2,882,460	2,882,460	215,664,780	225,249,968	20,415.82	5,847,290	0	0	29,521,628	254,771,596	74.820	78.145
New Mexico, Nevada, and Utah	18,367,051	18,367,051	357,335,445	357,335,445	178,860.98	1,327,748	0	0	31,201,684	388,537,129	19.455	19.455

Year and State	Employment				Output per man-shift[g]					Output per man-hour[h]				
	Average number of men employed	Average number of days worked	Number of man-shifts	Number of man-hours	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)	Ore sold or treated (short tons)	Ore and tailings, sold or treated (short tons)	Copper from ore (pounds)	Copper from ore and tailings (pounds)	Copper plus copper equivalent of accessory metals (pounds)
1935, all mines	8,845	271	2,394,755	19,158,040	7.619	7.879	289.189	297.030	329.146	0.952	0.985	36.149	37.129	41.143
Arizona	3,086	247	763,772	6,110,174	7.871	7.871	361.267	361.267	409.577	0.984	0.984	45.158	45.158	51.197
Michigan	1,547	265	410,242	3,281,939	1.837	3.356	134.045	156.270	156.317	0.230	0.420	16.756	19.534	19.540
Alaska, North Carolina, and Tennessee	434	263	114,199	913,591	6.228	6.228	281.356	281.356	298.760	0.779	0.779	35.170	35.170	37.345
California, Idaho, and Montana	2,699	283	764,564	6,116,512	1.764	1.764	189.514	202.148	229.234	0.220	0.220	23.689	25.269	28.654
New Mexico, Nevada, and Utah	1,079	317	341,978	2,735,824	27.548	27.548	539.782	539.782	590.361	3.443	3.443	67.473	67.473	73.795
1936, all mines	12,341	311	3,836,618	30,694,017	9.527	9.930	292.714	300.208	330.032	1.191	1.241	36.588	37.525	41.253
Arizona	3,696	311	1,149,455	9,195,653	11.135	11.135	362.913	362.913	407.063	1.392	1.392	45.364	45.364	50.883
Michigan	1,838	309	568,252	4,546,021	2.954	5.676	135.153	168.883	168.883	0.369	0.710	16.894	21.110	21.110
Alaska, North Carolina, and Tennessee	510	269	137,202	1,097,611	6.013	6.013	408.739	408.739	430.231	0.752	0.752	51.093	51.093	53.779
California, Idaho, and Montana	4,705	301	1,415,369	11,324,552	2.037	2.037	152.374	159.146	180.004	0.255	0.255	19.044	19.890	22.497
New Mexico, Nevada, and Utah	1,592	356	566,340	4,530,180	32.431	32.431	630.956	630.956	686.049	4.054	4.054	78.879	78.879	85.766

[a]Same as table A-7, ftn. a. Because only the major producers are covered in this table, the mines in Colorado, Oregon, and Washington are excluded.

[b-h]Same as table A-7, ftns. b-f, h, and i, respectively.

Table A-9.- PRODUCTION, EMPLOYMENT, AND OUTPUT PER MAN-SHIFT AT OPEN-CUT COPPER MINES, 1914-36[a]

Year	Production				Employment		Output per man-shift	
	Material handled			Copper recoverable (pounds)	Average number of men	Total man-shifts	Ore (short tons)	Copper (pounds)
	Overburden removed (short tons)	Ore sold or treated (short tons)	Stripping ratio[b]					
	(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)
1914	21,404,787	11,051,663	1.94	223,115,689	3,406	1,205,158	9.170	185.134
1915	21,615,845	13,986,381	1.55	281,274,919	3,487	1,242,990	11.252	226.289
1916	24,655,664	18,103,532	1.36	357,807,015	3,637	1,323,319	13.680	270.386
1917	19,595,303	21,106,608	0.93	385,938,976	4,066	1,478,585	14.275	261.019
1918	17,718,680	22,133,143	0.80	392,896,417	4,125	1,496,252	14.792	262.587
1919	12,906,837	11,081,070	1.16	230,623,058	2,702	956,917	11.580	241.006
1920	13,842,572	11,879,461	1.17	236,667,022	2,579	934,231	12.716	253.328
1921	2,119,274	3,011,450	0.70	64,129,336	1,357	183,726	16.391	349.049
1922	4,912,495	7,746,579	0.63	161,810,473	1,482	363,487	21.312	445.162
1923	13,424,574	18,433,838	0.73	351,645,795	3,283	1,193,947	15.439	294.524
1924	24,757,879	21,298,354	1.16	415,399,308	3,986	1,399,284	15.221	296.866
1925	29,906,974	22,738,583	1.32	424,526,891	4,384	1,600,005	14.212	265.328
1926	30,063,576	25,264,255	1.19	479,163,193	4,414	1,611,153	15.681	297.404
1927	26,000,635	24,062,939	1.08	460,795,448	3,787	1,344,769	17.894	342.658
1928	28,590,404	28,615,549	1.00	551,833,019	4,167	1,505,502	19.007	366.544
1929	31,486,168	30,640,936	1.03	561,029,232	4,790	1,712,206	17.896	327.665
1930	22,474,467	17,320,623	1.30	339,331,261	3,046	1,065,248	16.260	318.547
1931	15,824,809	14,690,625	1.08	292,256,872	2,335	729,407	20.141	400.677
1932	5,946,135	5,963,153	1.00	122,161,249	1,653	341,465	17.463	357.756
1933	5,422,210	5,821,488	0.93	119,608,509	1,271	281,485	20.681	424.920
1934	7,187,744	8,388,549	0.86	172,097,969	1,769	380,711	22.034	452.044
1935	n.a.	12,568,312	n.a.	250,868,968	1,625	454,935	27.627	551.439
1936	22,887,692	23,265,345	0.98	452,477,985	2,197	739,797	31.448	611.625

[a]For source of figures for "Overburden removed" see table A-7, ftn. g. Except where otherwise indicated, figures for other items for 1917-36 are from table A-7; those for 1914-16 were obtained from sources specified in table A-7 and are directly comparable with figures for 1917-36.

[b]Col. (1) divided by col. (2).

n.a. Data not available.

Table A-10.- PRODUCTION AND EMPLOYMENT AT UNDERGROUND COPPER MINES, 1914-36

Year	Production[a]		Employment[b]	
	Ore and tailings, sold or treated (short tons)	Copper recoverable (pounds)	Men	Man-shifts
1914	24,265,429	908,919,530	41,280	11,639,900
1915	29,538,300	1,175,209,253	43,687	12,978,510
1916	40,424,860	1,630,290,791	57,591	17,608,823
1917	39,153,378	1,506,804,067	57,209	17,606,978
1918	42,182,044	1,517,157,181	55,322	17,607,926
1919	26,686,057	979,467,388	36,625	10,899,574
1920	26,803,799	975,433,497	32,675	10,247,888
1921	10,368,278	389,724,260	16,943	4,277,570
1922	19,083,904	765,346,972	24,257	7,141,496
1923	26,225,228	1,036,018,434	29,194	9,112,519
1924	26,882,599	1,115,247,419	28,491	8,828,361
1925	29,765,117	1,175,324,548	28,882	8,820,798
1926	31,327,117	1,143,280,407	28,309	8,900,821
1927	31,937,379	1,088,727,413	26,937	8,280,548
1928	32,733,404	1,139,493,986	26,394	8,395,145
1929	36,922,985	1,279,203,501	32,357	10,271,506
1930	29,563,700	1,003,444,389	24,646	7,184,989
1931	19,806,540	756,955,629	17,352	4,346,455
1932	6,613,407	351,668,232	7,902	1,949,204
1933	2,809,139	257,875,047	5,705	1,408,303
1934	3,626,166	298,097,180	6,315	1,460,087
1935	6,133,276	438,443,783	8,563	2,332,148
1936	14,445,674	676,605,499	11,905	3,614,892

[a]Computed from table A-2; figures for open-cut production were deducted from those for total production.

[b]Obtained by deducting the number of men employed and the number of man-shifts worked for four mines - Utah, Chino, New Cornelia, and Copper Flat-Ruth - from the respective total number of men and total number of man-shifts for the copper-mining industry. Thus the figures include the workers engaged in surface mining at the two other open-cut mines - United Verde and Sacramento Hill - because it was not possible to segregate employees working underground from those working on the surface at these operations. The figures for total number of men employed and man-shifts are from *Metal-Mine Accidents in the United States* (U. S. Depts. Com. and Int., Bur. Mines); those representing the four mines are from reports of individual companies to the Bureau of Mines.

Table A-11.- POUNDS OF COPPER PER TON OF ORE, BY MINING METHOD, 1910-36[a]

Year	All methods	Mining method						
		Open-cut[b]	Underground					
			Total	Block-caving	Open-stope	Square-set	Cut-and-fill	Shrinkage
1910	n.a.	22.56	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
1911	n.a.	22.21	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
1912	n.a.	19.63	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
1913	n.a.	18.20	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
1914	n.a.	20.01	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
1915	n.a.	20.10	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
1916	n.a.	19.45	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
1917	30.56	17.06	39.09	23.07	19.97	77.43	65.41	59.07
1918	29.39	17.62	37.43	22.96	19.53	71.71	65.87	45.25
1919	32.72	20.46	39.00	24.51	23.94	78.90	65.96	44.84
1920	32.26	19.85	39.24	21.38	24.02	75.12	72.62	54.95
1921	35.42	21.30	40.29	24.66	22.60	90.74	53.20	78.04
1922	36.54	19.60	43.99	24.40	30.28	77.52	65.71	75.71
1923	32.42	18.12	42.97	22.88	32.05	72.26	68.76	73.76
1924	32.01	19.09	42.55	22.93	30.09	68.87	75.71	66.94
1925	31.41	18.41	42.19	22.12	28.02	75.99	76.21	59.14
1926	29.84	18.84	39.43	20.14	29.64	75.09	75.01	52.23
1927	29.04	18.67	37.14	20.09	29.29	67.94	74.70	43.70
1928	28.97	18.50	38.07	19.15	30.42	73.87	77.51	38.42
1929	28.74	18.02	37.99	18.25	29.73	73.53	83.30	38.57
1930	29.10	18.49	35.02	17.60	30.63	72.72	78.58	37.42
1931	30.06	19.11	37.85	18.54	35.02	83.96	83.55	44.12
1932	36.65	19.83	52.21	19.90	49.63	126.02	116.66	72.37
1933	42.29	19.78	94.55	23.22	55.55	123.54	124.94	25.32
1934	38.17	19.70	85.75	41.33	48.48	125.23	114.75	16.80
1935	37.96	19.67	77.79	32.56	48.38	113.16	93.98	123.26
1936	30.72	19.12	50.47	20.21	38.09	87.32	82.64	77.28

[a]Except for open-cut method, data are from table A-7.

[b]Data are for the Utah Copper, Chino, Copper Flat (excluding figures for the Ruth mine), and New Cornelia mines and are from reports of individual companies to the Bureau of Mines. Figures for ore mined rather than for ore sold or treated were used in the case of the Copper Flat mine because figures for ore sold or treated for this mine are not available. The differences between "ore treated" and "ore mined" are negligible.

n.a. Data not available.

Table A-12.- PRODUCTION AND OUTPUT PER MAN-HOUR AT MICHIGAN COPPER MINES, 1917-36[a]

Year	Production						Output per man-hour			
	Ore and tailings, sold or treated			Copper recoverable from -			Ore		Copper from -	
	Ore (short tons)	Tailings (short tons)	Ore and tailings (short tons)	Ore (pounds)	Tailings (pounds)	Ore and tailings (pounds)	Ore (short tons)	Ore and tailings (short tons)	Ore (pounds)	Ore and tailings (pounds)
1917	10,557,888	730,543	11,288,431	216,354,054	9,075,457	225,429,511	0.359	0.383	7.349	7.657
1918	9,634,045	715,007	10,349,052	192,728,637	9,245,388	201,974,025	.315	.338	6.301	6.603
1919	6,806,645	915,659	7,722,304	164,389,036	9,082,952	173,471,988	.311	.353	7.509	7.924
1920	5,461,486	1,378,500	6,839,986	132,392,620	14,138,240	146,530,860	.345	.432	8.357	9.250
1921	2,930,605	367,000	3,297,605	76,705,748	5,301,736	82,007,484	.311	.350	8.152	8.715
1922	3,301,029	1,378,250	4,679,279	106,608,361	11,362,500	117,970,861	.234	.332	7.565	8.372
1923	3,408,970	1,743,100	5,152,070	118,095,317	16,901,200	134,996,517	.260	.393	9.020	10.310
1924	3,561,921	1,687,000	5,248,921	115,566,797	18,627,000	134,193,797	.276	.407	8.957	10.401
1925	4,261,483	2,731,605	6,993,088	127,626,404	26,957,000	154,583,404	.310	.509	9.283	11.243
1926	4,885,505	2,756,177	7,641,682	144,393,489	30,988,076	175,381,565	.297	.464	8.766	10.647
1927	5,027,334	3,019,000	8,046,334	148,820,775	28,717,000	177,537,775	.317	.508	9.391	11.203
1928	4,732,658	2,629,000	7,361,658	148,546,704	29,896,000	178,442,704	.314	.488	9.841	11.822
1929	5,154,180	2,444,000	7,598,180	152,891,218	33,511,000	186,402,218	.295	.436	8.764	10.685
1930	5,114,036	1,545,000	6,659,036	151,539,413	17,842,000	169,381,413	.320	.417	9.491	10.607
1931	3,570,748	(b)	(b)	118,059,491	(b)	(b)	.354	(b)	11.691	(b)
1932	1,142,775	(b)	(b)	54,396,108	(b)	(b)	.194	(b)	9.256	(b)
1933	697,158	(b)	(b)	46,853,130	(b)	(b)	.249	(b)	16.705	(b)
1934	700,055	(b)	(b)	48,215,859	(b)	(b)	.228	(b)	15.707	(b)
1935	753,803	623,000	1,376,803	54,990,689	9,118,000	64,108,689	.230	.420	16.756	19.534
1936	1,678,600	1,547,000	3,225,600	76,801,019	19,167,000	95,968,019	.369	.710	16.894	21.110

[a]Data are from table A-8. Figures for "tailings" were obtained by subtracting figures for "ore" from those for "ore and tailings."

[b]Production of tailings not reported.

Table A-13.- PRODUCTION OF COPPER ORE AND COPPER AT COPPER MINES, BY MINING METHOD, 1917-36[a]

Year	Open-cut[b]		Block-caving		Cut-and-fill		Square-set		Shrinkage		Open-stope		All methods, copper recoverable	
	Ore and tailings, sold or treated (short tons)	Copper recoverable (pounds)	Ore and tailings, sold or treated (short tons)	Copper recoverable (pounds)	Ore and tailings, sold or treated (short tons)	Copper recoverable (pounds)	Ore and tailings, sold or treated (short tons)	Copper recoverable (pounds)	Ore and tailings, sold or treated (short tons)	Copper recoverable (pounds)	Ore and tailings, sold or treated (short tons)	Copper recoverable (pounds)	Pounds[c]	Percent of United States total[d]
1917	20,059,859	359,275,077	13,298,640	327,500,952	3,337,818	215,667,300	6,009,926	474,703,601	2,360,156	142,314,732	9,852,884	198,515,584	1,717,977,246	92.1
1918	20,822,032	366,897,693	15,657,751	375,484,887	3,112,376	203,047,088	7,521,411	501,613,466	2,563,620	120,378,789	9,064,627	179,386,429	1,746,808,352	92.7
1919	10,351,224	211,824,818	10,006,075	257,050,476	2,259,265	147,652,639	3,745,729	285,759,396	1,758,210	80,353,252	6,763,164	150,762,203	1,133,402,784	95.1
1920	11,340,071	226,253,315	10,693,009	242,939,997	2,178,516	155,294,196	5,033,409	308,453,509	1,733,262	94,933,850	6,299,212	133,244,342	1,161,119,209	96.6
1921	3,028,104	65,983,473	3,758,421	94,775,165	1,247,308	66,628,518	1,248,508	83,776,997	918,926	71,709,401	2,877,095	62,024,975	444,898,529	97.6
1922	7,809,343	181,952,133	7,648,916	196,129,893	1,362,464	79,533,798	4,165,549	261,534,818	1,335,320	101,101,568	4,191,320	96,611,742	916,863,952	96.8
1923	19,319,330	413,761,783	11,682,562	285,376,480	1,762,197	113,602,828	5,571,405	350,561,833	1,634,349	120,802,227	5,026,941	122,191,936	1,406,297,087	97.0
1924	22,389,783	469,772,877	12,813,025	302,467,317	2,323,428	171,545,728	4,450,646	358,681,352	1,870,766	130,129,219	5,033,100	119,335,628	1,551,932,121	97.9
1925	23,430,259	474,966,934	13,632,635	305,193,829	2,530,507	187,019,199	4,306,646	381,052,996	2,070,908	126,834,646	6,809,340	141,266,208	1,616,333,812	97.9
1926	25,953,658	546,762,300	15,197,416	309,415,626	2,427,406	172,242,063	4,485,549	365,235,330	2,097,369	109,562,320	7,214,968	163,213,594	1,666,431,233	98.6
1927	24,856,799	527,200,263	15,311,769	319,622,903	2,074,771	154,281,952	4,329,782	323,054,866	2,338,065	102,411,263	7,697,267	165,804,713	1,592,375,960	98.5
1928	29,364,499	634,625,700	16,190,664	332,903,698	2,192,445	171,604,600	4,535,673	346,739,300	2,385,049	92,032,485	7,240,038	170,404,810	1,748,310,593	98.5
1929	31,498,868	682,356,603	18,652,457	352,986,218	2,058,332	172,256,785	5,667,272	438,880,571	2,369,342	91,813,613	7,574,349	186,467,295	1,924,761,085	98.1
1930	17,817,809	381,724,370	14,798,423	282,558,733	1,584,950	125,088,065	3,926,121	310,238,514	1,941,028	72,744,637	6,720,744	176,484,037	1,348,838,356	97.4
1931	14,441,663	285,557,534	10,964,606	210,984,575	991,780	82,632,677	3,299,970	284,888,072	955,833	42,174,381	3,466,441	121,401,584	1,027,638,823	98.6
1932	5,706,787	113,188,720	3,678,694	77,087,497	699,805	81,639,411	894,893	121,063,361	157,556	11,401,930	1,039,829	51,608,627	455,989,546	98.1
1933	5,578,473	110,349,669	375,149	12,326,457	718,171	89,725,282	719,953	98,100,505	2,117	53,600	849,464	47,190,320	357,745,833	97.2
1934	8,097,472	159,549,245	647,203	27,273,069	1,017,222	116,730,556	667,898	88,805,768	3,018	50,714	1,069,391	51,842,800	444,252,152	97.1
1935	12,978,778	299,333,801	1,451,634	47,217,851	1,087,188	109,705,238	1,466,628	181,806,826	137,477	16,944,797	1,762,970	64,264,380	719,272,893	97.5
1936	24,068,571	526,596,388	5,919,136	122,634,628	1,348,855	124,866,669	2,644,861	259,515,709	600,734	46,423,444	3,586,716	96,861,800	1,176,898,638	97.8

[a]Compiled from reports of individual companies to the Bureau of Mines. The mines covered in this table are those of the principal domestic operators who produced copper as their principal metal and whose output exceeded 2,000,000 pounds in either 1918 or 1929. In addition, the table covers the Mountain City Copper Co. for 1933, 1935, and 1936. It should be noted that figures in this table do not correspond with those in table A-7. The differences are accounted for by the fact that more mines are covered in this table and that the production of those companies which conducted both open-cut and underground mining was separated into open-cut production and production by the underground-mining method used. Copper production represents copper metal recoverable from ores, tailings, and mine-water precipitates. The mines covered in this table accounted for 92 to 99 percent of the total mine production of recoverable copper (see last column).

[b]Represents the production of six open-cut mines. Includes the production of the Utah Copper Co., 1917-36; Nevada Consolidated Copper Co., Chino mine, 1917-34 (idle in 1935-36); Nevada Consolidated Copper Co., Copper Flat pits, 1917-36; New Cornelia mine, 1917-32 and 1934-36 (idle in 1933); United Verde mine (pit), 1920, 1922-31, and 1935-36 (no open-cut production in 1921 and 1932-34); and Sacramento Hill mine, 1917 and 1919-29 (no open-cut production in 1918, open-cut operations discontinued at the beginning of 1930). Figures for ore mined rather than for ore sold or treated were used for United Verde, Copper Flat pits, and Sacramento Hill. The differences between "ore mined" and "ore treated" are negligible.

[c]Total for mines covered in this table.

[d]Preceding column as percent of table A-1, second column.

Table A-14.- NUMBER OF WAGE EARNERS AND HORSEPOWER AT COPPER MINES, 1880-1929[a]

Year	Average number of wage earners	Horsepower		Percent electrical is of total horsepower	Horsepower per wage earner	
		Total	Electrical		Total	Electrical
1880	6,039	13,511	n.a.	n.a.	2.24	n.a.
1889	9,750[b]	34,390	n.a.	n.a.	3.53	n.a.
1902	26,007	200,819	4,648	2.3	7.72	0.18
1909	51,643[c]	376,464	78,174	20.8	(d)	(d)
1919	43,717	522,426	296,922	56.8	11.95	6.79
1929	44,502	701,791	605,133	86.2	15.77	13.60

[a]Data include figures for milling of ores. They are from *Eleventh Census of the United States: 1890*, "Report on Mineral Industries" (U. S. Dept. Int., Census Office, 1892), p. 156; *Special Reports of the Census Office*, "Mines and Quarries: 1902" (U. S. Dept. Com. and Labor, Bur. Census, 1905), pp. 469, 476-7; *Thirteenth Census of the United States: 1910*, Vol. XI, "Mines and Quarries: 1909" (U. S. Dept. Com., Bur. Census, 1913), p. 37; *Fourteenth Census of the United States: 1920*, Vol. XI, "Mines and Quarries: 1919" (1922), p. 37; and *Fifteenth Census of the United States: 1930*, "Mines and Quarries: 1929" (1933), pp. 294, 300.

[b]Includes foremen.

[c]The figure includes wage earners at smelters operated in conjunction with mines and is therefore not comparable with figures for other years.

[d]Not computed. See ftn. c.

n.a. Data not available.

Table A-15.- TRENDS IN COPPER CONTENT OF CONCENTRATOR HEADS AND COPPER RECOVERIES AT TYPICAL CONCENTRATORS, 1911-35[a]

(Percentages)

Year	Utah Copper Co.[b]		Nevada Consolidated Copper Co., Hayden Mill		Calumet & Hecla Consolidated Copper Co., Conglomerate Mill	
	Copper in heads	Copper recovery	Copper in heads	Copper recovery	Copper in heads	Copper recovery
1911	1.51	69.5	1.83	63.0	1.96	78.2
1912	1.36	66.3	1.68	68.3	1.91	78.3
1913	1.25	64.0	1.72	66.1	1.81	77.2
1914	1.42	66.0	1.76	67.9	1.78	75.9
1915	1.43	64.1	1.67	64.1	1.95	76.7
1916	1.43	62.3	1.61	70.2	1.97	76.6
1917	1.34	61.1	1.64	74.5	1.91	75.9
1918	1.23	65.1	1.61	74.9	1.74	80.8
1919	1.26	78.5	1.81	79.5	1.73	85.8
1920	1.16	81.4	1.72	82.0	1.93	91.0
1921	1.16	83.9	1.65	77.8	2.07	91.6
1922	1.26	80.0	1.59	76.7	2.53	93.3
1923	1.12	81.0	1.57	74.9	2.42	93.5
1924	1.07	85.9	1.52	78.3	2.36	93.0
1925	1.02	87.0	1.50	84.5	2.25	93.8
1926	1.01	87.0	1.4[c]	88.2[c]	2.27	94.5
1927	0.98	89.1	1.3[c]	87.4[c]	2.19	94.8
1928	0.99	85.6	1.25	87.1	2.26	95.1
1929	0.99	85.7	1.1[c]	84.1[c]	2.22	94.6
1930	0.97	89.0	1.0[c]	80.9[c]	2.55	95.7
1931	0.96	91.2	1.2[c]	81.8[c]	n.a.	n.a.
1932	0.97	93.2	n.a.	n.a.	n.a.	n.a.
1933	1.03	92.8	n.a.	n.a.	n.a.	n.a.
1934	1.02	92.9	n.a.	n.a.	n.a.	n.a.
1935	1.00	89.2	n.a.	n.a.	n.a.	n.a.

[a]Except as noted, data for 1911-30 are from T. G. Chapman, *Concentration of Copper Ores in North America* (U. S. Dept. Int., Bur. Mines, Bull. No. 392, 1936), pp. 10-1. Figures for "Copper in heads" represent the percentages that copper comprised of the concentrator heads; those for "Copper recovery" represent the percentages of the copper content of the ores processed which were actually recovered.

[b]Data for Utah Copper Co. for 1931-35 from annual reports of the Kennecott Copper Corp.

[c]A. B. Parsons, *The Porphyry Coppers* (1st ed.; New York: The American Institute of Mining and Metallurgical Engineers, 1933), p. 200.

n.a. Data not available.

Table A-16.- ACCIDENT RATE AT COPPER MINES, 1911-36

Year	Number of lost-time accidents[a]			Man-hours of employment[b]	Number of accidents per million man-hours of exposure[c]		
	Total	Killed	Injured		Total	Killed	Injured
1911	10,580	238	10,342	110,194,600	96.01	2.16	93.85
1912	13,876	239	13,637	126,650,000	109.56	1.89	107.67
1913	13,599	236	13,363	138,954,128	97.87	1.70	96.17
1914	13,532	165	13,367	102,760,464	131.69	1.61	130.08
1915	15,430	176	15,254	113,691,200	135.72	1.55	134.17
1916	20,398	230	20,168	151,457,136	134.68	1.52	133.16
1917	20,309	374	19,935	152,684,504	133.01	2.45	130.56
1918	20,733	220	20,513	152,833,424	135.66	1.44	134.22
1919	12,376	140	12,236	94,851,928	130.48	1.48	129.00
1920	12,175	128	12,047	89,456,952	136.10	1.43	134.67
1921	4,777	55	4,722	35,690,368	133.85	1.54	132.30
1922	8,100	75	8,025	60,039,864	134.91	1.25	133.66
1923	12,100	107	11,993	82,451,728	146.75	1.30	145.45
1924	11,979	121	11,858	81,821,160	146.40	1.48	144.93
1925	12,281	102	12,179	83,366,424	147.31	1.22	146.09
1926	10,223	121	10,102	84,095,792	121.56	1.44	120.12
1927	8,490	111	8,379	77,002,536	110.26	1.44	108.81
1928	7,393	100	7,293	79,205,176	93.34	1.26	92.08
1929	9,062	121	8,941	95,869,696	94.52	1.26	93.26
1930	5,397	76	5,321	66,001,896	81.77	1.15	80.62
1931	2,631	51	2,580	41,019,314	64.14	1.24	62.90
1932	882	23	859	18,608,421	47.40	1.24	46.16
1933	748	14	734	13,471,547	55.52	1.04	54.49
1934	681	12	669	14,726,617	46.24	0.81	45.43
1935	1,485	19	1,466	22,293,255	66.61	0.85	65.76
1936	2,857	38	2,819	34,900,287	81.86	1.09	80.77

[a] Data are from issues of *Metal-Mine Accidents in the United States* (U. S. Depts. Com. and Int., Bur. Mines).

[b] Table A-1.

[c] Number of persons killed or injured divided by number of man-hours of employment.

Table A-17.- ESTIMATED USE OF COPPER IN THE UNITED STATES, BY PRINCIPAL USE, 1919-37[a]

(Short tons)

Year	Total	Electrical manufactures[b]	Telephones and telegraphs	Light and power lines[c]	Automobiles[d]	Buildings[e]	Radios	Refrigerators[f]	Other wire[g]	Castings[h]	Other uses[i]	Manufactures for export
1919	621,373	142,000	40,500	27,250	48,950	30,600	n.a.	n.a.	39,300	63,150	129,923	99,700
1920	684,035	170,000	61,000	29,500	60,575	34,500	n.a.	n.a.	48,900	65,000	107,760	106,800
1921	459,885	130,000	54,000	33,000	47,700	22,150	900	n.a.	30,200	24,000	68,935	49,000
1922	539,113	134,500	60,000	48,750	74,600	36,950	1,800	n.a.	43,100	39,750	51,663	48,000
1923	737,700	178,500	75,000	85,850	105,100	38,050	1,800	n.a.	59,750	64,200	77,350	52,100
1924	765,950	195,500	80,000	90,000	94,950	40,450	2,250	n.a.	69,100	58,850	80,750	54,100
1925	836,000	183,500	86,000	110,000	107,900	46,700	4,000	7,900	75,350	65,100	93,250	56,300
1926	909,900	201,000	104,000	122,000	104,500	50,200	5,000	15,000	88,400	69,000	99,900	50,900
1927	862,450	196,500	93,000	103,000	100,800	52,800	4,500	15,800	81,450	67,200	90,000	57,400
1928	990,700	213,000	119,000	115,000	127,400	62,000	8,400	13,800	94,200	72,500	98,800	66,600
1929	1,159,800	261,000	164,000	127,000	138,200	59,000	15,500	17,200	115,200	79,500	108,300	74,900
1930	956,150	221,000	122,000	130,000	87,000	50,000	13,000	15,000	102,900	55,400	88,750	71,100
1931	650,000	162,000	70,000	85,000	61,600	45,000	10,000	13,000	59,900	36,000	59,300	48,200
1932	368,000	90,000	27,000	49,000	32,300	29,000	7,000	9,000	34,200	27,000	40,900	22,600
1933	415,000	90,000	18,000	33,000	49,000	36,000	11,500	11,400	51,000	36,000	63,500	15,600
1934	463,000	101,000	18,000	36,000	63,000	36,000	12,500	15,700	44,600	36,000	74,700	25,500
1935	574,700	128,000	18,000	55,500	95,000	49,000	16,000	15,400	53,600	36,000	78,700	29,500
1936	749,000	164,000	26,000	72,000	108,000	71,000	24,000	15,000	96,500	39,000	101,900	31,600
1937	860,000	212,000	40,000	83,000	112,000	70,500	23,100	13,500	108,800	40,000	112,100	45,000

[a] Data for 1919-25 are from *Year Book of the American Bureau of Metal Statistics: 1926* (New York: American Bureau of Metal Statistics, 1927), p. 34; for 1926-27 from same, *1932* (1933), p. 34; for 1928 from same, *1937* (1938), p. 36; and for 1929-37 from same, *1938* (1939), p. 36.

[b] Generators, motors, electric locomotives, switchboards, light bulbs, etc.

[c] Transmission and distribution wire and bus bars (public-utility companies only).

[d] Does not include starter, generator, and ignition equipment.

[e] Includes electrical work.

[f] Exclusive of electrical equipment.

[g] Includes wire cloth, trolley wire, and other rod and wire.

[h] Bearings, bushings, lubricators, valves, and fittings.

[i] Ammunition, clocks and watches, coinage, copper-bearing steel, fire-fighting apparatus, heating radiators, railway equipment, shipbuilding, washing machines, household water heaters, air conditioning, and others.

n.a. Data not available.

Table A-18.- ESTIMATED COPPER RESERVES, BY STATE, DISTRICT, AND MINE[a]

State, district, and mine	Ore (tons)	Grade (percent)	Type of deposit	Copper (tons)	Percent of total reserves
United States	–	–	–	35,924,707	100.00
Alaska	–	–	–	110,000	0.31
Copper River					
Kennecott[b]	n.a.	n.a.	Vein and replacement	110,000	0.31
Arizona	–	–	–	16,289,100	45.34
Ajo					
New Cornelia[c]	200,000,000	1.10	Porphyry	2,200,000	6.12
Eureka					
Bagdad[b]	n.a.	1.20	Porphyry	570,000	1.59
Globe	–	–	–	4,845,000	13.49
Inspiration[c]	150,000,000	1.35	Porphyry	2,025,000	5.64
Miami[c]	130,000,000	1.10	Porphyry	1,430,000	3.98
Old Dominion[d]	2,000,000	2.00	Porphyry	40,000	0.11
Pinto Valley[b]	n.a.	n.a.	Porphyry	1,300,000	3.62
Van Dyke[d]	1,000,000	5.00	Vein and replacement	50,000	0.14
Greenlee					
Morenci[c e]	450,000,000	1.10	Porphyry	4,950,000	13.78
Mineral Creek					
Ray[c]	135,000,000	1.60	Porphyry	2,160,000	6.01
Pioneer					
Magma[b]	n.a.	n.a.	Vein and replacement	220,000	0.61
Verde	–	–	–	564,100	1.57
United Verde[b]	n.a.	4.65	Vein and replacement	550,000	1.53
United Verde Extension[b]	n.a.	6.17	Vein and replacement	14,100	0.04
Warren	–	–	–	780,000	2.17
Copper Queen[b]	n.a.	5.30	Vein and replacement	670,000	1.86
Shattuck-Denn[b]	n.a.	n.a.	Vein and replacement	110,000	0.31
California	–	–	–	218,137	0.61
Plumas	–	–	–	137,137	0.38
Engels[f]	556,837	2.00	Vein and replacement	11,137[g]	0.03
Walker[b]	n.a.	1.81	Vein and replacement	126,000	0.35
Shasta					
Balaklala[h]	3,000,000	2.70	Vein and replacement	81,000[g]	0.23
Michigan	–	–	–	373,400	1.04
Lake Superior	–	–	–	373,400	1.04
Calumet and Hecla	n.a.	1.20[f]	Vein and replacement	185,000[b]	0.51
Copper Range[b]	n.a.	n.a.	Vein and replacement	110,000	0.31
Quincy	6,000,000[l]	n.a.	Vein and replacement	40,000[b]	0.11
Seneca[d]	3,486,895	1.10[j]	Vein and replacement	38,400[g]	0.11
Montana	–	–	–	1,500,000	4.17
Butte					
Butte	n.a.	4.00[d]	Vein and replacement	1,500,000[b]	4.17
Nevada	–	–	–	2,665,000	7.42
Ely	–	–	–	2,335,000	6.50
Consolidated Copper[k]	35,000,000	1.10	Porphyry	385,000	1.07
Nevada Consolidated[c]	130,000,000	1.50	Porphyry	1,950,000	5.43
Cope					
Mountain City[b]	n.a.	n.a.	Vein and replacement	330,000	0.92
New Mexico	–	–	–	2,679,000	7.46
Central	–	–	–	2,679,000	7.46
Burro Mountain[l]	11,725,000	1.95	Porphyry	229,000	0.64
Chino[c]	175,000,000	1.40	Porphyry	2,450,000	6.82
Tennessee	–	–	–	158,370	0.44
Ducktown	–	–	–	158,370	0.44
Ducktown	3,029,173[m]	n.a.	Vein and replacement	60,000[b]	0.17
Tennessee Copper[d]	6,148,132	1.60	Vein and replacement	98,370	0.27
Utah	–	–	–	11,131,700	30.99
Bingham	–	–	–	11,131,700	30.99
Ohio Copper[f]	41,000,000	0.32	Porphyry	131,700[g]	0.37
Utah Copper[c]	1,000,000,000	1.10	Porphyry	11,000,000	30.62
Custom ores and byproducts[b]	–	–	–	800,000	2.22

[a] Figures represent reserves of copper that, under conditions of present technology, can be mined to be sold within a price range of 8 to 17 cents per pound.

[b] K. Leith and D. M. Liddell, *The Mineral Reserves of the United States and Its Capacity for Production* (National Resources Committee, mimeo., Mar. 1936), p. 56.

[c] A. B. Parsons, *The Porphyry Coppers* (1st ed.; New York: The American Institute of Mining and Metallurgical Engineers, 1933), p. 9.

[d] J. J. Croston, "Recent Trends in Copper Production, Ore Reserves, and Costs," *Mining Technology*, Vol. I, No. 4 (July 1937), A.I.M.E. Tech. Pub. No. 826.

[e] Includes Clay ore body, which according to Leith and Liddell, *op. cit.*, p. 57, is of 1.02 percent grade and contains 570,000 tons of copper.

[f] G. A. Roush, ed., *The Mineral Industry: Its Statistics, Technology and Trade During 1929* (New York: McGraw-Hill Book Co., Inc., 1930), XXXVIII, 123.

[g] Estimated on basis of grade and tonnage of ore.

[h] L. H. Rand and E. B. Sturgis, *The Mines Handbook* (Suffern, N. Y.: Mines Information Bureau, Inc., 1931), vol. XVIII, pt. I, p. 496.

[i] *Mines Register* (New York: Mines Publications, Inc., 1937), XIX, 783.

[j] Twenty-one to 23 pounds per ton (Rand and Sturgis, *op. cit.*, p. 1204).

[k] A. Notman, "Estimated World Reserves of Copper," *Copper Resources of the World* (Washington, D. C.: XVI International Geological Congress, 1935), I, 32.

[l] G. A. Roush, ed., *The Mineral Industry . . . . 1930* (1931), XXXIX, 126.

[m] *Poor's Industrial Volume, 1937* (New York: Poor's Publishing Co., 1937), p. 1815.

n.a. Data not available.