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COPPER MINING IN NORTH AMERICA

BY

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and B. S. BUTLER

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COPPER MINING IN NORTH AMERICA[1]

By E. D. Gardner,[2] C. H. Johnson,[3] and B. S. Butler[4]

INTRODUCTION

From the standpoint of tonnage and dollar value copper is the most important nonferrous metal mined in North America. The opening of copper mines has had an important bearing on the development and settlement of many sections of the continent. The prosperity of many of the mining States or provinces depends directly upon the copper-mining industry. Normally, about 35,000 men are employed in the copper mines, mills, and smelters of the United States.

The United States is the most important producer of copper among the nations; more copper is mined in North America than in any other continent, and for many years the United States alone produced about half of the copper of the world. During the last 10 years increasing production in South America and Africa has reduced the relative importance of North America and the United States.

In this paper are assembled and summarized many subjects relating to the copper industry in North America. Production of mines and districts, history of the industry, geology of the principal deposits, and mining methods and costs are discussed. The principal copper mines of the continent are listed and pertinent data regarding their operations tabulated. Methods are illustrated by descriptions of practices at typical mines. Milling methods and costs have been described in a companion paper by Chapman.[5]

This paper was first prepared in 1932, but as no funds for its publication were available at that time it was laid aside until 1936, when it was revised. A summary of the chapter on mining methods and costs was published in Copper Resources of the World (vol. 2, pp. 775–813) by the Sixteenth International Geological Congress in 1935.

ACKNOWLEDGMENTS

Publications of the Bureau of Mines and the Geological Survey, State publications, official Canadian reports, company reports, and transactions of technical societies and the technical press have been used freely. Original authors are credited by footnote acknowledgments.

The section on mining methods and costs is largely a compilation of information previously published in Bureau of Mines information circulars, supplemented by articles published elsewhere and original data collected by the authors.

S. E. Casterton and Paul T. Allsman, assistant mining engineers, Mining Division, Bureau of Mines. assisted in compiling production figures.

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[5] Chapman, Thomas G., Concentration of Copper Ores in North America: Bull. 392, Bureau of Mines, 1936, 169 pp.

PART 1. PRODUCTION OF COPPER AND HISTORY OF COPPER MINING

PRODUCTION

The United States has been the principal producer of copper in North America; however, Canada and Mexico also hold important places in the copper industry. The first production in North America in modern times was in Cuba, where copper mining was begun in 1542. Table 1 shows the output of copper in the world and by countries in North America.

It will be noted that world production has been about 51,000,000 short tons, of which 45 percent has come from the United States. Nearly 60 percent of the total has been produced in the last 22 years and 30 percent in the last decade.

About 95 percent of the production in the United States has come from 16 districts in 9 States. Virtually all the Canadian production has come from four fields and the Mexican from three districts.

TABLE 1.—*Copper production of North America, by countries and world, 1801–1934, in short tons* [1]

Period	World	North America		United States		Mexico		Canada		Newfoundland		Cuba	
		Quantity	Percent	Quantity	Percent	Quantity	Percent	Quantity	Percent	Quantity	Percent	Quantity	Percent
1801–50	1, 502, 256	77, 482	5. 2	2, 688	0. 2	----------	----	----------	----	----------	----	74, 794	5. 0
1851–1900	9, 970, 111	3, 718, 376	37. 3	3, 358, 784	33. 6	159, 918	1. 6	69, 805	0. 7	52, 997	0. 5	81, 872	. 8
1901–25	25, 726, 406	16, 920, 711	65. 8	14, 542, 269	56. 5	1, 298, 438	5. 0	894, 702	3. 5	28, 208	. 1	157, 094	. 6
1926	1, 668, 897	1, 001, 235	60. 0	862, 625	51. 7	62, 111	3. 7	66, 547	4. 0	----------	----	9, 952	. 6
1927	1, 678, 818	975, 846	58. 1	824, 967	49. 1	66, 000	3. 9	70, 072	4. 2	----------	----	14, 807	. 9
1928	1, 892, 649	1, 087, 537	57. 5	904, 885	47. 8	74, 216	3. 9	101, 347	5. 4	----------	----	7, 089	. 4
1929	2, 137, 360	1, 222, 081	57. 2	997, 540	46. 7	95, 408	4. 5	124, 058	5. 8	----------	----	5, 075	. 2
1930	1, 775, 825	956, 074	53. 9	705, 063	39. 9	80, 922	4. 6	151, 737	8. 6	1, 054	. 1	17, 298	1. 0
1931	1, 546, 527	751, 271	48. 6	528, 867	34. 1	59, 758	3. 9	146, 149	9. 4	1, 608	. 1	14, 889	1. 0
1932	1, 001, 991	409, 568	40. 9	238, 108	23. 7	38, 716	3. 9	123, 838	12. 3	2, 373	. 2	6, 533	. 6
1933	1, 153, 006	397, 117	34. 4	190, 642	16. 5	43, 128	3. 7	149, 989	13. 0	3, 485	. 3	9, 873	. 8
1934	1, 413, 161	479, 269	33. 9	236, 950	16. 8	48, 797	3. 5	182, 443	12. 8	4, 254	. 3	6, 825	. 5
Total	51, 467, 007	27, 996, 567	54. 4	23, 388, 388	45. 5	2, 027, 412	3. 9	2, 080, 687	4. 0	93, 979	. 2	406, 101	. 8

[1] 1801–1925, from Julihn, C. E., Summarized Data of Copper Production: Ec. Paper 1, Bureau of Mines, 1928, table 26; 1926–29, from Mineral Resources of United States; not strictly comparable, as mine production figures were used throughout, whereas Julihn used smelter production for United States and mine production for other countries. Totals for 1930–34 from Minerals Yearbook, 1935 ed.

Table 2 shows the total production of copper in the principal copper-producing States in the United States, and table 3, the relative rank of the States as copper producers, 1930–34. Since 1930 few of the copper mines have been producing at capacity. Many were closed during part of the period, and operations were still suspended at some properties in the summer of 1936. The growth of the principal copper-producing States and their contributions to the total smelter production of the United States from 1845 through 1934 are shown graphically in figure 1.

The total mine production of copper in the principal copper districts of the United States is shown in table 4. Figure 2 illustrates the start and growth of production in these districts from 1880 through 1934.

The production of copper in Mexico, by districts, is given in table 5 and shown graphically in figure 3; Canadian production by Provinces is shown graphically in figure 4. Table 6 gives production from 1886 through 1934 and table 1 the total production for Cuba and Newfoundland.

Part 3 gives production of copper at representative mines.

TABLE 2.—*Smelter output of copper in the United States, by States, 1845–1934, short tons*[1]

State	Total tons	Total pounds	Percent of total	State	Total tons	Total pounds	Percent of total
Arizona	7, 689, 210	15, 378, 420, 000	32. 74	Colorado	196, 116	392, 232, 000	0. 83
Montana	5, 318, 165	10, 636, 330, 000	22. 64	Idaho	74, 202	148, 404, 000	. 32
Michigan	4, 337, 713	8, 675, 426, 000	18. 47	Wyoming	15, 862	31, 724, 000	. 07
Utah	2, 477, 190	4, 954, 380, 000	10. 55	Washington	14, 269	28, 538, 000	. 06
Nevada	998, 620	1, 997, 240, 000	4. 25	Oregon	10, 354	20, 708, 000	. 04
New Mexico	769, 419	1, 538, 838, 000	3. 28	Undistributed	162, 758	325, 516, 000	. 69
Alaska	616, 106	1, 232, 212, 000	2. 62				
California	547, 534	1, 095, 068, 000	2. 33	Total	23, 487, 026	46, 974, 052, 000	100. 00
Tennessee [2]	259, 508	519, 016, 000	1. 11				

[1] Compiled from Minerals Yearbook, 1935.
[2] Approximate production through 1928; figures for 1929–34 are confidential and are included under "Undistributed."

TABLE 3.—*Copper production in the United States, according to mine returns, by States, 1930–34, short tons*[1]

	1930	1931	1932	1933	1934
Alaska	16, 325	11, 307	4, 350	15	50
Arizona	288, 096	200, 672	91, 944	57, 021	88, 533
California	13, 643	6, 466	552	495	285
Colorado	5, 257	4, 083	3, 616	4, 834	5, 647
Idaho	1, 556	572	548	781	765
Michigan	84, 690	59, 030	27, 198	23, 427	24, 108
Montana	98, 094	92, 278	42, 359	32, 738	31, 625
Nevada	54, 602	36, 317	15, 667	14, 245	20, 875
New Mexico	32, 575	30, 752	15, 352	13, 473	11, 815
Oregon	88	1	15	6	19
Utah	90, 263	75, 618	32, 953	36, 791	43, 012
Washington	603	101	2	3	7
Wyoming	6	5			2
Undistributed	19, 276	11, 673	5, 436	6, 814	10, 207
Total	705, 074	528, 875	239, 992	190, 643	236, 950

[1] Compiled from Minerals Yearbook, 1935.

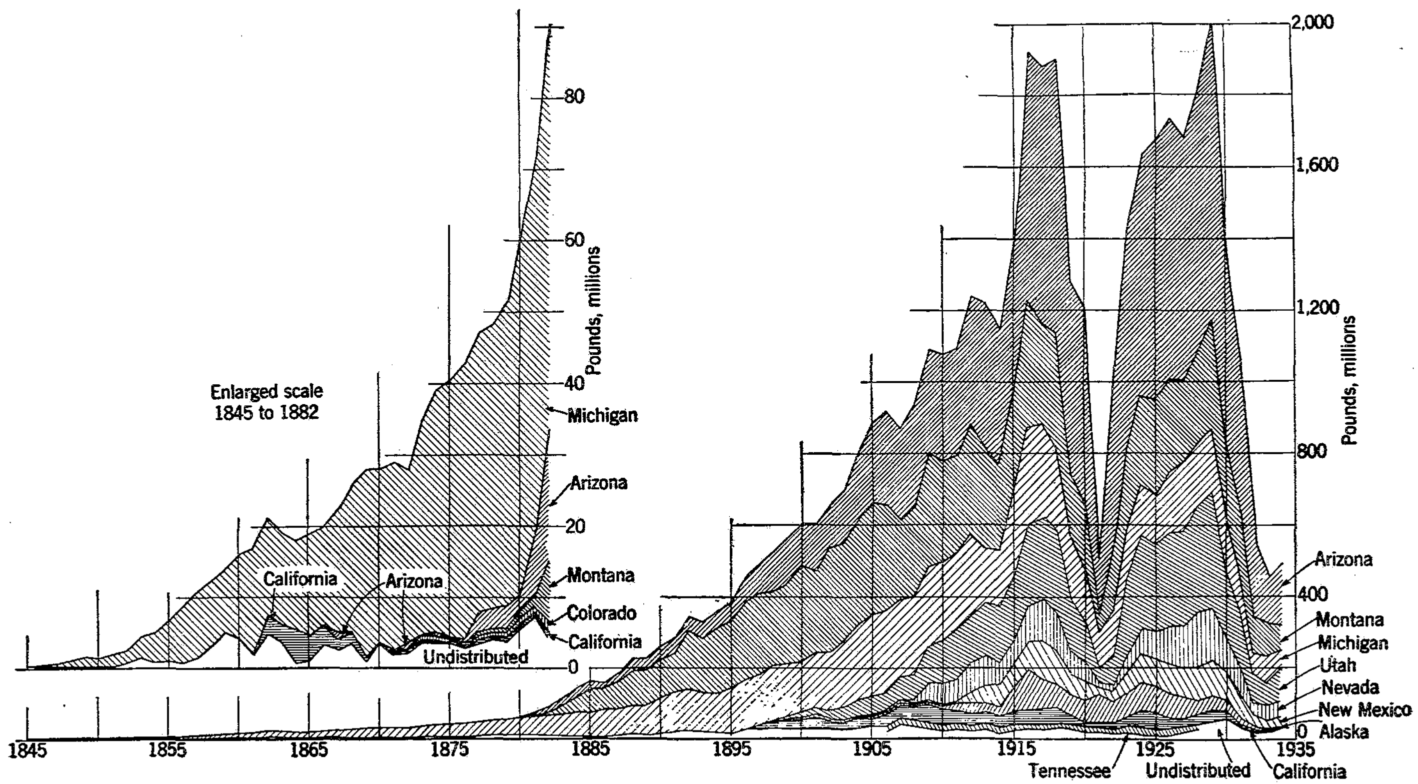

FIGURE 1.—Distribution of United States copper production (smelter) by States, 1845–1934.

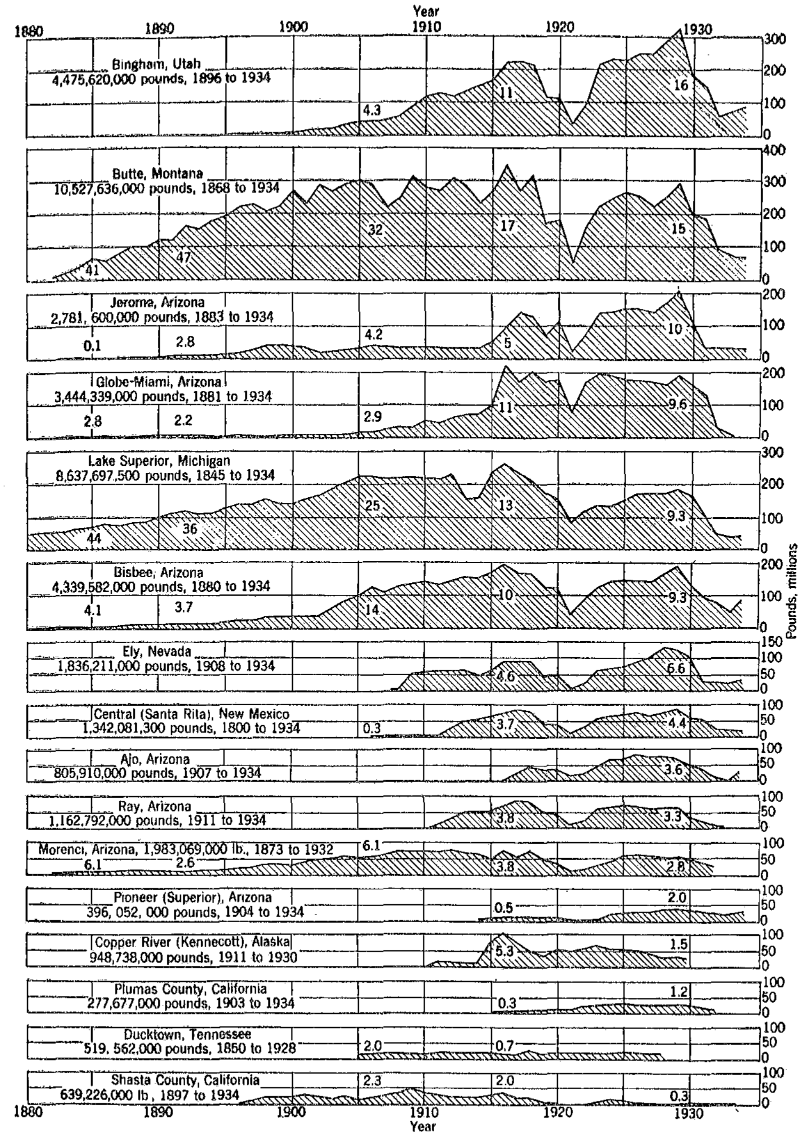

FIGURE 2.—Production of principal United States copper mining districts, 1880–1934. (Figures for 1885, 1892, 1906, 1916, and 1929 indicate percentages of United States total.)

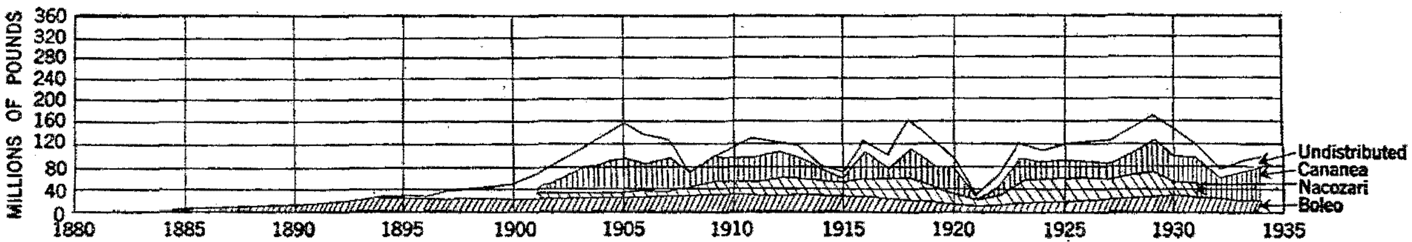

FIGURE 3.—Copper production of Mexico, 1879–1934.

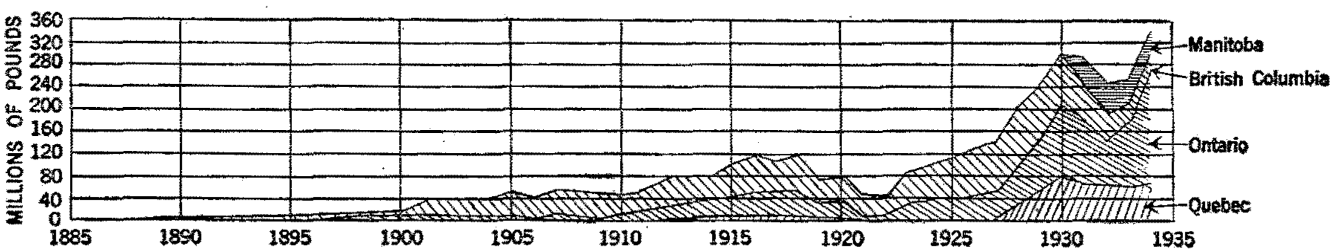

FIGURE 4.—Copper production of Canada, 1886–1934.

TABLE 4.—*Total mine production of copper in the principal copper-producing districts of the United States*

Rank	District	State	Date production began	Total production of copper to end of 1934, pounds
1	Butte	Montana	1868	10, 527, 636, 000
2	Lake Superior	Michigan	1845	8, 637, 697, 500
3	Bingham	Utah	1896	4, 475, 620, 000
4	Bisbee	Arizona	1880	4, 339, 582, 000
5	Globe-Miami	do	1881	3, 444, 339, 000
6	Jerome	do	1883	2, 781, 647, 000
7	Morenci-Metcalf	do	1873	1, 983, 069, 000
8	Ely (Robinson)	Nevada	1908	1, 836, 211, 000
9	Santa Rita	New Mexico	1800	1, 342, 081, 000
10	Ray (Mineral Creek)	Arizona	1911	1, 162, 792, 000
11	Copper River	Alaska	1911	948, 738, 000
12	Ajo	Arizona	1917	805, 910, 000
13	Shasta County	California	1897	639, 226, 000
14	Ducktown	Tennessee	1850	[1] 519, 562, 000
15	Superior	Arizona	1912	392, 897, 000
16	Plumas	California	1903	277, 677, 000

[1] To end of 1928; Bureau of Mines is not at liberty to publish subsequent production.

TABLE 5.—*Copper production from the principal districts of Mexico,*[1] *pounds*

	Cananea	Boleo	Nacozari	Undistributed	Total
1851–1930	1, 106, 318, 000	952, 387, 000	759, 940, 000	855, 381, 000	3, 674, 026, 000
1931	41, 873, 000	25, 410, 000	26, 994, 000	25, 239, 000	119, 516, 000
1932	36, 820, 000	22, 972, 000	0	17, 640, 000	77, 432, 000
1933	51, 793, 000	18, 860, 000	0	15, 604, 000	86, 257, 000
1934	60, 431, 000	18, 076, 000	0	19, 086, 000	97, 593, 000
1931–34	190, 917, 000	85, 318, 000	26, 994, 000	77, 569, 000	380, 798, 000
1851–1934	1, 297, 235, 000	1, 037, 705, 000	786, 934, 000	932, 950, 000	4, 054, 824, 000

[1] Figures to the end of 1930 were taken from Copper Resources of the World, vol. 1, 1935; others were taken from Bureau of Mines Minerals Yearbooks.

TABLE 6.—*Canadian copper production by Provinces,*[1] *pounds*

	Ontario	British Columbia	Quebec	Manitoba	Yukon	Total
1886–1929	908, 861, 000	1, 569, 546, 000	203, 893, 000	9, 866, 000	12, 913, 000	2, 705, 079, 000
1930	127, 719, 000	93, 319, 000	80, 310, 000	2, 088, 000	43, 000	303, 479, 000
1931	112, 883, 000	65, 223, 000	68, 377, 000	45, 821, 000	0	292, 304, 000
1932	77, 055, 000	50, 580, 000	67, 337, 000	52, 706, 000	0	247, 678, 000
1933	145, 505, 000	42, 608, 000	69, 944, 000	41, 921, 000	0	299, 978, 000
1934	205, 060, 000	48, 085, 000	73, 968, 000	37, 773, 000	0	364, 886, 000
1930–34	668, 222, 000	299, 815, 000	359, 936, 000	180, 309, 000	43, 000	1, 508, 325, 000
1886–1934	1, 577, 083, 000	1, 869, 361, 000	563, 829, 000	190, 175, 000	[2] 12, 956, 000	4, 213, 404, 000

[1] Figures for 1930 to 1932 were compiled from Copper Resources of the World, vol. 1, 1935; those for 1933 and 1934 were from Province Bureau of Mines reports; those for 1886 to 1929 in order of reference were taken from Mineral Resources, Annual Report on the Mineral Production of Canada, Province Bureau of Mines reports, and the Canada Yearbook.
[2] From Whitehorse district.

Owing to the overexpansion of the copper industry, which started in 1928, and the world-wide depression, which began in 1929, unwieldly stocks of refined copper were built up in the United States, culminating in a stock of more than a billion pounds in 1933. Stocks had, however, been reduced to 350,000,000 pounds at the end of 1935. Table 7 shows the mine production, imports, exports, and stocks of refined and blister copper in the United States, 1928–35.

Production figures have been taken from various sources, and there are numerous discrepancies, particularly in the larger tables where it was necessary to use several references. Company reports and production figures received from company officials were given priority when available; then the order was Mineral Resources of the United States or issues of Minerals Yearbook, State and Province reports, and finally statistical reports. Mine-production figures were used when available, but often it was necessary to use smelter or refinery output. Metal content from mine production would be higher than that from smelters; that from refineries would be lower. Some references give copper output in kilograms or metric tons; the degree of accuracy of the conversion factors varies. As early figures are only approximate, the totals obtained when they are used with recent production are only approximate.

TABLE 7.—*Production, consumption, and stocks of copper in the United States, 1928–35,*[1] *pounds*

Year	Mine production	Unmanufactured, imports	Metallic copper, exports	Stocks, Jan. 1		
				Refined copper	Blister [2]	Total
1928	1,809,796,907	787,073,640	1,121,186,640	171,000,000	401,000,000	572,000,000
1929	1,995,110,398	974,312,201	992,895,119	114,000,000	423,000,000	537,000,000
1930	1,410,147,374	817,154,236	753,114,927	306,000,000	500,000,000	806,000,000
1931	1,057,750,000	585,892,098	557,574,000	615,000,000	450,000,000	1,065,000,000
1932	476,221,076	391,991,342	295,356,719	924,600,000	348,000,000	1,272,600,000
1933	381,285,194	287,433,540	303,825,790	1,004,000,000	378,000,000	1,382,000,000
1934	[3] 474,810,458	[4] 426,571,568	592,718,891	813,000,000	388,000,000	1,201,000,000
1935	[5] 760,995,826	514,364,526	590,396,106	569,000,000	389,000,000	958,000,000
1936	1,201,422,000	380,677,700	518,064,333	350,000,000	472,000,000	822,000,000

[1] Compiled from Mineral Resources, 1928–31, and Minerals Yearbooks, 1932–35.
[2] Blister and materials in process of refining.
[3] Subject to revision.
[4] Includes material entering under bond.
[5] Preliminary estimate.

ORE RESERVES

Copper-ore reserves in North America are shown in table 8.[6] The data are compiled largely from tables prepared, respectively, by the Minerals Division of the United States Bureau of Foreign and Domestic Commerce and the Planning Committee for Mineral Policy. Barbour gives the ore reserves at Cananea, Mexico, as 40,000,000 tons of 4-percent copper ore, with a copper content of 1,600,000 tons.[7]

The dividends paid by the copper industry, 1922–32, totaled $448,650,000. The gross value of the output for this period was $2,150,314,000.

[6] Keiser, H. D., Ore Reserves and Economic Planning: Eng. and Min. Jour., vol. 135, 1934, p. 397. Leith, Kenneth, and Liddell, Donald M., the Mineral Reserves of the United States and Its Capacity for Production: Natural Resources Committee, Washington, D. C., March 1936, p. 56.
[7] Barbour, P. E., World Copper-ore Reserves: Eng. and Min. Jour., vol. 135, 1934, p. 448.

TABLE 8.—*Copper-ore and copper reserves at principal North American copper mines*

Country and mine	Grade, percent	Maximum production capacity of copper per year, tons	Reserves of ore, tons	Reserves of copper, tons
United States:[1]				
Anaconda group:				
Butte[2]		100, 000		1, 500, 000
Inspiration[2]	1. 37	60, 000	69, 000, 000	945, 000
Mountain City (Nevada)[2]		20, 000		330, 000
Walker[2]	1. 81	8, 000	6, 961, 000	126, 000
Bagdad[2 3]	1. 20		47, 500, 000	570, 000
Calumet & Hecla[2]		45, 000		185, 000
Consolidated Copper Mines	1. 10	21, 000	35, 000, 000	385, 000
Copper Range[2]		15, 000		110, 000
Custom ores and byproducts[2]		40, 000		800, 000
Ducktown[2]		6, 000		60, 000
Kennecott group:				
Utah	1. 07	180, 000	640, 000, 000	6, 848, 000
Kennecott		20, 000		110, 000
Nevada:				
Ely[2]	[4] 1. 48	125, 000	74, 324, 000	1, 100, 000
Ray[2]	[4] 1. 65		84, 848, 000	1, 400, 000
Chino[2]	[4] 1. 40		139, 286, 000	1, 950, 000
Magma[2]		20, 000		220, 000
Miami	. 96		83, 333, 000	800, 000
Morenci (outside Phelps Dodge)	1. 00		100, 000, 000	1, 000, 000
Phelps Dodge group:				
Clay[2 3]	1. 02		239, 216, 000	2, 440, 000
Copper Queen[2]	5. 3	50, 000	12, 641, 000	670, 000
New Cornelia[2]	1. 02	70, 000	118, 627, 000	1, 210, 000
Morenci	1. 9	50, 000	75, 263, 000	1, 430, 000
United Verde	4. 65	70, 000	11, 829, 000	550, 000
Pinto Valley[3]				1, 300, 000
Quincy[2]				40, 000
Shattuck-Denn[2]		6, 000		110, 000
Tennessee Copper		6, 000		60, 000
United Verde Extension	6. 17		229, 000	14, 100
Total United States				26, 263, 000
Canada:[5]				
International Nickel[6]	4. 93		206, 705, 000	10, 190, 000
Hudson Bay	1. 71		18, 000, 000	308, 000
Noranda	7. 53		3, 426, 000	258, 000
Do	2. 00		3, 000, 000	60, 000
Do	1. 00		238, 000	2, 000
Granby	1. 81		14, 342, 000	260, 000
Sherritt-Gordon	2. 50		5, 254, 000	131, 000
Britannia	1. 30		10, 000, 000	130, 000
Amulet	3. 00		527, 000	16, 000
Total Canada			261, 492, 000	11, 355, 000
Mexico:[5]				
Phelps Dodge	2. 7		3, 500, 000	94, 000
Nicaragua:[5]				
Tonapah Nicaragua	5. 0		1, 500, 000	75, 000
Cuba:[5]				
Matahambre	5. 0		965, 000	48, 000
Grand total				37, 835, 000

[1] Leith, Kenneth, and Liddell, Donald, The Mineral Reserves of the United States and Its Capacity for Production: National Resources Committee, Washington, D. C., March 1936, p. 56.
[2] Estimates made by Ira B. Joralemon, W. W. Lynch, and C. K. Leith, Arizona Tax Commission, October 1935.
[3] Unequipped properties.
[4] Parsons, A. B., The Porphyry Coppers: 1933.
[5] Barbour, P. E., World Copper Ore Reserves: Eng. and Min. Jour., vol. 131, no. 4, Feb. 23, 1931, p. 178.
[6] Metal Bulletin (London), Nov. 17 ,1931.

PRICES OF COPPER

Table 9 gives the average price of refined copper, received by producers in the United States, by years, from 1850 to 1935. The fluctuations in prices are shown graphically in figure 5, *A* and *B*.

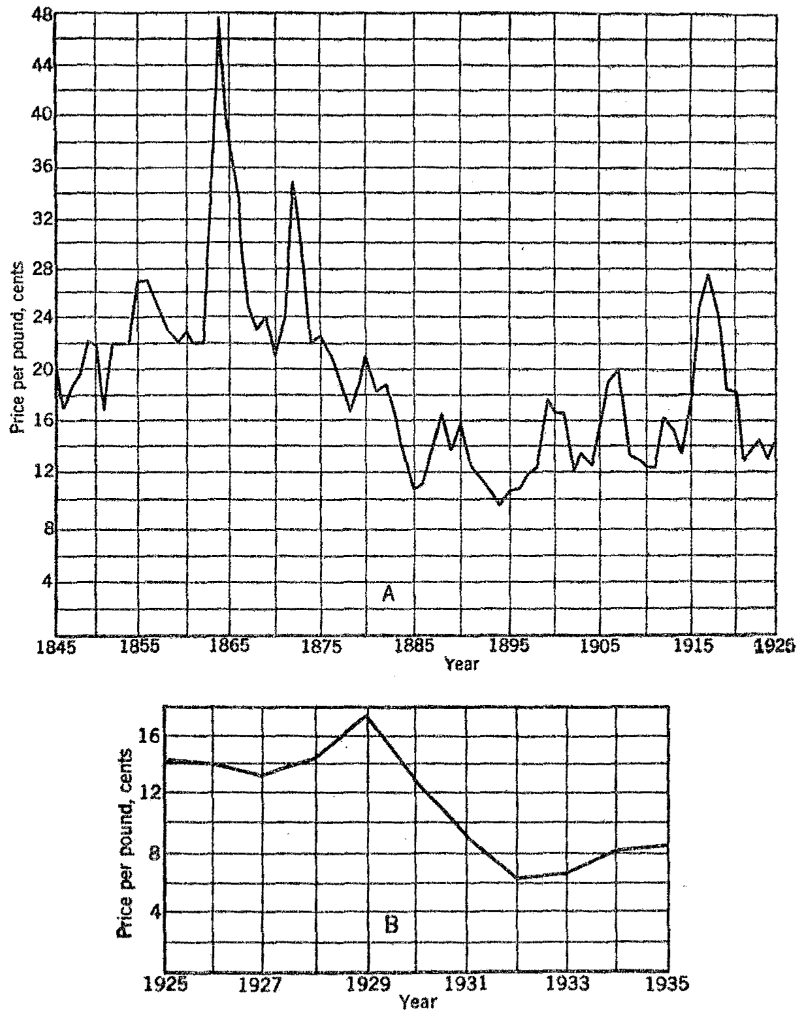

FIGURE 5.—Annual average price per pound of copper received by producers: *A*, 1845–1925; *B*, 1925–35.

TABLE 9.—*Average price per pound of copper by years, 1850 to 1935*

Year	Price	Year	Price	Year	Price	Year	Price
1850	22.0	1872	35.6	1894	9.5	1916	24.6
1851	16.6	1873	28.0	1895	10.7	1917	27.5
1852	22.0	1874	22.0	1896	10.8	1918	24.7
1853	22.0	1875	22.7	1897	12.0	1919	18.6
1854	22.0	1876	21.0	1898	12.4	1920	18.4
1855	27.0	1877	19.0	1899	17.8	1921	12.9
1856	27.0	1878	16.6	1900	16.6	1922	13.5
1857	25.0	1879	18.6	1901	16.7	1923	14.7
1858	23.0	1880	21.4	1902	12.2	1924	13.1
1859	22.0	1881	18.2	1903	13.7	1925	14.2
1860	22.9	1882	19.1	1904	12.8	1926	14.0
1861	22.2	1883	16.5	1905	15.6	1927	13.1
1862	21.9	1884	13.0	1906	19.3	1928	14.4
1863	33.9	1885	10.8	1907	20.0	1929	17.6
1864	47.0	1886	11.1	1908	13.2	1930	13.0
1865	39.2	1887	13.8	1909	13.0	1931	9.1
1866	34.2	1888	16.8	1910	12.7	1932	6.3
1867	25.4	1889	13.5	1911	12.5	1933	6.4
1868	23.0	1890	15.6	1912	16.5	1934	8.0
1869	24.2	1891	12.8	1913	15.5	1935	8.3
1870	21.2	1892	11.6	1914	13.3		
1871	24.1	1893	10.7	1915	17.5		

HISTORY

The discovery and development of copper mines have closely paralleled economic development in North America. During each period of increasing industrial activity new copper mines have come into production. For many years the copper mines of Michigan supplied the continent's needs. About 1882, Butte, Mont., became an important producer. The next large surge in production began about 1908, when copper from the so-called porphyries began to come on the market. The latest group of large copper mines to be developed is in The Pas district of Canada.

Improvements in mining methods and metallurgy have greatly influenced the production and economics of the red metal. The technique of mining has continuously improved, and mining costs have been progressively lowered; the most important advance in mining probably was the development of methods for mining the low-grade porphyry deposits on a large scale. The advent of flotation in the treatment of copper ores has increased recovery of copper in milling to a marked extent. These developments have greatly reduced the cost of producing the metal and made available immense tonnages of ore which otherwise could not have been handled at a profit.

CUBA

The first copper mining in North America was done in 1524 [8] by Spaniards at El Cobre, Santiago Province, Cuba. The Cobre mines were acquired by an English company about 1834, and considerable rich ore was shipped to Wales for smelting. In 1902 an American company started to unwater the mines, then idle for over 30 years, built a mill and a smelter, and began production; the property has been idle since 1919. The only present producer is the Matahambre mine of the American Metals Co., Ltd., which began operations in 1914.

[8] Lawrence, B. B., Two Cuban Mines: Canadian Min. Inst. Jour., vol. 13, 1910, pp. 91–106. Tuttle, E. G., Mines and Minerals, vol. 31, 1911, p. 449.

UNITED STATES [9]

ATLANTIC SEABOARD

Copper was mined at Simsbury, Conn., as early as 1709, and shortly thereafter at Hanover, N. J. However, copper mines in Vermont, discovered in 1820, were the principal source of the small production in the United States from that time until the sudden growth of the Michigan mines, about 1850.

MICHIGAN COPPER DISTRICT

Although native copper was noted by explorers in the Lake Superior region in the seventeenth century, mining did not begin until the middle of the nineteenth. In 1843 a rush started for the Upper Peninsula of Michigan, inspired largely by a report by Dr. Douglass Houghton, State geologist. The first production from a lode mine was made in 1844, when a few tons of black copper oxide was shipped from Copper Harbor at the north end of the peninsula by the Pittsburgh & Boston Mining Co.[10] In 1845, the same company opened a fissure vein on the Cliff property, at the north end of the range; this was a successful venture. Two or three other fissure veins, first mined in 1849–56, likewise proved profitable. These veins contained huge masses of native copper, some weighing 500 tons. Of the amygdaloid lodes, the Isle Royale was discovered in 1852, the Pewabic and Quincy in 1856, and the Atlantic in 1864. The great Calumet & Hecla conglomerate lode, from which has come roughly half the copper production of the district, was discovered by E. J. Hulbert, a surveyor, in 1858. In 1864 he sank a shaft into the lode on a tract of land he had acquired.[11] Two companies, the Calumet Mining Co. and the Hecla, both financed by Boston interests, after 4 or 5 years of development on the lode in the face of adverse conditions, were firmly established and prospering; they produced about 325 tons of copper monthly. These companies in 1871 were merged to form the Calumet & Hecla Mining Co., which in 1923, on combining with several other companies, became the present Calumet & Hecla Consolidated Copper Co. This organization from the beginning of operations to the end of 1934 treated about 118,000,000 tons of ore and recovered about 3,970,000,000 pounds of copper, an average of 33.7 pounds to the ton. To the end of 1930 dividends of about $185,000,000 have been paid. The concentrating mills of the company have a total daily capacity of 25,000 tons of ore and 8,000 tons of tailing; a reclamation plant has a daily capacity of 4,000 to 6,000 tons.

The second largest company in the Lake Superior copper district is the Copper Range Co.; it was formed in 1899 to operate the Champion mine and soon acquired two more mines on the Baltic lode which was discovered in 1882 and opened in 1897. The three principal mines of this company, the Champion, Tri-Mountain, and Baltic, from about 1900 through 1934 yielded 1,016,000,000 pounds of copper. The mill has a capacity of 4,000 tons daily.

[9] For an excellent short history of copper mining in the United States see Richter, F. E., Copper-Mining Industry in the United States, 1845–1925: Quar. Jour. Econ., vol. 41, 1927, pp. 236–291, 684–717.
[10] Butler, B. S., and Burbank, W. S., The Copper Deposits of Michigan: Geol. Survey Prof. Paper 144, 1929, p. 63.
[11] Agassiz, G. R., Alexander Agassiz and the Early History of the Calumet & Hecla: Min. Cong. Jour., vol. 17, 1931, p. 473.

The yield of the Lake Superior district to the end of 1934 has been about 8,668,000,000 pounds of copper, or 18 percent of the country's total output. This is shown graphically in figure 1.

TENNESSEE

DUCKTOWN

In 1847, 2 years after the discovery of the Cliff fissure in the Lake Superior district, A. J. Wevar, or Webber, shipped 15 tons of 25-percent copper ore from the Burra Burra lode, in the Ducktown district, Tennessee, to the Revere smelter at Boston.[12] In 1850, three other mines were opened, and by 1855 all deposits of later importance had been discovered. Very rich chalcocite-pyrrhotite ores were then mined; these occurred at shallow depths in a zone of secondary enrichment below the leached barren gossans. Between 1854 and 1860 several smelting plants were erected, which treated roasted ore in small charcoal furnaces.

In 1858, the Union Consolidated Co. was formed, which controlled about half of the developed ore bodies. The rich, black ores were exhausted in 1879, and the mines were closed until 1890. At this time, owing to improved rail connections and the introduction of blast furnaces, the former Union Consolidated properties were reopened by an English concern, the Ducktown Sulphur, Copper & Iron Co., Ltd., which in 1925 became the Ducktown Chemical & Iron Co. In 1901 the production of this company was about 3,000,000 pounds.

The Tennessee Copper Co. was organized in 1899. By the middle of 1901 a smelter had been built and production started on a basis of 1,000 tons a day. In 1907, after 3 years of experimental work, design, and construction, the Tennessee Copper Co. began to produce sulphuric acid (as did the Ducktown company 2 years later). This ended a serious smoke nuisance and led eventually to an annual production of 300,000 to 500,000 tons of sulphuric acid and the entrance of the Tennessee company into the fertilizer business on a large scale.

From the beginning of operations through 1928 the Ducktown district produced about 519,500,000 pounds of copper. In recent years (1921–28) the average yield of copper from ores of the district has dropped gradually, ranging from 1.08 to 1.34 percent, except in one year, when it was 1.53 percent.

Production figures for 1929–34 are not available for publication. The Ducktown Chemical & Iron Co. operated throughout 1934 on a curtailed basis, although it doubled its 1933 production.

MONTANA

BUTTE DISTRICT

Placer gold was discovered at Butte, in Silver Bow County, Mont., in 1864 and the first silver vein the next year.[13] Some of the copper

[12] Emmons, W. H., and Laney, F. B., Geology and Ore Deposits of the Ducktown Mining District, Tenn.: Geol. Survey Prof. Paper 139, 1926, pp. 30–33.
[13] Weed, W. H., Geology and Ore Deposits of the Butte District, Mont.: Geol. Survey Prof. Paper 74, 1912, p. 18.

veins were located in 1864. Early attempts at copper smelting were failures, but some rich ore was hauled 400 miles to the railroad at Corinne, Utah, and thence shipped to various custom smelters. This eventually attracted the attention of a Colorado smelting company, which built a smelter at Butte about 1880. The advent of the railroad in 1881 led to increased activity, and in the succeeding 15 years a number of mills and smelters were erected. Copper production mounted steadily, from 9,000,000 pounds in 1882 to 222,000,000 pounds in 1896; in 1887 the Butte production first exceeded that of the Lake Superior district.

The Amalgamated Copper Co., formed in 1899 as a holding corporation, gradually acquired ownership or control of the more important properties, including the Anaconda Copper Mining Co. In 1915 the Amalgamated Co. was dissolved, leaving most of the mines of the district under the control and management of the Anaconda Co. This consolidation resulted in tremendous operating benefits, elimination of property lines, avoidance of lawsuits, and general unification of effort.[14] It also brought into existence the largest copper-mining company in the world.

Production of copper in Montana during 1934 and 1935 was only about 22 percent of the peak of 1929. The output was nearly all from the Anaconda mines at Butte. The gross production of Silver Bow County from 1882 through 1934 was about 10,500,000,000 pounds of copper valued at $1,580,000,000; silver, zinc, gold, and lead, named in order of decreasing importance, swelled the total value to $2,200,000,000. Nearly all of this came from the mines now (1936) owned by the Anaconda Co. in the Butte district. Dividends of about $310,000,000 had been paid by this company to the end of 1930.

The second largest copper producer in the Butte district in recent years has been the East Butte Copper Mining Co. It was organized in 1905 but produced only small amounts for a few years. In 1909 it acquired the Pittsmont mine and smelter at Butte and until 1923 produced 10,000,000 to 20,000,000 pounds of copper annually, partly from direct smelting ores with a copper content of about 6 percent and partly from a somewhat larger tonnage of milling ore. After 1923 production dwindled, but the company operated its own mill and smelter until 1924, after which its ore was concentrated and smelted by the Anaconda Co. The mine was closed in October 1930, and the company was liquidated early in 1931; it had produced roughly 275,000,000 pounds of copper and paid dividends of about $2,600,000.

In 1930 the third largest copper producer in Montana was the North Butte Mining Co., which mined about 66,000 tons of copper ore containing about 5,500,000 pounds of copper. This company, whose chief mine is the Granite Mountain, was one of the spectacular successes of the Butte district from 1905, when it was organized, to 1920, when its production fell off sharply. In that period dividends of nearly $15,000,000 were paid on a capitalization of $6,450,000 (stock issued). Production figures prior to 1910 are not available; from 1910 through 1930 the mine produced about 265,000,000 pounds of copper from 4,900,000 tons of ore. More than 200,000,000 pounds, however, were produced in the first 10 years of this period.

[14] Daly, W. B., Evolution of Mining Practice at Butte: Eng. and Min. Jour., vol. 128, Aug. 24, 1929, pp. 280–285.

NEW MEXICO

SANTA RITA

The Chino mine at Santa Rita, N. Mex., is the oldest of the large copper mines on the North American Continent; it was first worked in 1804 by Manuel Elguea, of Chihuahua, Mexico, under a contract to supply copper for the coinage of the Spanish Government of Mexico.[15] During most of the nineteenth century the mine was worked intermittently by Elguea and his heirs and by a succession of American, French, and Spanish frontiersmen. The hostility of the Apaches necessitated a military guard and a strong fort for refuge in case of attack and caused abandonment of the venture for short periods. In 1873 Denver interests bought the property from its current owners, who presumably held title under American locations made after the treaty of Guadalupe Hidalgo. The Land Office, however, refused patents to the ground until after the new occupants had traced and settled with the heirs to the Elguea Estate, by then scattered over Mexico and Europe.

Early miners had worked the deposit for the native copper of the outcrops and upper zones of the ore bodies. The new owners erected a small smelting plant and shipped not only high-grade ore but black copper to the nearest railroad, in Colorado. In 1881, again under different ownership, a concentrator was erected. In 1891, a railroad entered the district. For a few years after 1899, while a group of men associated with Amalgamated Copper Co. (Anaconda) had control, lessees made a considerable output. In 1905 and 1906 John M. Sully examined the property for the Hermosa Copper Co., a subsidiary of the General Electric Co. It was 1909, however, before the Chino Copper Co. was organized to exploit the deposit, late 1910 before steam-shovel work started, and a year later when the first ore went through the new mill at Hurley, 10 miles from the mine. The Chino Co., identified from the start with Utah Copper Co. interests, was absorbed by the Ray Consolidated Copper Co. in 1924, and the latter was absorbed by the Nevada Consolidated Copper Co. in 1926. In 1933 the Kennecott Copper Co. acquired all the properties of the Nevada Consolidated Copper Co., which is now known as the Nevada Consolidated Copper Corporation.

The Chino mine is estimated to have produced about 1,334,000,000 pounds of copper from 1804, when production began, to the end of 1934.[16] In 1929, 3,973,300 tons of ore were mined; the net production of copper was probably about 85,000,000 pounds. Operations were suspended in October 1934; from 1931 to 1934 the mine was operated at about 20 percent of capacity.

ARIZONA

MORENCI[17]

In 1870 members of a scouting party of the United States Army discovered placer gold and outcrops of copper lodes in the Morenci district of Arizona. In 1872, after the retirement of the Apache Indians under Cochise to their reservation, a number of prospectors returned to the district to hunt for gold, and several copper claims

[15] Thorne, H. A., Mining Practices at the Chino Mines, Nevada Consolidated Copper Co., Santa Rita, N. Mex.: Inf. Circ. 6412, Bureau of Mines, 1931, 28 pp.
[16] Thorne, H. A., work cited, p. 4.
[17] Colquhoun, James, The History of the Clifton-Morenci Mining District: London, 1924.
Hamilton, Patrick, The Resources of Arizona: San Francisco, 1884.
Wendt, A. F., The Copper Ores of The Southwest: Trans. Am. Inst. Min. Eng., vol. 15, 1886, pp. 25–77.

were located. In 1873 a small adobe cupola was built and some rich copper ore smelted; the product was hauled 700 miles to the nearest railroad at La Junta, Colo., whence it was shipped to Baltimore for refining. In 1880 Lezinsky Bros., who had become the owners of the principal mines, were producing 200,000 pounds of copper monthly from 25-percent ore from the Longfellow mine. In 1882 their holdings were sold to the Arizona Copper Co., of Edinburgh, Scotland. This company erected a smelter at Clifton on the San Francisco River and built a 70-mile railroad to the Southern Pacific R. R. at Lordsburg, N. Mex., and a narrow-gage road to the mines.

About 1880 Phelps, Dodge & Co., of New York, on the advice of Dr. James Douglas, acquired a half interest in the Detroit Copper Mining Co., which held a group of claims in the Morenci district and needed money to build a smelter. At first a small smelting plant was built at Morenci and then, in 1884, a larger one; the latter plant produced 3,345,000 pounds of copper in 1885. In 1886, after the bonanza ores were exhausted, both companies in the district built concentrators. New concentrators and converter plants were built for both mines between 1895 and 1900.

In 1900 the Shannon Copper Co. started operations in the district. The company mined and smelted chiefly oxide ores from claims adjoining those of the Arizona Copper Co., to which it sold out in 1919 after a very profitable career. Two years later the Arizona Copper Co. was merged with the Detroit Copper Co. of the Phelps Dodge Corporation, which has since been the only operator in the district.

When operations were resumed in 1922, after a general shut-down, the block-caving method was used to mine the Humboldt ore body; since then all ore from the district has been produced by this method, except an insignificant amount mined by lessees.

Production was maintained at about 5,000 tons of concentrating ore per day during 1930–32. In July 1932, owing to the low price of copper, operations were discontinued indefinitely.

In 1930 and 1931, diamond drilling in the Clay ore body indicated ore reserves of approximately 200,000,000 tons, having an average grade of about 1 percent of copper; the estimate was later raised to 284,000,000 tons carrying 1.036 percent copper by further drilling. The open-cut method of mining will probably be used in this ore body when operations are resumed.

WARREN (BISBEE) [18]

The first mining location in the Warren district of southern Arizona was made in August 1877 by John Dunn, a scout attached to the Army post at Bowie. Ore from a large and rich body on the Copper Queen claim was smelted in 1878 and a small shipment of matte made. About 1880 this claim was sold to San Francisco men, who built a new furnace and worked for a few years with fair success; dividends of $1,350,000 had been paid to the end of 1884.

In 1881, on the advice of Dr. James Douglas, the metal importing and manufacturing firms of Phelps, Dodge & Co., of New York,

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[18] Douglas, James, The Copper Queen Mines and Works, Part 1, Historical Sketch: Trans. Inst. Min. and Met., vol. 22, 1913, pp. 532–550.
Ransome, F. L., The Geology and Ore Deposits of the Bisbee Quadrangle, Arizona: Geol. Survey Prof. Paper 21, 1904, pp. 13–16.
Douglas, Walter, Historical Sketch (of the Phelps Dodge Enterprise): Eng. and Min. Jour., vol. 126, 1928, pp. 642–643.
Elsing, M. J., The Bisbee Mining District—Past, Present, and Future: Eng. and Min. Jour., vol. 115, 1923, pp. 177–184. (Contains bibliography on district.)
Copper Queen Bulletin, Bisbee, Ariz., June 1922, p. 7.

acquired the Atlanta claim, adjacent to the Copper Queen, and formed the Atlanta Mining Co. which struck ore in 1884. In 1885 the two companies were merged to form the Copper Queen Consolidated Mining Co., which built a concentrator and a larger smelter, acquired additional property, and about 1888 made a rail connection with Fairbank, on the Southern Pacific R. R. In 1893 the first converters were built to permit treatment of ores higher in sulphur. In 1900 a new smelter was built at Douglas at the terminal of the railroad from Nacozari, Sonora, Mexico, where the Phelps Dodge interests had recently acquired the Pilares mine.

About 1900 a group of Pittsburgh and Lake Superior men started prospecting a number of claims in the Warren district and met with almost immediate success; these claims were merged, constituting the nucleus of the Calumet & Arizona mine, which was the second largest in the district until its acquisition by the Phelps Dodge Corporation in 1931. This company likewise built a smelter at Douglas; in 1903 the first dividend was paid.

Until after the World War the mines of the Warren district produced direct-smelting ores, chiefly by square-set stoping methods. In 1923, after several years of pilot-plant operation, churn drilling, and stripping, the Copper Queen began milling 4,000 tons daily of low-grade porphyry ore from a steam-shovel pit on Sacramento Hill, in the center of the district. Low-grade material from the pit was leached in heaps, and at one time 1,000,000 pounds of copper was produced monthly, at very low cost, in the form of precipitates. In 1925 the block-caving method was used to mine one section of the porphyry ore body; in 1929 steam-shovel work in Sacramento Hill ceased, and the glory-hole method was used to mine the ore remaining in the bottom of the pit.

Table 10 shows the total production of Cochise County and the Warren district through 1934. In 1929 the production from 2,800,000 tons of ore was 186,000,000 pounds of copper, $1,410,000 in gold, 2,260,000 ounces of silver, and 2,040,000 pounds of lead, valued in all at $35,500,000.

The Phelps Dodge Corporation has paid dividends of about $129,000,000 since 1885; the Calumet & Arizona, of about $77,000,000; and the Shattuck-Denn, a third company in the district, of about $8,000,000 (mostly from lead ore).

TABLE 10.—*Total production of major metals, Cochise County and Warren district*

COCHISE COUNTY [1]

Date	Ore, tons	Copper, pounds	Gold, ounces	Silver, ounces	Lead, pounds	Zinc, pounds	Total value
1858-90	------------	270, 000, 000	227, 362	23, 500, 000	4, 640, 000	------------	$65, 790, 000
1900-1934	48, 398, 461	4, 156, 305, 124	1, 454, 953	69, 433, 893	204, 318, 198	16, 612, 464	754, 396, 266
1858-1934	------------	4, 426, 305, 124	1, 682, 315	92, 933, 893	208, 958, 198	16, 612, 464	820, 186, 266

WARREN DISTRICT

Date	Ore, tons	Copper, pounds	Gold, ounces	Silver, ounces	Lead, pounds	Zinc, pounds	Total value
1902-33 [1]	39, 355, 735	3, 872, 407, 999	23, 215, 240	44, 531, 919	158, 120, 842	14, 944, 997	$688, 957, 615
1934 [2]	521, 963	71, 110, 775	1, 717, 150	2, 318, 908	127, 540	------------	8, 907, 370
1902-34	39, 877, 698	3, 943, 518, 774	24, 932, 390	46, 850, 827	158, 248, 382	14, 944, 997	697, 864, 985

[1] From Elsing, Morris J., and Heineman, Robert E. S., Arizona Metal Production: Arizona Bureau of Mines Bull. 140, 1936.

[2] 1934 figures from Bureau of Mines Mineral Yearbook 1935.

AJO [19]

The mines in the Ajo district were worked earlier than those in any other large copper camp of Arizona. About 1855, rich native copper and culprite ore, mined from small deposits on the edge of what is now known as a large body of low-grade porphyry ore, were hauled in oxcarts first to San Diego, later to Yuma, and thence shipped to Swansea, Wales, for smelting. The building of the Southern Pacific R. R. in 1876 brought rail transportation to Gila Bend, 43 miles by wagon north of Ajo.

Despite previous reports of unfavorable results the Calumet & Arizona Mining Co., under the direction of John C. Greenway, in 1911 took an option on the New Cornelia Copper Co., one of the two major properties in the district. Within 2 years drilling had developed 12,000,000 tons of carbonite ore and 28,000,000 tons of sulphide ore, of which both averaged close to 1.5 percent of copper. The mine was developed as an open pit. A 5,000-ton leaching plant was built and leaching was begun in May 1917 and concluded in July 1930, when practically all the carbonate ore had been mined. In January 1924 a 5,000-ton concentrator was put in operation to treat the sulphide ore; this plant in 1928 and 1929 was enlarged to a rated capacity of 16,000 tons per day. Concentrates were shipped to the Calumet & Arizona smelter at Douglas, Ariz.

Two adverse factors in the early mining ventures had been lack of water supply and high transportation costs; before starting construction, the New Cornelia Copper Co. built a railroad from Gila Bend and developed water by sinking a shaft in the valley 7 miles north of the mine.

In 1917 the New Cornelia Co. purchased the adjoining property of the other major company in the district. In 1929 the New Cornelia Co. was merged with the Calumet & Arizona, and in 1931 it became the New Cornelia Branch of the Phelps Dodge Corporation.

To January 1, 1934, 36,200,000 tons of ore averaging 1.42 percent of copper had been mined and treated. Of this total, 17,300,000 tons were oxidized ore which averaged about 1.4 percent of copper and 18,900,000 tons sulphide ore which averaged about 1.5 percent copper.

The total production of the mine from the beginning of leaching operations in 1917 through 1934 was about 36,200,000 tons of ore, which yielded about 803,851,000 pounds of copper, an average of 22.2 pounds of salable copper per ton of ore. The New Cornelia Copper Co., from 1918 to the time of its absorption by the Calumet & Arizona in 1929, paid $18,630,000 in dividends. Operations were suspended early in April 1932 until July 1, 1934.

The reserves available in 1936 for extraction by open-pit methods were estimated at 155,000,000 tons of ore averaging 1 percent copper. An additional 40,000,000 tons, probably at too great a depth for open-cut mining, lie to the south of the pit.

JEROME [20]

The ore body of the United Verde mine at Jerome was discovered in 1876 by M. A. Ruffner, a prospector. The property was sold to New York men, who in 1882 organized the United Verde Copper

[19] Ingham, G. R., and Barr, A. T., Mining Methods and Costs at the New Cornelia Branch, Phelps Dodge Corporation, Ajo, Ariz.: Inf. Circ. 6666, Bureau of Mines, 1932, pp. 1–3.
[20] Rickard, T. A., The Story of the U.V.X. Bonanza: Min. and Sci. Press, vol. 116, 1918, pp. 9–16, 47–52.

Co. The same year the railroad was built through Ash Fork 60 miles north of the mine; with a possible route to market thus provided, a smelter was built which in 1883 and 1884 was operated with fair success. Then leaner ores and rising mining costs halted operations for a few years. In 1889 W. A. Clark, a mine owner and operator of the Butte district, Montana, acquired control of the United Verde mine, and production was resumed. Rail connections were completed in 1894. Between 1910 and 1915 a new smelter was built on the Verde River, and the annual output was nearly doubled—from about 400,000 to 800,000 tons. In 1923, after the postwar curtailment, production again increased until 1929, when the smelter treated about 1,750,000 tons of ore and concentrates. In 1930 the main ore shaft was being sunk from the 3,000 to the 3,600 level.

Underground mining in recent years has been done largely by cut-and-fill methods, although other methods are used where conditions demand. Power-shovel mining of the upper levels of the ore body, which were on fire, was begun in 1918. By the end of 1929 over 10,000,000 cubic yards of material had been removed, including nearly 8,500,000 yards of waste stripping and over 2,000,000 yards (5,000,000 tons) of ore. Mining was done in the face of unusual and considerable difficulties caused by the fire.

The United Verde Copper Co. has not released production figures for a number of years. From 1888 through 1922 production totaled 1,111,971,696 pounds of copper, 563,375 ounces of gold, and 18,406,232 ounces of silver having a value of approximately $167,000,000.[21] Production from 1923 through 1930 was probably about 850,000,000 pounds of copper, making a total copper production of almost 2,000,000,000 pounds. The company is reported to have paid dividends, from 1892 through 1928, of more than $66,847,000.[22]

Production at the mine ceased from the spring of 1931 until early in 1935, when mining in the open pit was resumed. The property was purchased by the Phelps Dodge Corporation in February 1935 and is now known as the United Verde Branch of the Phelps Dodge Corporation.

The Jerome district contains one other large mine besides the United Verde. After 2 or 3 years of fruitless development work the United Verde Extension Mining Co. at the end of 1914 opened up an unusually rich ore body. Production started in 1915 and jumped to a peak in 1917, when 63,000,000 pounds of copper was obtained from 115,064 tons of ore. The ore has been mined largely by square-set methods. A smelter was built in 1917–18 at Clemenceau and a 200-ton concentrator in 1930. In 1930 the mine produced 300,000 tons of direct smelting and concentrating ore, which averaged 6.65 percent copper and 0.04 ounce of gold and 1.27 ounces of silver to the ton; the yield of copper was about 39,000,000 pounds. The total production from company ores from 1915 through 1931 was about 3,570,000 tons of ore, which yielded 756,000,000 pounds of copper, $3,017,000 in gold, and $4,183,000 in silver, with a total value of $127,495,000. Dividends from 1916 to the end of 1930 amounted to $39,742,500. Early in 1936 it was reported that the ore bodies were about exhausted.

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[21] Lindgren, Waldemar, Ore Deposits of the Jerome and Bradshaw Mountains Quadrangle, Ariz.: Geol. Survey Bull. 782, 1926, p. 63.

[22] The Mines Handbook, vol. 18, 1931, p. 448.

GLOBE-MIAMI [23]

Silver was discovered in the Globe district in 1874 and was the center of interest until 1881, when copper mining began. In 1882 the Old Dominion Copper Mining Co. took over and began to operate the property of the Globe Mining Co. at Globe. A smelting plant was set up which comprised two 36-inch water-jacketed furnaces, each capable of smelting 40 tons daily.

In 1888 a new shaft proved the ore body to be of considerable extent, and in 1892 a new and larger smelter was blown in. The new shaft likewise proved the existence of a heavy flow of water, which thereafter remained one of the major problems of the mine (3,660,000 gallons was pumped daily in 1928). The wagon haul from the railroad was 130 or 140 miles during the early period; freighting of supplies and copper bullion cost $15 to $36 a ton. The railroad was built to Globe in 1898.

The Phelps Dodge Corporation, which already owned the United Globe, a neighboring mine on the same vein, gained control of the Old Dominion Copper Mining Co. in 1902; the Old Dominion Co. was formed in 1904. A gravity concentrator with a capacity of 300 tons a day was completed in 1905; this was replaced in 1914 by an 800-ton plant which included not only tables and vanners but a flotation machine. In 1923 the mill went on an all-flotation basis, and in 1930 it had a capacity of 1,500 tons a day. The smelter was closed in 1924; since then the crude ore and concentrates have been shipped to the International smelter at Miami. From 1904 to the end of 1930 the Old Dominion Co. produced more than 600,000,000 pounds of copper. Returns to stockholders have totaled nearly $19,000,000. The mine was closed in November 1931; it was still idle at the time of writing (spring of 1936).

The bulk of the copper production of the Globe-Miami district has come from two large mines, the Miami and the Inspiration. The Miami Copper Co. was organized late in 1907 to develop a property on which the General Development Co. (a Lewisohn concern) had already found disseminated copper ore.[24] After 15 months' work and 2,500 feet of drifting from the discovery shaft, 2,000,000 tons of 3-percent ore was blocked out.[25] The railroad was extended from Globe to Miami and a concentrator built; milling began in 1911. Top-slicing methods were used at first to mine the main ore body, which contained 18,000,000 tons of relatively high grade ore (1.5 to 2 percent). A caving method developed in other ore bodies was used in 1919 to mine the main ore body, which was exhausted in 1924. Reserves at this time comprised 35,000,000 tons of sulphide ore containing 1 percent copper. A low-cost caving method was devised which permitted mining this low-grade ore at a profit. The milling plant, through continued remodeling, had been changed from a 2,000-ton gravity concentrator to a 6,500-ton all-flotation plant.

[23] Walker, A. L., Early-day Copper Mining in the Globe District of Arizona: Eng. and Min. Jour., vol. 125, 1928, pp. 604–608, and 694–698.
Ransome, F. L., Geology of the Globe Copper District, Arizona: Geol. Survey Prof. Paper 12, 1903, pp. 114–118.
[24] Ransome, F. L., The Copper Deposits of Ray and Miami, Ariz.: Geol. Survey Prof. Paper 115 1919, pp. 19–22.
[25] Maclennan, F. W., History of the Development of Miami Copper: Min. Jour. (Phoenix), vol. 13, no. 13, 1929, p. 9.

The capacity was increased in 1924 to 10,000 and later to 18,000 tons a day for treating the low-grade ore.

Until May 15, 1932, when operations were suspended, the Miami Copper Co. mined and milled about 55,000,000 tons of ore and produced about 1,083,000,000 pounds of copper. Dividends from 1912 through 1930 amounted to $37,300,000. Ore reserves at the end of 1932 were approximately 78,500,000 tons of sulphide ores averaging 0.88 percent copper and 6,900,000 tons of mixed ores averaging 1.88 percent copper. A leaching plant was built to treat the mixed ores in 1933 and 1934. In the spring of 1936 the mine was being operated on a reduced scale.

The Inspiration Copper Co. was formed in December 1908 and at once began to develop its ground, which was adjacent to that of the Miami Copper Co. "By the beginning of 1911 the Inspiration Copper Co. had driven about 27,000 feet of drifts and crosscuts, had put down over 80 drill holes, and had developed over 21,000,000 tons of ore." [26] In 1912 the Inspiration Co. acquired the Live Oak Development Co. property and in 1915 that of the New Keystone Co., which contained the principal ore bodies at Miami not held by the Inspiration or Miami companies; ore reserves of the Inspiration Copper Co. were increased to about 45,000,000 tons of sulphide ore containing 2 percent copper.

An 18,000-ton gravity and flotation plant began operation in June 1915. Meanwhile the International Smelting Co. had built a smelter at Miami and contracted to treat the products of the two concentrators. A 7,500-ton leaching plant was built and began treating mixed sulphide and oxide ore late in 1926. The total production from 1915 through May 1932, when operations were suspended, was about 72,000,000 tons of ore, which yielded 1,334,000,000 pounds of copper. Dividends for the same period were slightly over $53,000,000.

Ore reserves on December 31, 1935, were approximately 69,000,000 tons with a grade of 1.373 percent copper. Operations were resumed at the mine on a reduced scale late in 1935.

RAY

Gold mining was in progress at Ray in 1880. A copper-mining company was formed in 1883.[27] Another company built a mill in 1900 but was unable to treat the low-grade ore profitably. In 1907 D. C. Jackling and others connected with the Utah Copper Co. organized the Ray Consolidated Copper Co. In a little over 2 years, during which time $300,000 was expended in prospecting, the company developed about 50,000,000 tons of ore.

Actual production from the Ray Consolidated Copper Co. mines was begun in March 1911. Up to that time the company had spent approximately $10,000,000 in land, preparatory work, and equipment. This work included the drilling of more than 350 churn-drill holes to an average depth of 418 feet, the sinking and equipping of two main shafts, the driving of about 30 miles of drifts and crosscuts, the installation of adequate water works at Ray and Hayden, the construction of a standard-gage railway between Kelvin and Ray, the completion of three 1,000-ton units of an 8,000-ton concentrating mill at Hayden, the construction of a power plant capable of generating 10,000 kilo-

[26] Ransome, F. L., work cited, p. 21.
[27] Thomas, R. W., Mining Practice at Ray Mines, Nevada Consolidated Copper Co., Ray, Ariz.: Inf. Circ. 6167, Bureau of Mines, 1929, p. 2.

watts at Hayden, the building of a transmission line between this plant and the mines, and the erection of numerous buildings for various purposes.[28]

A smelter, started at Hayden by the Ray company in 1912, was taken over and completed by the American Smelting & Refining Co. The Ray company eventually absorbed several smaller companies holding ground in the district; in 1924 the Ray company and the Chino Copper Co. of New Mexico were merged, and in 1926 both were absorbed by Nevada Consolidated Copper Co. In 1933 the Kennecott Copper Co. acquired all of the properties of the Nevada Consolidated.[29] Operations at Ray were suspended in March 1933.

An undercut block-caving method of mining was used at Ray from the start; the system, however, has been greatly modified to suit various conditions. To the end of 1934 about 51,000,000 tons of ore had been mined and milled; it yielded roughly 1,155,000,000 pounds of copper.

Before the merger, dividends of over $25,000,000 had been paid to Ray stockholders. Since then they have participated in $12,000,000 or $13,000,000 additional from Nevada Consolidated earnings.

Reserves at the end of 1929 were estimated at 85,000,000 tons of 1.65-percent ore, or almost double the tonnage of ore of nearly the same grade that had been mined to that time.

SUPERIOR

The famous Silver King ore body, about 3 miles north of Superior, was discovered in 1875; this was the first discovery in the district. The Silver Queen claim, now owned by the Magma Copper Co., was located the same year. Some silver-copper ore was mined from the Silver Queen about 1880,[30] but from then until the formation of the Magma Copper Co., in 1910, the ground lay idle. Shipments of 26-percent copper ore began in 1912. This was trucked 30 miles to the railroad at Florence. In 1914 a 150-ton gravity-flotation mill was built; in 1915 its capacity was doubled, and a narrow-gage railroad was completed from Florence to Superior; this was rebuilt to standard gage in 1923. In 1922 further changes in the mill and the addition of a third section brought the capacity to 600 tons daily. In 1924 a smelter was completed and production stepped up to 750 tons daily.

The mine at present (1935) is developed to the 3,800-foot level. The ore bodies, comprising bornite, chalcopyrite, and pyrite, with a gangue of quartz and silicified country rock, occur in a large single vein. The ore is mined by a combination of rill and pillar stopes, usually heavily timbered and filled.

The total production of the Magma Copper Co., from about 3,167,000 tons of its own ores, to the end of 1934 was 387,000,000 pounds of copper, 11,600,000 ounces of silver, and 150,000 ounces of gold. Total dividends paid to the end of 1930 were $9,663,000.

UTAH

BINGHAM

The Bingham district of Utah ranks third in total production among North American copper districts; it was the largest producer in 1929

[28] Ransome, F. L., work cited, p. 19.
[29] Parsons, A. B., The Porphyry Coppers: New York, 1933, p. 203.
[30] Ransome, F. L., Copper Deposits near Superior, Ariz.: Geol. Survey Bull. 540, 1914, p. 143.

and in 1930 was second only to Butte. This position is due to the mine of the Utah Copper Co., the greatest of the North American porphyry copper mines. From the first production in 1905 to the end of 1934 it yielded about 11 percent of the total production of the United States for the same period.

The first three mineral locations in Bingham Canyon were made September 17, 1863, on lead-silver lodes. The first sustained production of copper in the district was in 1896, from the Highland Boy mine. The Utah Consolidated Gold Mines, Ltd., acquired the mine in 1897. This company was reorganized in 1903 as the Utah Consolidated Mining Co., under Lewisohn control; in 1924, as a result of a disastrous apex suit, it was sold at foreclosure to the International Smelting Co. and reincorporated as the Utah Delaware Mining Co. The production of the Highland Boy in 1899 was between 2,000,000 and 3,000,000 pounds of copper; the output jumped rapidly to more than 10,000,000 pounds annually and ranged from 5,000,000 to 20,000,000 pounds until after the World War. About 1912, however, lead production began to increase, and in recent years the output of lead and zinc has been more important than the output of copper, which has fallen to 3,000,000 or 4,000,000 pounds annually.

The success of the Highland Boy stimulated interest in copper mining elsewhere in the district. At first production was confined to the sulphide deposits in limestone; in 1898, however, a company was formed in England, known as the Boston Consolidated Gold & Copper Mining Co., Ltd., which proposed mining operations on the basis of huge reserves (291,000,000 tons) of ore, ranging from 0.75 to 2.50 percent copper. In 1899, regardless of the ridicule and criticism aroused by the announcement of the Boston Consolidated plans, D. C. Jackling and R. C. Gemmel reported favorably on a similar and adjoining property held and partly developed by Enos A. Wall.[31] The report showed 12,000,000 tons of ore averaging 2 percent copper. It also estimated that copper could be produced at a cost of 6 cents a pound on a basis of 2,000 tons of ore daily with open-pit methods of mining. In 1903 the undertaking was financed, and in July 1904 the first "porphyry" mill in the country began regular operation. Mining at first was done by underground methods. Presently, after a careful examination, the Guggenheim interests furnished the large capital necessary to build a 6,000-ton mill and strip the overburden to a point where steam-shovel production could start; they also, through the American Smelting & Refining Co., built a smelter at Garfield which contracted to take the concentrates from the mill. Stripping was begun in 1906, and the first section of the new mill was started in 1907.

The Boston Consolidated Co. started operations soon after open-cut work began at the Utah Copper property. A concentrator was built at Arthur at the same time that the Utah Copper plant was constructed. In 1910 this company was consolidated with the Utah Copper Co., which at the same time acquired a controlling interest in the Nevada Consolidated Copper Co. This eventually brought four great porphyry-copper mines—Utah, Nevada, Ray, and Chino—under the management of Jackling, who had been in charge of the Utah Copper Co. from the start.

[31] Parsons, A. B., The Porphyry Coppers: New York, 1933, pp. 53-62.

The Utah Copper Co. expanded its production steadily from more than 40,000,000 pounds in 1908 to more than 195,000,000 pounds in 1917, and after the post-war depression its output reached a peak of 296,000,000 pounds in 1929. The copper in 1929 was derived from the treatment of 17,700,000 tons of ore which contained 0.99 percent copper and yielded also about 13 cents in gold and 0.06 ounce of silver to the ton. The Utah Copper Co. produced, from 1905 through 1934, about 3,834,000,000 pounds of copper from the mining and treating of about 223,000,000 tons of ore. It paid dividends to the end of 1931 of about $228,473,000, of which $40,000,000 or $45,000,000 represented the dividends of the Nevada Consolidated Copper Co.

Present ore reserves at Bingham are estimated at 632,000,000 tons, with an average grade of 1.01 percent copper.

NEVADA

ELY

The second of the great porphyry copper mines to produce in the United States was the mine of the Nevada Consolidated Copper Co. at Ely, Nev. The Ely (Robinson) mining district, in which this property lies, was organized in 1868,[32] but little interest was shown in copper mining, except for a few small shipments of rich ore to San Francisco, Baltimore, or Salt Lake, until after 1900. The wagon haul was 150 miles. In 1902 low-grade copper ore had been developed on one or two properties, and a concentrator was under construction. In 1903 M. L. Requa formed the White Pine Copper Co. In 1904 this company, of which the Ruth claim was the nucleus, and the New York & Nevada Copper Co., with property at Copper Flat, were united as the Nevada Consolidated Copper Co.; their combined ore reserves at that time were estimated as 26,000,000 tons.[33] A railroad was completed to Ely and a smelter built at McGill in 1906; in 1908 the milling of porphyry ore was begun.

Production started in 1908, with about 15,000,000 pounds of copper, which was doubled the next year and again the next. At first mining was confined to open-pit work at the Copper Flat mine, but in 1915 enough underground development had been done for large-scale production to be started at the Ruth mine, the caving system being used. Since then the two operations have been carried on together. Production reached a peak of about 110,000,000 pounds of copper from 5,220,000 tons of ore in 1929.

In 1926 the Nevada Consolidated Copper Co. absorbed the Ray Consolidated Copper Co., which had just absorbed the Chino Copper Co. Since then it has been impossible to segregate the dividend of the mother company into the separate units, and the copper production of the different mines has not been published. However, the estimated output of the Copper Flat and Ruth mines, from 1908 through 1934, is 1,652,000,000 pounds of copper from 72,900,000 tons of ore. Parsons [34] estimates that a total dividend of $60,000,000 can be attributed to the Nevada unit through 1930.

Since the formation of the Nevada Consolidated the remaining productive ground has been brought under the ownership of the Consolidated Copper Mines Co. This company produced a total of

[32] Spencer, A. C., The Geology and Ore Deposits of Ely, Nev.: Geol. Survey Prof. Paper 96, 1917, p. 92.
[33] Parsons, A. B., Work cited, p. 118.
[34] Parsons, A. B., Work cited, p. 133.

40,000,000 to 45,000,000 pounds of copper from 1912–19, then suspended operations until 1923. Since then output has increased, until in 1930 the company mined 1,115,000 tons of ore averaging 1.64 percent copper, which yielded 32,612,000 pounds of copper. Mining operations were suspended in 1932. The output of this company is concentrated by the Nevada Consolidated Copper Co. at McGill.

ALASKA

KENNECOTT

The first copper-lode locations in the Kennecott (Nizina) district of Alaska were made in 1899 or 1900; in August 1900 C. L. Warner and Jack Smith discovered the Bonanza ore body, in which veins and irregular masses of chalcocite were exposed at the surface.[35] Development progressed slowly because of the climate and the location (150 miles over a rough trail from tidewater). The Kennecott Mines Co., a Guggenheim subsidiary, obtained the property about 1907. A railway was begun in 1908 and completed in 1911. By that time a large tonnage of very rich ore had been blocked out, a 15,000-foot aerial tramway built to connect the mine with the railroad terminal, and a concentrator erected at the latter point. In 1911 Kennecott's output of copper was more than 20,000,000 pounds.

In 1916 both the Bonanza and its neighboring mine, the Jumbo, were developed to the 700-foot level; in the Jumbo a single ore body had already yielded "50,000 tons of copper ore, much of which ran 76 percent copper." [36] By this time a 700-ton mill was in operation, also an ammonia leaching plant which treated the mill tailings. In 1916 the property yielded its greatest production—300,000 tons of ore averaging 18.7 percent copper and yielding an estimated 100,000,000 pounds of copper. Since then the annual tonnage has declined steadily, and the grade has ranged from 8 to 14 percent. The total yield from 1911 through 1930 was about 690,000,000 pounds of copper. From the start mining has been done largely by the shrinkage method.

The Mother Lode ore body at Kennecott, discovered and located soon after the Bonanza, eventually proved to be on the same ore zone; it was brought into production on a small scale about 1913. In 1918 the Kennecott Mines Co. acquired control of the mine, and thereafter it was worked under the same management as the Bonanza and Jumbo, although keeping its identity as the Mother Lode Coalition Mines Co. From 1918 through 1934 this mine produced roughly 1,160,000 tons of ore and 258,000,000 pounds of copper. The mine was closed in 1933 and 1934.

LATOUCHE

The Beatson mine, on Latouche Island in Prince William Sound, began production in 1904. It yielded a few million pounds annually from the time the Guggenheims acquired it in 1910 to 1915. After 1916, when it was equipped with a flotation plant, the Latouche mine treated 200,000 to 500,000 tons of 1- to 2-percent ore annually, which yielded 10,000,000 to 15,000,000 pounds of copper. From

[35] Moffit, F. H., and Capps, S. R., Geology and Mineral Resources of the Nizina District, Alaska: Geol. Survey Bull. 448, 1911, p. 76.
[36] Moffit, F. H., Mining in the Lower Copper River Basin; Geol. Survey Bull. 662, 1918, pp. 164–175.

1910 through 1930 the mine produced an estimated total of 5,500,000 to 6,000,000 tons of ore and 200,000,000 to 210,000,000 pounds of copper. At first the ore was mined largely by surface glory holes, with a little shrinkage and square-setting, but after 1923 and until the mine was finally closed in 1930 the total tonnage was mined by a modified shrinkage method.

In 1915 the Kennecott Copper Corporation was formed; it acquired the Beatson mine and the property of Kennecott Mines Co.

As the Kennecott Copper Corporation (1933) owns the Braden Copper Co. (Chile) and virtually all of the Utah Copper Co. and the Nevada Consolidated Copper Co., controls the Mother Lode mines as well as railroad and steamship lines, and has recently acquired the Chase Companies, Inc. (an Eastern copper and brass fabricating concern) it is not feasible to estimate the profits derived from the individual Alaskan mines. Those at Kennecott have unquestionably been immensely profitable.

MEXICO

The production of copper in Mexico has been shown graphically in figure 3. Until 1885 this country contributed insignificant amounts, 300 or 400 tons annually, to North America's total output. The development of the Boleo mines in Baja California, about 1885, and the sudden growth of the copper mines at Nacozari and Cananea, Sonora, resulted in a production which for the 5 years, 1903–7, and again in 1911 placed Mexico second only to the United States among the copper-mining countries of the world.

BAJA CALIFORNIA

BOLEO

The Boleo deposit in Baja California is said to have been discovered by a rancher in 1868, after he observed the green flames of his cooking fire built on a hearth of copper-ore float.[37] For a few years after the ensuing rush the deposits were worked by a number of small companies, who shipped lots of 20- to 35-percent smelting ore to Guaymas, Mexico, or to Europe. About 1885 the French house of Rothschild acquired the properties and formed the Compagnie du Boléo to work them on a large scale. The first production by the new company, in 1886, was about 200 metric tons of copper; the next year production was 2,000 tons, and in 1894 it reached 10,000 tons. Since then the company's annual production of copper has been 10,000 to 13,000 metric tons, except in the few years following the World War. At first only ore with a minimum yield of about 5 percent could be mined profitably. In 1900 the newly rebuilt smelter comprised seven water-jacketed blast furnaces, and about 3,300 men were employed by the company.

In 1907–08 the smelter was again reconstructed, 9 larger furnaces replacing 10 old ones; at this time, 2,500 to 3,000 men were engaged in mining and treating about 300,000 tons of 3.6-percent ore annually. In 1913 the company made its peak production, 13,020 metric tons of copper from 374,000 tons of ore. Throughout the years of political upheaval, which seriously disturbed mining in other Mexican States,

[37] Huttl, J. B., The Boleo Enterprise: Eng. and Min. Jour., vol. 132, 1931, pp. 346-348.

Boleo continued to operate. In 1929, 307,000 metric tons of ore yielded 11,705 tons of copper; thus, the grade of ore mined was about what it had been for the last 25 years. The total production of the Boleo mines to the end of 1934 has been about 447,000 metric tons of copper, or about 985,000,000 pounds.

SONORA

NACOZARI

At the beginning of the twentieth century, Boleo lost her position of leadership to two great copper mines in the State of Sonora. The Pilares mine in the Nacozari district became an important producer soon after it was acquired by the Phelps Dodge Corporation from Guggenheim & Co. in 1897. By 1900 a 400-ton mill and two 150-ton furnaces were in operation.[38] The ore was said to contain 8 percent copper. Cut-and-fill stoping has been the chief method of mining throughout the life of the mine. Production figures for the early years are not available, but in 1901 the mine produced over 8,000,000 pounds of copper. In 1904 upon completion of a railroad from Nacozari to Douglas, Ariz., where the Phelps Dodge Co. had just completed its new smelter, the furnaces at Nacozari were abandoned; since then all concentrates and direct-smelting ore have been treated at Douglas. A 1,500-ton mill completed in 1908 tripled the capacity of the mine plant, and the annual production increased to 26,000,000 pounds of copper in 1909. The mill was progressively enlarged, and in 1918 the output increased to a maximum of 42,000,000 pounds. Flotation was used to some extent after 1916 and was the basis for the design of a 3,000-ton mill built in 1920–22. A reconstruction program that started in 1918 involved not only a new mill but the introduction of Diesel-engine power, the installation of a Diesel-driven compressor plant, the sinking of a new service shaft for the mine, and the building of new shops. After 1923 Pilares produced 35,000,000 to 45,000,000 pounds of copper annually, until its shut-down in 1930. The total production of the Phelps Dodge subsidiary, the Moctezuma Copper Co., which owns the mine, to the end of 1930 was about 14,650,000 tons of ore, which yielded about 800,000,000 pounds of copper and possibly 9,000,000 ounces of silver and 25,000 ounces of gold. Ore reserves have remained nearly unchanged at 3,500,000 tons, the grade of which, however, dropped slowly from 4 percent in the early years of the mine to 2.71 percent in January 1929. Dividends of about $10,000,000 were paid from 1902 to 1914. Operations were suspended in September 1931.

CANANEA

The company that has done the most to give Sonora its prominence as a copper producer is the Cananea Consolidated Copper Co., now a subsidiary of the Anaconda Copper Co. The Cananea district is reputed to have been, like the Nacozari, the scene of mining operations for hundreds of years, although reliable details are not known for any operations earlier than about the middle of the nineteenth century. The first copper mining at Cananea of which there is an authentic account was in 1881, when an American company began smelting

[38] Layton, H. B., The Nacozari Mines, Mexico: Eng. and Min. Jour., vol. 69, 1900, pp. 678–679, 707.

copper-silver ores. About the end of the century Col. W. C. Greene organized a Mexican company, obtained control of the prinicipal mines in the district, and started mining operations. The first furnace appears to have been blown in December 1900, and in the fiscal year ended July 31, 1901, 14,000,000 pounds of copper was produced. Mineral Industry for 1901 reported the total cost of producing copper, presumably for the fiscal year 1900–1901, as $0.0446 (United States currency) per pound.

Early in 1902 six furnaces were in blast, and a 600-ton concentrator had been build, and by the end of the year the mines were reported to be producing 1,000 tons of 7-percent ore a day. The production for that year was about 28,000,000 pounds of copper and some gold. In 1904 the mill was remodeled, and roasters and reverberatory furnaces were added to the smelter. Although the grade of ore could not be maintained, the cost of producing copper was reported to be only 8 cents a pound. By 1906, however, expenses could not be met; the company was reorganized and control passed to the Cole-Ryan interests, at that time identified with the Amalgamated Copper Co., the predecessor of the present Anaconda Copper Co. The new management increased the capacity of the mill from 2,500 to 3,500 tons daily, introduced the top-slicing system into the mines, and replaced the old furnaces with eight modern ones. Top-slicing was said to have saved nearly 40 percent in mining cost over the old square-set system. Since 1909 the mines have produced, in normal years, 30,000,000 to 40,000,000 pounds of copper, as much as 1,500,000 ounces of silver, and a few thousand ounces of gold.

Early in 1913, because of the revolution, conditions became so threatening that all Americans left the camp, and operations practically ceased. The concentrator was closed, all development work was stopped, and four of the mines were operated at only part capacity under the direction of Mexican bosses. Operations were gradually resumed, but in 1914 the railroad was again cut, and in 1915 conditions became so bad that the mine was closed virtually all the year; even the Nacozari mines were forced to suspend work after October, and all Americans left for the United States. Production increased slightly in 1916, in spite of continued interruptions in railroad communication. In 1917 the Cananea mines were shut down from June to December, but in the next 3 years production was nearly normal.

The mines were again closed in 1921, owing to the general industrial depression. Since August 1922 the Cananea mines have produced steadily in spite of minor revolutions in 1924 and 1929 which threatened trouble. In 1926 the discovery and rapid development of the famous La Colorada ore body made another peak in the ragged chart of the Cananea Consolidated Copper Co. production. It was estimated [39] unofficially that the ore body contained 375,000 tons of copper above the 1,300 level and that the mine was capable of producing at the rate of 45,000 tons annually, or double Cananea's normal rate. Expansion was checked by the depression of 1930–33, but in 1934 production was the greatest in the history of the camp, amounting to 60,431,430 pounds of copper, 473,720 ounces of silver, and 20,851 ounces of gold.

The following table shows the company output:

[39] Mineral Industry, vol. 36, 1927, p. 155.

Production of Cananea Consolidated Copper Co., 1901–34

Period	Copper, pounds	Silver, ounces	Gold, ounces
1901–28 [1]	1,005,065,009	21,477,872	130,501
1929	58,826,951	402,030	12,366
1930	42,424,773	279,729	7,941
1931	41,872,903	278,231	8,447
1932	36,820,166	259,620	9,596
1933	51,793,290	378,336	13,528
1934	60,431,430	473,720	20,851
	1,297,234,522	23,549,538	203,230

[1] Ore mined, 1901–28, 20,059,229 wet tons.

From 1901 through 1930 the company has paid dividends amounting to about $31,700,000 (United States currency).

PUEBLA

TEZIUTLAN

The ore body of the Teziutlan mine of the Mexican Corporation, S. A., in Puebla, was discovered in 1892.[40] As depth was attained increasing amounts of zinc made the operations unprofitable. The mine was closed for a number of years until the advent of selective flotation. After the necessary changes had been made in the concentrator the mine was reopened in September 1925 and operated until December 1931, when it was again closed owing to the low prices of metals. The production of copper from 1925 to 1930 was 17,900,000 pounds.

CANADA

Copper production in the various Canadian Provinces from 1886 to 1930 was shown in figure 4.

The first recorded attempt at copper mining in Canada, other than by natives who used native copper found on the north shore of Lake Superior and in the Coppermine River district, was brought about in 1767 by the discovery by a trader of lead and copper ores at Mamainse Point, at the east end of Lake Superior.[41] Three years later an English company was formed to work the property. The vein pinched out at a depth of about 30 feet, and the project failed.

In 1845 a company was organized in Montreal to prospect the north shore of Lake Superior for metals. The two large parties sent out by this company made several locations, none of which, however, developed anything of interest. In 1847 the same company purchased the Bruce mine, at the west end of Lake Huron, near Sault Ste. Marie, where ore had been discovered, shortly before, and spent considerable sums in developing and equipping it. Operations were continued unprofitably until 1865, and since then several further attempts have been made to work the mine but with little or no success. It is estimated that since 1846 the mine has produced 400,000 tons of ore averaging 4½ percent copper.

[40] Herivel, E. P., Mining methods and Costs at the Teziutlan Copper Mine of the Mexican Corporation, S. A., Teziutlan, Puebla, Mexico: Inf. Circ. 6736, Bureau of Mines, 1933, 16 pp.

[41] Canada Department of Mines, Report on the Mining and Metallurgical Industries of Canada. 1907–8: 1908, p. 312.

QUEBEC

EASTERN QUEBEC

Copper ore was known to occur in eastern Quebec as early as 1841; but production dates from about 1858, when high-grade ores were first mined at the Harvey Hill and Acton Vale properties.[42] The mines produced a few thousand tons of ore ranging from 12 to 30 percent but were exhausted before 1900. The flurry of excitement caused by these discoveries led to further prospecting and the formation of several mining companies, some of which met with fair success. By 1870 almost all the mines that have since been of importance in eastern Quebec had been opened. Control of the best properties passed chiefly into the hands of two companies—the Eustis Mining Co. and the Nichols Chemical Co. The manufacture of sulphuric acid from the high pyritic ores of the district, which began in 1877, boosted the declining production and maintained it at a fair level until about 1920 when competition with natural sulphur deposits became too severe and a shut-down ensued. Recently activity has been renewed at the Eustis mine.[43] The total production of the district from 1886, the earliest year for which production figures are available, through 1921 was about 103,000,000 pounds of copper (based on the estimated recovery of copper from ore shipped). Young[44] estimated the total production to about 1925 at 125,000,000 pounds.

ROUYN

In the last decade two new copper districts of great importance have come to the fore—one in northwestern Quebec and the other in northwestern Manitoba. The Rouyn district of Quebec was the scene of a rush in the fall of 1922 as a result of a gold strike, although the Horne deposit had been discovered and staked 2 years previously. It soon became evident that the copper-mining possibilities of the district outweighed the gold prospects when the Noranda Mines, Ltd., which was drilling the Horne mine, cut 130 feet of solid sulphide ore averaging $4.36 in gold to the ton and 8.23 percent copper.

Early in 1924 nearly 400,000 tons of $20 ore had been developed by trenching and drilling. Railroad construction started late in 1925, and the first train came into Rouyn on October 1, 1926. Before then, $40,000,000 worth of ore had been developed above the 300-foot level, a power contract arranged, and the company capitalization increased sufficiently to start smelter construction. In December 1927 the first copper was poured, and a 500-ton concentrator was being built. A second railroad was brought into the district. In 1929 the smelter capacity was stepped up to 2,000 tons and the concentrator enlarged; in that year production was over 50,000,000 pounds of copper, and a first dividend was paid.

The following table shows the production from 1927 through 1934:

[42] Douglas, James, Early Copper Mining in the Province of Quebec: Jour. Canadian Min. Inst., vol. 13, 1910, pp. 254-272.
Young, G. A., Geology and Economic Minerals of Canada: Canada Dept. Mines, Econ. Geol., ser. no. 1, 1926, pp. 112-116; appendix, pp. 16-17.
[43] Goodwin, W. M., Canada's Oldest Copper Mine: Canadian Min. Jour., vol. 52, 1931, pp. 571-576.
[44] Young, G. A., work cited, Appendix, p. 17.

Smelter production, Noranda Mines, Ltd.[1]

Year	Ore, etc., tons	Copper, pounds	Gold, ounces	Silver, ounces
1927	10, 740	552, 345	767	2, 644
1928	271, 926	33, 065, 261	52, 949	186, 277
1929	428, 221	51, 223, 115	68, 732	334, 279
1930	734, 072	73, 509, 373	117, 393	691, 920
1931	765, 544	62, 859, 355	253, 363	558, 801
1932	918, 567	63, 013, 485	341, 350	619, 597
1933	1, 010, 629	65, 008, 731	284, 675	510, 739
1934	1, 050, 684	70, 175, 512	248, 615	552, 809
Total	5, 190, 383	419, 407, 177	1, 367, 844	3, 457, 066

[1] Quebec Bureau of Mines Report for Year 1934; Pt. A, 1935.

Ore reserves at the end of 1934 were as follows:

Ore reserves, Noranda Mines, Ltd.

December 1934	Tons	Copper, percent	Gold, ounces
Sulphide, over 4 percent	6, 826, 000	7. 25	0. 166
Sulphide, under 4 percent	20, 497, 000	1. 04	. 191
Siliceous flux	982, 000	. 15	. 142
Total	28, 305, 000		

This unique mine was able to continue operations profitably from 1930–34 because some ores could be mined of a much higher gold content than the average of its reserves.

ONTARIO

SUDBURY

Ontario was the scene of the second important event in Canadian copper mining, when in 1883 or 1884 railroad building through this Province not only opened the country to prospectors but also uncovered an outcrop of copper ore in the Sudbury district.[45] In a few years all the deposits of later importance had been found, and the district was well established. The first production was made in 1886, when the Canadian Copper Co. exported sorted ore to the United States; this led to the realization that the ore contained considerable nickel, and before long the niter-cake process was devised to separate the nickel from the copper. The first furnace for a smelting plant was blown late in 1888 at Copper Cliff near Sudbury; in a few years it was supplemented by a converter plant and in 1899 was replaced by a new smelter. In 1902 the International Nickel Co. was organized as a holding company for the Canadian Copper Co. and for several other firms engaged in the nickel business. Shortly before, the Mond Nickel Co., owner of the Mond nickel-refining process, had purchased the Victoria mine at Sudbury, laid out roast yards, and built a smelter. Production was started in 1901.

These two companies were the sole producers in the district for many years. In 1904 the International Nickel Co. completed an entirely new smelter and in 1908, a new bessemer plant. In 1913 the Mond Nickel Co., having acquired several large mines, likewise built a new smelter. Both companies developed hydroelectric power on nearby rivers. In 1917 the International Nickel Co. completed a refinery at Port Colborne, while the Mond Nickel Co. continued to

[45] Report of Royal Ontario Nickel Commission, Toronto, 1917, pp. 20-56.

ship its matte to Wales for refining. In 1915 the Mond company built the first sulphuric-acid plant in the district.

In 1925 diamond drilling disclosed very rich ore at depth in the Frood mine of the International Nickel Co. The underground development of this tremendous ore body was the first step in a program of expansion that involved a new surface plant for the Frood and other mines, larger power plants, a new crushing and screening plant at one mine, an 8,000-ton concentrator, a new smelter, additions to the Port Colborne refinery, and (by allied interests) the construction of a refinery and acid plant at Sudbury.

Early in 1929 the two companies were merged, leaving the International Nickel Co. with a near monopoly of the world nickel industry. In 1929 it sold 63,000 tons of the 68,000 tons of nickel on the world market. In 1930 the company mined more than 2,000,000 tons of ore, milled 1,500,000 tons, and produced 164,000 tons of matte containing about 60,000 tons of nickel and 70,000 tons of copper. The treatment of refinery residues yielded 23,000 ounces of gold, 1,070,000 ounces of silver, and 68,000 ounces of platinum. The company employed over 5,000 men at Sudbury and over 1,200 at Port Colborne in 1930.

The four principal mines of the company were reported to have reserves at the end of 1930 of more than 206,000,000 tons of 2-percent copper ore, including some 40,000,000 tons in the Frood mine below the 1,400-foot level that averaged 3.6 percent copper and 2.4 percent nickel.[46] The smelting plant is estimated to have an annual capacity of 140,000,000 pounds of nickel and 200,000,000 to 240,000,000 pounds of copper.[47] The company paid $16,000,000 in dividends in 1930, which brought its total disbursements to about $136,000,000.

A new company, the Falconbridge Nickel Mines, Ltd., blew in a 325-ton smelter in the Sudbury district in February 1930 and shipped its matte to Norway for refining.

The total production of copper in the Sudbury district from 1886 through 1930 (which is equivalent to the total production of the International Nickel Co. and its predecessors) has been about 990,000,000 pounds.[48]

The following table shows the production of copper in the district from 1929 through 1934.

Production of copper, Sudbury district, 1929–34

	Sudbury nickel-copper district	Falconbridge Nickel Mines, Ltd.	International Nickel Co.	
	Copper, pounds	Copper, pounds	Copper, pounds	Ore, tons
1929	88, 880, 000	0	88, 880, 000	1, 991, 910
1930	126, 640, 000	1, 311, 940	125, 328, 000	2, 041, 701
1931	112, 812, 000	2, 067, 000	110, 740, 000	1, 580, 355
1932	77, 014, 000	2, 393, 260	74, 620, 000	666, 468
1933	145, 442, 000	3, 940, 094	141, 500, 000	1, 336, 040
1934	205, 036, 000	4, 626, 535	200, 410, 000	2, 690, 814
1931–34	540, 300, 000	13, 030, 000	527, 270, 000	6, 273, 000

[46] Engineering and Mining Journal, vol. 129, no. 5, 1930, p. 268.
[47] Ontario Department of Mines, 40th Ann. Rept., pt. 1, 1931, p. 20.
[48] Does not include production from the Falconbridge Nickel Mines, Ltd.

The production of ore from the four producing mines of the International Nickel Co. from 1929 through 1934 is shown in the following table:

Production of ore, International Nickel Co., tons

	Frood	Creighton	Garson	Levack
1929	199, 852	1, 177, 323	246, 049	368, 686
1930	902, 531	861, 770	277, 500	0
1931	1, 068, 978	301, 394	209, 983	0
1932	513, 590	96, 850	56, 028	0
1933	952, 725	383, 315	0	0
1934	1, 868, 186	822, 628	0	0

BRITISH COLUMBIA

SOUTHERN BRITISH COLUMBIA DISTRICTS

Shortly after copper mining was begun in the Sudbury district copper was discovered in southern British Columbia as a direct consequence of railroad building. In 1887, 2 years after the completion of the Canadian Pacific R. R., rich silver-lead ores were shipped from the Slocan district to United States smelters.[49] Attracted perhaps by this bait, prospectors swarmed into the district and within 2 or 3 years had made many important discoveries throughout the south-central part of the Province. The first recorded production of copper, in 1889, comprised 100 tons of ore from the Hall mine near Nelson, which assayed 220 to 574 ounces of silver to the ton and 17 to 43 percent copper. All of the more important mines in the Rossland district were located in 1890, and in 1891 the first small shipment of ore, carrying 4 ounces of gold and 3 ounces of silver to the ton and 5 percent copper, was made. Regular production from this district was begun in 1894. In 1891 the low-grade copper deposits of the Boundary district were discovered, but because rail transportation was lacking they were not developed on a large scale for about 10 years, or until the Columbia & Western R. R. was built

Two smelting plants were built in 1895. The first was the Hall Mines smelter near Nelson; this had one 160-ton blast furnace, and another was added later in the year with a daily capacity of more than 275 tons of charge. This furnace was said to have been the largest in the world at the time The available copper ore was soon exhausted, and changes were made in the smelter to adapt it to the custom smelting of lead-silver ores, which was carried on until 1907; about 200,000 tons of ore, which yielded about 4,000,000 ounces of silver and 14,000,000 pounds of copper, was treated in the plant.

The other of these plants, the Trail smelter, a custom plant owned by F. Augustus Heinze, was started in February 1896. It contained four 40-ton reverberatories, one 50-ton blast furnace, and a 200-ton water-jacketed furnace. The first ores treated came from the Rossland mines. The reverberatories proved unsuitable for the work and were soon replaced by blast furnaces. In 1898 the plant was sold to the Canadian Pacific R. R., which in 1906 transferred the property to the newly organized Consolidated Mining & Smelting Co. of Canada, Ltd. The plant has grown steadily since then. Its principal business has been the smelting and refining of lead-zinc ores from the company's Sullivan mine, but its copper section, which

[49] Canada Department of Mines, Report on the Mining and Metallurgical Industries of Canada: 1908, p. 82.

comprises three blast furnaces and two converters, with accessory equipment, and an electrolytic copper refinery having a capacity of 60 tons a day, has produced about 170,000,000 pounds of copper from the beginning of operations through 1930.

In the next few years (after 1896) five or six more smelters were built in southern British Columbia, none of which was in operation in 1932, but two which served the Boundary district until after the World War made this the most important copper district of British Columbia for many years.

Late in 1918 the Canada Copper Corporation, one of the principal operators in the Boundary district, closed its smelter at Greenwood, because ore reserves in its Mother Lode mine were exhausted, and began extensive development of a large low-grade ore body at Copper Mountain in the Similkameen district. A mill was built and about $2,500,000 spent on the property; then control passed to the Granby Consolidated Mining, Smelting & Power Co. in 1923. Reconstruction of the mill followed, and production was begun in 1925. The output during the next 4 years ranged from 18,000,000 to 23,000,000 pounds of copper per year. Because of adverse market conditions the mine was closed in December 1930.

British Columbia was the leading copper producer in Canada from about 1900 (when the Boundary district began large-scale production) to 1930, when Ontario took the lead owing to the International Nickel Co. development program at Sudbury.

HOWE SOUND

In 1898 claims, now a part of the Britannia mine, were staked on the east shore of Howe Sound, 30 miles north of Vancouver.[50] About 500,000 pounds of copper were produced in 1905, 4,500,000 pounds in 1906, and 1,700,000 pounds in 1907. In 1908 the present Britannia Mining & Smelting Co. was organized, and in 1910 production was resumed. Thereafter production increased rapidly, and successive additions were made to the mill. This necessitated the construction of extensive waterpower works, and eventually power was brought also from Vancouver. In 1916–20 the plant worked to capacity, producing 16,000,000 to 18,000,000 pounds of copper annually. In 1921 the mill burned, and a flood wiped out the lower townsite, causing great loss of life. A new mill, of 2,500 to 3,000 tons daily capacity, was erected. In 1930 the mill treated 2,150,000 tons of ore and produced 44,300,000 pounds of copper, 203,000 ounces of silver, and 13,062 ounces of gold. The total production of the mine through 1934 was about 477,000,000 pounds of copper, 2,400,000 ounces of silver, and 144,000 ounces of gold from 19,000,000 tons of ore. The mine operated at one-fourth capacity during 1935.

HIDDEN CREEK

In 1901 claims were staked on a large outcrop of massive sulphide ore near Hidden Creek on the shore of Observatory Inlet, British Columbia; in 1910 these claims were acquired by the Granby Consolidated Mining, Smelting & Power Co., then operating the Phoenix mines in the Boundary district.[51] The claims were developed by tunneling;

[50] Brewer, W. M., Britannia Mining & Smelting Co.: British Columbia Minister of Mines, Ann. Rept., 1924, p. 229.

[51] Campbell, D. G., The Hidden Creek Mine and Smeltery: Eng. and Min. Jour., vol. 103, 1917, pp. 274–276.

several large ore bodies were developed, and by 1913 a 2,000-ton smelter was being built at Anyox. Production in 1914, the first year, was nearly 16,000,000 pounds of copper from about 475,000 tons of ore.

In 1923 the mine was producing about 3,000 tons daily, and a flotation concentrator was started. In 1927 the addition to a sintering plant permitted local smelting of all concentrates, part of which had previously been shipped to Tacoma. Since then virtually the entire output of the mine (5,500 tons daily) has been milled locally. In 1924–29 production ranged from 35,000,000 to 40,000,000 pounds of copper annually. During the first few years of production the ore was derived largely from glory-hole mining; in 1932, however, about 90 percent of the output was won by open-stope methods. In 1930 development of deep levels reached 960 feet below sea level.

The total production of the mine (including precious metals in fluxes and 1 year's output of a neighboring property, the Bonanza mine) from 1914 through 1931 was about 18,100,000 tons of ore, which yielded 541,000,000 pounds of copper, 5,683,000 ounces of silver, and 104,000 ounces of gold. The Anyox property was sold to the Consolidated Mining & Milling Co. of Canada in November 1935.

MANITOBA

THE PAS DISTRICT

In 1915 a number of claims were staked in Manitoba, near Schist Lake; from one of these, the Mandy mine, about 9,000,000 pounds of copper was produced during the period of high prices from 1917 to 1920.

The large, low-grade, Flin Flon copper-zinc deposit on Flin Flon Lake was also discovered in 1915. The original locators did some trenching, and other interests in the next 3 years completed more than 25,000 feet of diamond drilling. The Mining Corporation of Canada bought control in 1921 and continued work until about 16,000,000 tons of ore had been developed above the 900-foot level. In 1927 the Hudson Bay Mining & Smelting Co. was formed. In 1928–30 an 85-mile railroad was built to the mine, a 44,000-horsepower hydroelectric plant was built on the Churchill River (65 miles to the north), and a 3,000-ton concentrator and smelter were erected. Both open-pit and underground mining were started. The cost of the mining and metallurgical plant was stated to have been $13,500,000 [52] and that of the hydroelectric power plant and transmission line, $7,100,000. From 1,477,000 tons of ore mined during 1934, 37,486,000 pounds of copper, 1,335,000 ounces of silver, and 99,000 ounces of gold were recovered. Ore reserves, estimated in 1928 as 18,000,000 tons averaging 1.71 percent copper, 3.45 percent of zinc, and 0.074 ounce of gold and 1.06 ounces of silver to the ton, were raised later to 20,000,000 tons by further development.

Another Manitoba copper deposit, on Kississing (Cold) Lake 30 miles east of the Flin Flon, was discovered in 1922, and after several examinations and considerable drilling by various parties it became the property of Sherritt-Gordon Mines, Ltd., in 1927. Preliminary drilling, trenching, and geologic mapping were followed by building an 80-mile road and bringing in equipment and supplies for an in-

[52] Hudson Bay Mining & Smelting Co., 3d Ann. Rept., March 1931.

tensive campaign of diamond drilling and underground development, which was started in January 1928. In about a year 15,000 feet of underground work and 27,000 feet of drilling had outlined 4,000,000 tons of ore which averaged 2.5 percent copper and 3 to 6 percent zinc, besides considerable material of lower grade. Arrangements for a railroad were made, and power and smelting agreements entered into with the Hudson Bay Mining & Smelting Co. Plant construction, including an 1,800-ton concentrator, was started in 1929. A working shaft was sunk, and preparations were made to mine by a sublevel open-stope method. Milling was begun in March 1931.

The Flin Flon property alone produced about 2,000,000 pounds of copper and 3,800,000 pounds of zinc in 1930. The Flin Flon and Sherritt-Gordon together produced about 46,000,000 pounds of copper in 1931; the Flin Flon contributed the larger part of this and 35,000,000 pounds of zinc. In 1934 Flin Flon mined 1,477,000 tons of ore and produced 37,486,000 pounds of copper, 99,000 ounces of gold, and 1,335,000 ounces of silver.

PART 2. GEOLOGY OF COPPER DEPOSITS OF NORTH AMERICA [1]

By B. S. Butler

INTRODUCTION

A brief summary of the geological occurrence of the copper deposits of North America is presented in the following pages. Those interested in a more comprehensive discussion are referred to the recently published Copper Resources of the World.[2] No attempt is made to give complete bibliographies, but the more comprehensive and later reports on the different districts and subjects are cited.

The copper deposits of North America are widely distributed geographically and were deposited abundantly in at least five geologic periods extending from early pre-Cambrian to Tertiary. The deposits of the various periods and localities show differences in structural characteristics, in their relation to associated intrusive bodies, in mineralogy, and in their reaction to oxidizing agencies. Recognition of these differences should be helpful in prospecting for, development of, and mining of copper deposits.

GEOLOGIC CLASSIFICATION OF COPPER DEPOSITS

Mineral deposits may be classified on several bases and from the standpoint of prospecting, development, and mining; each has its use. Classifications that will be considered here are: (1) Age of formation, (2) relation to structure, (3) relation to intrusive bodies, (4) mineralogy and occurrence, and (5) changes due to oxidation.

AGE OF FORMATION

Geologic processes work in cycles that are repeated over and over again in the life of a continent. A cycle may be thought of as starting with erosion of rock from an elevated area and deposition of the material in an adjacent basin. The basin slowly settles under the steadily increasing load of sediment, and the eroded area rises as it is lightened by removal of material. In the deep zone of flowage, material moves from the sinking to the rising area. Finally the basin, which has been weakened by settling into the zone of flowage, collapses with folding and faulting of the sedimentary rocks and intrusions of igneous material into the folded and faulted area, accompanied usually by extrusion. Most of the copper deposits were formed by material escaping from the crystallizing magma soon after the igneous intrusion.[3]

[1] Published by permission of the Director, Geological Survey.
[2] 16th International Geological Congress, vol. 1, Washington, D. C., 1935.
[3] Butler, B. S., Ore Deposits of the United States in Their Relation to Geologic Cycles: Econ. Geol., vol. 28, 1933, pp. 301–328.

PRE-CAMBRIAN

Geologic cycles are recognized to have existed in North America since early pre-Cambrian time. It is impossible definitely to group all of the pre-Cambrian deposits, but Bruce [4] has recognized two pre-Cambrian periods of copper deposition in the Canadian Shield area, namely, an early pre-Cambrian period, during which the copper-zinc districts of Manitoba and the copper-gold district of Quebec were formed, and a late pre-Cambrian period, during which the copper-nickel ores of Ontario and the native copper ores of the Lake Superior region were deposited. The deposits of the Jerome district, Arizona, belong to the pre-Cambrian period, possibly to the early pre-Cambrian. Pre-Cambrian deposits have yielded a large part of North American copper ores.

PALEOZOIC

The intrusive period of the Appalachian cycle probably culminated in the Permian for the southern Appalachians and earlier for the northern Appalachians. The copper deposits from Quebec to Georgia, which have been most productive at Ducktown, Tenn., may be regarded as of this cycle and as of middle to late Paleozoic age.

Small deposits associated with basic intrusives on the Pacific coast probably also belong to the Paleozoic age. This period has produced only a small part of the copper of North America.

Small copper deposits are associated with basic intrusions of Triassic age from New Jersey to Virginia and may represent a final stage of the Appalachian revolution.

MESOZOIC

The next great period of copper deposition followed the intrusion of the Sierra Nevada and Coast Range batholiths along the western coast from Alaska to Mexico. Intrusion belonging to this cycle began in the Paleozoic age and continued through early Mesozoic; doubtless some ore deposits were formed before the culmination, which is generally regarded as Late Jurassic and early Cretaceous.

Assigned to this period are the main deposits of California and western Nevada and probably most of the deposits of British Columbia. The deposits of Copper River and Prince William Sound, Alaska, may be assigned to the Mesozoic, although perhaps they are earlier than the Coast Range batholiths.

Deposits of this period though large have yielded much less copper than those of the next period.

CENEZOIC

The period of igneous activity in the Cordilleran region, which extended from Late Cretaceous through the Tertiary, was the most productive period for copper deposits. It may be regarded as a continuation of the previous Mesozoic period, although on the whole igneous activity was farther east in the Rocky Mountain area. To the north in British Columbia it seems to overlap on the Coast Range region. Igneous activity was intermittent, and accompanying

[4] Bruce, E. L., Mineral Deposits of the Canadian Shield: Macmillan Co., Toronto, 1933, p. 413.

mineralization can be classified into subperiods, but most of the copper mineralization seems to fall in the early part.

Probably some of the deposits of British Columbia were formed in this period, as were the deposits of Montana, Utah, eastern Nevada, and Arizona (except the Jerome area), New Mexico, and northern Mexico. The districts of this period that border the Colorado plateau form one of the great metalliferous provinces of the world. Deposits of this period have yielded most of the copper of North America in the past and have the largest known reserves.

RELATION TO STRUCTURE

The close relation between copper deposits and igneous bodies, discussed later, indicates that the major structural control of the ore deposits is the same as that which governed the emplacement of igneous bodies.

The major periods of igneous intrusion mark the culminations of geologic cycles whose earlier history was erosion in one area and sedimentation in an adjacent area. The areas of erosion and basins of sedimentation were ordinarily much elongated and roughly paralleled the outlines of the continent at the time. Most of the eroded material came from one side of the basin of deposition and the greatest thickness was deposited near that side. The strong supporting rocks underlying the basin were thinned most in this area of greatest sedimentation and sinking. Collapse came along the zone of thinning, with folding of the sedimentary rocks and formation of reverse and overthrust faults that dip toward the areas of erosion and away from the basin of sedimentation. Along these faults the area of sedimentation was carried beneath that of erosion or that of erosion over that of sedimentation With folding and faulting the area was raised, and igneous material was forced into the zone of folding and thrust faulting, guided in its movement largely by the faults. Some of the material reached the surface as flows, while much of it solidified as various kinds of intrusive bodies with some of which mineral deposits are associated.

PRE-CAMBRIAN

The outlines of the early pre-Cambrian basins of North America are not well known, but such basins existed in early pre-Cambrian time which ended by mountain building and intrusion of igneous bodies with which the early pre-Cambrian copper deposits are associated. In late pre-Cambrian time an east-west basin of deposition existed in the Lake Superior region which was folded at the close of the pre-Cambrian period with accompanying intrusion, forming the Penokan Range.[5] The deposits of native copper of the Lake Superior region and the nickel-copper deposits of Sudbury were formed along this east-west line.

PALEOZOIC—APPALACHIAN REGION

The Paleozoic basin of sedimentation paralleled the eastern coast of the continent from Quebec to Alabama, with a land mass to the east. A great thickness of sediments accumulated in the eastern part of the basin near the shore. The collapse of this basin, with accompanying

[5] Cooke, H. C., Studies of the Physiography of the Canadian Shield; the Pre-Pliocene Physiographies as Inferred from the Geologic Record: Trans. Royal Soc. Canada, 3d ser., vol. 25, 1931, p. 138.

folding, faulting, and intrusion, marked the close of the Paleozoic period for the southern Appalachians, although it was perhaps considerably earlier to the north. Thrust faults of easterly dip are prominent, and the eastern margin of the Paleozoic rocks has been overridden by the pre-Cambrian rocks. Intrusion occurred along the breaks near the boundary of the old land mass and the sedimentary basin. These intrusive bodies have been exposed by deep erosion. The lenses carrying iron, zinc, and copper sulphide that have furnished most of the copper from the Appalachian region are associated with the intrusive bodies in this belt (fig. 6),[6] although the association is less close than that for the Tertiary deposits.

MESOZOIC—PACIFIC REGION

From late pre-Cambrian through the Paleozoic and early Mesozoic, sedimentation continued in a series of basins in what is now the Cordilleran region. Folding and intrusion and extrusion of igneous material began in the Paleozoic, increased through the early Mesozoic (especially toward the north), and culminated in Late Jurassic and early Cretaceous with the intrusion to the north of the great Coast Range batholith extending through British Columbia from Alaska to the United States. At the International Boundary, it swings east for 300 miles along the north side of the Columbia Plateau and may continue south as the Idaho batholith along the east side of the plateau. To the south the Sierra Nevada batholith was intruded, extending from Oregon to Lower California, Mexico. Most of the copper deposits of British Columbia, Alaska, California, and western Nevada were formed at this time and were associated with these intrusives.

CENEZOIC

In Mesozoic time the area of maximum sedimentation moved eastward to the Rocky Mountain region. The western border of the Mesozoic basin passed through western Wyoming and eastern Idaho and thence through western Montana into British Columbia. This line also roughly marked the eastern boundary of the Paleozoic basin.

In late Cretaceous and Tertiary time, folding and thrust faulting took place along a great arc, extending southeastward from western Montana to central Utah and thence westward into Nevada and California (fig. 6), which roughly followed the western boundary of the Mesozoic basin.

Another major structure zone extends along the east side of the Colorado Plateau through central New Mexico into Colorado, where it swings westward to join the east-west Uinta structure, the westward part of which crosses the zone along the west side of the plateau. The Rocky Mountain Front Range extends northwesterly from Colorado through Wyoming and joins the western belt in northwestern Wyoming and southwestern Montana.

On the southwest border of the Colorado Plateau is a northwesterly trending belt of folding, faulting, and intrusion, termed the "mountain belt" by Ransome, which was formed at essentially the same time as those that border other parts of the plateau. In southeastern Arizona

[6] After King, Philip, 16th Internat. Geol. Cong. Guidebook 28, 1932, pl. 1., p. 58.

and southwestern New Mexico this belt turns southward into the Sierra Madre Oriental of Mexico.

These structures are brought out clearly on the structure map of the United States by Philip King, which is reproduced as figure 6 with the copper districts added. Billingsley and Locke have pointed out that junctions or crossings of structural zones (geologic cross-roads) are nodes of intrusion and consequently of mineralization.

The Boulder batholith of Montana, with its associated mineral deposits, is located at the junction of the structure zones along the east and west sides of the Colorado Plateau and the Wyoming Basin. The intersection of the west-side structure zone with the Uinta Mountain structure is the locus of the largest mineralized region of Utah (one of the greatest in the west), the Park City-Bingham-Tintic area. The location of some of the other copper districts by intersections is less obvious and therefore less certain but perhaps not less real. Near the junction of the structure belts along the east and south sides of the Colorado Plateau and their extension into Mexico are the great copper districts of Arizona, New Mexico, and Sonora, Mexico. Billingsley and Locke have pointed out structural crossings that may have located individual districts. For the west-side belt it may be noted that the mineralized districts are all in or above the most easterly of the west-dipping thrusts; that is, for this belt the intrusions and accompanying mineralization appears to be in the hanging wall of the lowest thrust.

SUMMARY

The copper districts are located along structure lines that have resulted from folding and faulting, with accompanying intrusions, in basins of sedimentation. The districts are therefore grouped along or around areas of erosion, or positive areas, near the margin of the Canadian Shield and the Piedmont Plateau, along the western margin of the continent, and on the margins of the Columbia and Colorado Plateaus.

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RELATION OF COPPER DEPOSITS TO IGNEOUS ACTIVITY

Copper deposits are characteristically associated both in time and place of formation with igneous activity, as has just been indicated. With the Tertiary deposits and less notably with earlier deposits, both extrusive and intrusive rocks are present; this indicates that the igneous forces broke through to the surface. In most copper districts the igneous rocks are of intermediate composition (diorites and monzonites), with the quartzose and porphyritic phases in the intrusive bodies and corresponding compositions in the extrusive rocks.

It has long been recognized that most copper deposits are associated in origin with intrusive bodies and that the deposits tend to group around relatively small stocks or cupolas[7] that are upward projections of much larger batholithic masses. There is, however, a notable difference in the relation of different types of deposits to the stocks. The Tertiary copper deposits of the eastern Cordilleran region are in and closely grouped around the tops of stocks. Much of the copper occurs in the stocks, and most of it is within a few thousand feet of the stocks. The deposits of pre-Cambrian, Paleozoic, and part of those of Mesozoic age are not so closely associated with exposed stocks. In fact, many of them are several miles from known intrusive bodies, and relatively little copper occurs in stocks. This difference seems to be due to the distance below the surface at which the igneous bodies were intruded and the copper deposits formed.

Many of the Tertiary stocks were intruded near the surface, as they cut lavas of the same general age which were at most a few thousand feet thick. Solutions moving to the top of and outward from such bodies could not travel far before the change in heat and pressure would cause precipitation of copper minerals in or only a few thousand feet from the stock.

Solutions from bodies that were far below the surface could travel farther from the source before loss of heat and decrease of pressure would cause precipitation of copper minerals, therefore the deposits may be many thousands of feet from their igneous source.

SUMMARY

The Tertiary deposits of the Rocky Mountain region are in and closely grouped around the tops of stocks. The deposits of the early pre-Cambrian and Paleozoic periods are rarely in stocks and are characteristically thousands of feet or may even be miles from the probable parent stock. The Mesozoic deposits are intermediate. This difference is believed to be due to the depth below the surface at which the deposits were formed.

CLASSIFICATION ACCORDING TO MINERALOGY AND OCCURRENCE

Copper occurs in many mineral combinations both in hypogene and supergene deposits. For hypogene deposits associated with igneous rocks, in which copper is the chief metal of economic value there is not a very wide range in temperature of deposition, but variations in physical and chemical conditions give corresponding variations in types of deposits. This does not imply that copper does not deposit over a considerable range of temperatures, because copper combined with iron as chalcopyrite or cubanite apparently deposits at a higher temperature than that combined with antimony as tetrahedrite or with sulphur as simple copper sulphide, chalcocite. In general, copper-iron sulphide deposits later than iron sulphides, and copper sulphide deposits still later, as do sulphantimonides and sulpharsenides of copper.

[7] Butler, B. S., Relation of Ore Deposits to Different Types of Intrusive Bodies in Utah: Econ. Geol., vol. 16, 1915, pp. 101–122.
Emmons, W. H., On the Mechanism of the Deposition of Certain Metalliferous Lode Systems Associated with Granitic Batholith. Ore deposits of the Western States: Am. Inst. Min. and Met. Eng., 1933, pp. 327–349; Relation of Disseminated Copper Deposits in Porphyry to Igneous Intrusions: Trans. Am. Inst. Min. and Met. Eng., vol. 75, 1927, pp. 797–808.

In the following general classification of types of copper deposits, it should be recognized that gradations will occur with gradual changes from one physical and chemical environment to another: (1) Lenticular replacements of schistose rocks; (2) Sudbury nickel-copper type; (3) native copper types; (4) replacement deposits in sedimentary rocks; and (5) vein, pipe, and disseminated deposits.

LENTICULAR REPLACEMENTS OF SCHISTOSE ROCKS

Deposits of this type include those whose primary mineralization consists mainly of sulphides of iron, copper, and zinc, replacing schistose rocks. Schistosity may be the result of widespread metamorphism, but the deposits are along zones of shearing. Such deposits seem to have formed at some depth under conditions that favored deformation by shearing rather than by brecciation or shattering. The solutions that deposited the ores were closely confined to the channels and replaced the sheared rock rather completely, forming massive sulphide lenses. The minerals are those of the upper range of sulphide deposition, and the deposits may or may not contain precious metals in important amount. Such deposits usually are not closely associated with igneous stocks, and few are in stocks.

At depth, solutions traveled considerable distances from their source through siliceous rocks before changes in temperature or pressure caused deposition of sulphides.

It is perhaps to be expected that the early pre-Cambrian deposits of the deeply eroded Canadian Shield area should be of this type. The later pre-Cambrian deposits of the Canadian Shield seem to be of shallower type.

The Paleozoic deposits of the deeply eroded Appalachian region also belong to this classification, as do some of the deposits of the Sierra Nevada and Coast Range batholiths of California and British Columbia where erosion has also been deep. Some West-coast deposits, however, seem to have formed at shallower depth. The Tertiary deposits of the Rocky Mountain area, which have relatively shallow erosion, are not of this type.

The primary ores are generally a rather low grade massive sulphide. Pyrrhotite is present in many of the ores and iron-bearing sphalerite in most of them. These minerals are notably absent in most of the Tertiary copper deposits, and depth of formation seems to be a factor in their formation. Possibly, the volatile zinc, under light pressure, moved away from the copper before conditions were favorable for precipitation of sulphides, while under high pressure it was held back until conditions for sulphide deposition were reached. Why pyrrhotite should favor this zone is not so obvious.

Supergene alteration in many deposits has produced a zone of enriched sulphides, usually thin and rich as at Ducktown, Tenn., and Iron Mountain, Calif., but sometimes deep and rich as at Jerome, Ariz.

Deposits of this type have yielded much copper, zinc, and iron sulphide for sulphuric acid manufacture.

SUDBURY NICKEL-COPPER DEPOSITS

The Sudbury nickel-copper deposits have many mineralogical similarities to deposits of the schist-replacement type, as has been pointed out by Ross,[8] and might be included as a subclass. Fracturing was not characteristically by shear, which suggests deposition at a shallower depth.

The nickel deposits also have physical resemblances to pipe deposits considered later, especially to the replacement breccia type. If, however, the Sudbury deposits were formed by magmatic differentiation rather than by replacement, they would belong in a separate group.

NATIVE COPPER DEPOSITS

Native copper deposits include those in which the copper occurs in shoots in tops of certain lava flows, in conglomerates interbedded with flows, and in sandstones. The rocks in which the copper occurs are characteristically red due to the presence of ferric oxide. It is thought that the oxidizing effect of the ferric oxide has resulted in the copper being deposited as native metal rather than as sulphide.

Deposits of this type have been mined nearly 10,000 feet down the dip and at vertical depths of more than 6,000 feet in the veins and lodes with no notable change in mineralogy, which suggests that saturation of the solutions with copper was brought about slowly and continued over a great range in depth. The solutions traveled far from their source before depositing their metal load and are not close to intrusive bodies. Fractures of this type do not suggest conditions favorable to shear. The ores are generally low-grade concentrating ores, although large masses of copper are present.

The Lake Superior district of Michigan is the only large producing region having deposits of this type, although similar deposits occur in New Jersey, Pennsylvania, Maryland, Arizona, Alaska, and northern Canada. The Michigan district is the second largest copper district on the continent.

REPLACEMENT DEPOSITS IN SEDIMENTARY ROCKS

Replacement deposits in sedimentary rocks include what are known as contact or pyrometasomatic deposits and those formed by the replacement of sedimentary rocks, commonly limestone, outward from fissures. The two classes may be separated, although no sharp line can be drawn between them. The typical contact or pyrometasomatic deposits are replacements of limestone or dolomite by silicates, oxides, and sulphides. Many contact deposits contain large bodies of andradite (the ferric iron garnet), with which are associated other silicates, the iron oxides (magnetite and hematite), and sulphides of iron and copper. Many such deposits are in limestone at or near the contact of an intrusive body, usually a stock.

The silicates and oxides formed first and the sulphides later. The sulphides, including copper sulphide, are most abundant between the silicate zone and the unreplaced limestone.[9] Away from the igneous contact garnet gives place to pyroxenes and amphiboles, and sulphides may become abundant. In some deposits the garnet zone is largely lacking, and the sulphide zone is close to the igneous contact. Contact deposits grade into fissure-replacement deposits, in which certain beds of limestone are replaced by sulphides outward from fissures with or

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[8] Ross, C. S., Origin of the Copper Deposits of the Ducktown Type in the Southern Appalachians: Geol. Survey Prof. Paper 179, 1935, p. 112.

[9] Umpleby, J. B., The Occurrence of Ore on the Limestone Side of Garnet Zones: Univ. of California Dept. of Geol. Bull., vol. 10, 1916, pp. 25–37.

without the formation of silicates. In the massive garnet zones fissure control is not obvious; outward it becomes increasingly apparent.

Deposits of the massive garnet type are widely distributed, especially in the eastern Cordilleran region; and they have been much prospected but have yielded relatively little copper. The pyroxene-amphibole-sulphide type and the sulphide-replacement type with little silicate are also widely distributed, and some large deposits have been mined. The great deposits at Bisbee, Ariz., and the limestone-replacement deposits at Bingham, Utah, are examples of such deposits, and many of less importance have been mined.

Deposits of this type in the eastern Cordilleran region are usually close to intrusive rocks, but deposits of similar mineralogical composition in the Boundary district of British Columbia and possibly in the Appalachian region may be far from igneous contacts. In this respect they resemble the replacement lenses in schist already discussed, and their distance from intrusive rocks may be attributed to the same cause.

With such deposits may be grouped those in limestone far from known intrusive bodies, of which the chalcocite deposits of the Copper River district, Alaska, are outstanding examples.

Oxidation has been deep in some of the deposits. In the massive garnet deposits oxidation is not very complete, and enrichment has usually been unimportant. In the Bisbee deposits oxidation has been extensive and there has been sulphide enrichment, but most of the copper has been fixed as carbonate by the associated limestone, resulting in the great oxidized deposits that furnished the early production.

A few large replacement deposits in limestone have been mined, but they have not yielded a very large part of the production of copper in North America.

VEIN DEPOSITS, PIPE DEPOSITS, AND DISSEMINATED DEPOSITS

Under veins, pipes, and disseminations may be grouped three types of deposits, each of which has distinctive features but shows such gradation that a definite separation is neither possible nor desirable. The deposits are similar in mineral composition and differ mainly in the character of the fractures that have controlled deposition.

VEIN DEPOSITS

The simplest, perhaps, are vein deposits, and the simplest veins are those that occupy a single fault or fissure. Large copper deposits of this character are uncommon but are approached in the Magma vein of Superior, Ariz., and the Old Dominion vein of Globe, Ariz.

Other districts, such as the Butte district, Montana, and the Morenci district, Arizona, have a complex vein system in which the larger veins can be mined as units, but parts of these areas resemble disseminated deposits.

In general, the veins tend to join and decrease in number as they go deeper, and finally a complex system of veins may merge into only a few root channels.

PIPE DEPOSITS

Pipe deposits are more or less cylindrical bodies that dip steeply. The rock within the pipe is usually much brecciated, although in some a core of relatively unbrecciated rock is surrounded by an envelope of highly brecciated rock. The ore minerals cement and partly replace the brecciated rock. In some pipes quartz has replaced most of the rock in a central channel, while in others replacement has been slight.

Pipes range in size from a few feet or tens of feet to hundreds of feet in diameter and in shape from nearly circular to elongated, in which case they resemble breccia veins. A few, like the Cactus pipe in Utah and the Colorada pipe at Cananea, which have been followed to what seems to be their roots, show a decrease in area and less brecciation with depth. The pipe type of deposit has been particularly productive at Cananea and Nacozari, Mexico, Pipelike deposits in limestone have been productive at Bisbee, Ariz.; the Campbell ore body is an example. Some, at least, of the disseminated deposits may be pipes on a large scale, such as the Utah copper deposits of Bingham, Utah. In many of the pipes the ore minerals, disseminated as a breccia cement and replacement, comprise only a small part of the ore, but in others, like the Colorada and Campbell pipes, parts of the ore bodies are massive sulphide. Most of the pipe deposits are in the Tertiary of the eastern Cordilleran region and were probably formed at shallow depth. The United Verde and United Verde Extension ore bodies of the Jerome district, Arizona, are pipelike and may link the pipe deposits with replacement deposits in schist.

The origin of this type of ore chamber is still open to question. In some the start was doubtless due to faulting or jointing under load, which favored brecciation. These permeable channels were enlarged by solution before the ore minerals were deposited. This has been called "mineralization stoping" by Locke.[10]

In other pipes the character of brecciation has suggested that explosive action has broken a more or less cylindrical body of rock. This permeable zone has later been traversed by mineralizing solutions which have cemented and partly replaced the breccia.

The importance of pipe deposits depends on where the distinction is drawn between them and other types, but typical pipes have produced much ore.

DISSEMINATED DEPOSITS

The disseminated (porphyry) type of copper deposit is confined to the Tertiary of the eastern Cordilleran region. It occurs characteristically in and around the tops of intrusive bodies that are upward extensions of batholithic intrusions. The batholiths rise in folded and faulted areas and advance farthest along lines of weakness, such as faults, and are guided by them. The intrusive bodies may therefore be simple upward projections in the form of stocks or cupolas, or they may follow a fault with a relatively flat dip and be modified forms of cupolas or stocks.

Development has shown that some of the disseminated deposits, such as those of the Bingham district, Utah, and the Ely district, Nevada, are in such modified stocks.[11]

[10] Locke, Augustus, The Formation of Certain Ore Bodies by Mineralization Stoping: Econ. Geol., vol. 11, 1916, pp. 601–622.

[11] Locke, Augustus, Disseminated Copper Deposits; Ore Deposits of the Western United States: Am. Inst. Min. and Met. Eng., 1933, p. 616.

Along the boundary of the stocks both the intruded and the intrusive rocks have been much jointed and fractured. The fracture zones were subsequently mineralized, mainly with pyrite and chalcopyrite in variable amounts and proportions. In some deposits, such as those of Bingham, Utah, most of the mineralization was in the stock; but some extended into the intruded quartzite, while in others, such as those of Miami, Ariz., much of the mineralization was in the intruded schist. The primary mineralization was characteristically low in grade but varied greatly in grade. The deposits have all undergone oxidation, which has affected them differently.

Deposits, such as those at Miami, Ariz., in which the primary mineralization or protore is low in copper, owe their commercial value to oxidation and leaching of the copper from the upper parts of the body and reprecipitation on the sulphides lower down. This process has raised the copper content of the enriched sulphide zone to several times that of the original protore. In places, this secondarily enriched sulphide zone has been partly oxidized and not greatly leached.

The sum of the oxidation process at Miami, for example, is a surface zone from which the copper is largely leached; below this is a relatively rich zone in which most of the copper is present as carbonate and silicate; still deeper is the zone of enriched sulphide; and, finally occurs the zone of unenriched sulphide.

In contrast are the deposits of the Ajo district, Arizona, in which the primary sulphides are mainly chalcopyrite and bornite and very little leaching has occurred during the process of oxidation. In the upper zone, starting practically at the surface, the copper is in the form of carbonate and silicate; this is followed by the primary zone of chalcopyrite and bornite. The copper content of the two zones is essentially the same.

The ore deposits of the disseminated type lie relatively flat but vary with the topography and are composed of four zones, any one of which, except the lowest, may be lacking: A leached gossan zone, a zone of oxidized copper minerals, a zone of enriched sulphide minerals, and the zone of primary minerals.

The downward extensions of the primary zones have been little developed, and their form is therefore little known. Locke has suggested that they will contract into pipes or veins, some of which may prove of value.

The main production of disseminated deposits has been from the enriched sulphide zone, although both the oxide and primary zones have yielded important quantities of copper.

In origin the deposits do not differ greatly from veins or pipes. The principal difference is the character of the ore chambers of the disseminated deposits, which favored distribution of the solutions through a large fracture zone and corresponding dissemination of the ore minerals.

SUMMARY

Copper deposits are grouped as lenticular deposits in schist; replacement deposits in sedimentary rocks; and veins, pipes, and disseminated deposits.

The lenticular deposits in schist are characteristic of the older, more deeply eroded areas; they are not closely associated with intrusive bodies. Mineralogically, they are characterized by the presence of pyrrhotite and sphalerite. The replacement deposits in sedimentary rocks are abundant in the Tertiary deposits, although some of similar character are present in the older deposits. In the Tertiary deposits they are closely associated with stocks. In the older deposits the association with stocks is not so evident.

The veins, pipes, and disseminated deposits are characteristic of the Tertiary deposits. They occur in or close to stocks. Structurally, they occupy zones of brecciation rather than of shear. Mineralogically, they differ from the lenticular deposits in their lack of pyrrhotite and sphalerite. Some grade outward into zinc deposits, but zinc is not abundant in the main copper deposits. These differences are thought to be due to the different depths at which the deposits were formed.

The native copper deposits of Michigan and the copper-nickel deposits of Sudbury are unusual types that seem to have resulted from unusual conditions.

SEDIMENTARY COPPER DEPOSITS

To the classes named may be added the disseminated deposits in little-altered sedimentary rocks, frequently referred to as the "Red Beds" deposits. The primary mineralization ordinarily comprises chalcocite disseminated through sandstone, shale, and conglomerate and commonly associated with vegetable matter. The original sulphides may be partly or entirely altered to carbonates, oxides, and silicates. In the United States such deposits are present in Texas, New Mexico, Arizona, Colorado, and Utah, but they have yielded little copper.[12] The deposits of Boleo, Mexico, perhaps belong with this group.[13]

EFFECT OF OXIDATION ON COPPER DEPOSITS

The effect of oxidation on the value of copper deposits is perhaps greater and more varied than on that of any other metal deposits. Several factors influence the oxidation process:

1. The character of the sulphide, particularly the proportion of pyrite to copper sulphide or copper-iron sulphides.
2. The character of the gangue minerals or enclosing rock, whether reactive or nonreactive.
3. The character of the deposit, whether massive sulphide or disseminated sulphide.
4. Factors such as climate, rate of erosion, position of water table, and oxidation resulting from mining.

CHARACTER OF SULPHIDES

Pyrite (FeS_2) in oxidizing is first changed to ferrous sulphate and sulphuric acid. If the oxidizing process continues the ferrous sulphate will oxidize to ferric sulphate, which may break down to give limonite and sulphuric acid. Pyrite gives abundant sulphuric acid. Pyrrhotite or more complex iron-copper sulphides may oxidize similarly

[12] Finch, J. W., Sedimentary Copper Deposits of the Western States; Ore Deposits of the Western States: Am. Inst. Min. and Met. Eng., 1933, p. 481.
[13] Touwalde, Marcel E., Origin of the Boleo Copper Deposits, Lower California, Mexico: Econ. Geol., vol. 25, 1930, pp. 113–114.

but obviously will yield less sulphuric acid than pyrite. The other common-metal sulphides, except manganese sulphide, oxidize to sulphates.

The formation of sulphuric acid and ferric sulphate promotes the solubility of the metals and is favorable to their movement. The presence of iron sulphide, especially pyrite, therefore favors leaching of copper.

CHARACTER OF GANGUE

Gangue minerals have been classed as reactive, intermediate, and inert.

The carbonates, calcite and dolomite, are most reactive; quartz and barite are inert; and the micas, feldspars, and other silicates are intermediate. An inert gangue, such as quartz, has no chemical influence on the oxidation products of sulphides and is favorable to their movement. Reactive gangues, such as calcite and dolomite, react with all the oxidation products of sulphides, neutralizing sulphuric acid, changing the metal sulphates to carbonates (most of which have low solubility), and causing precipitation of limonite from ferric sulphate. The presence of calcite or dolomite is unfavorable to the movement of copper and tends to produce copper carbonate deposits in the upper parts of the oxidizing body rather than a zone from which copper has been leached.

With an abundant carbonate gangue there will be little downward movement of copper. With a quartz or silicate gangue there may be much downward movement.

As the copper sulphate, other sulphates, and sulphuric acid move below the zone of oxidation they encounter sulphides, and the copper is precipitated as copper sulphide. Precipitation may result from direct reaction between the copper sulphate and the sulphide, or where pyrrhotite and sphalerite are present acid solutions may produce hydrogen sulphide, which will precipitate sulphides from the sulphate solutions.

In the massive sulphide bodies, such as the lenses in schist, the copper is precipitated before the solutions have penetrated far into the sulphides, resulting in a shallow zone of supergene (secondary) sulphides which in some deposits is very rich. This precipitation may be due to the abundance of sulphides, to the presence of pyrrhotite and sphalerite, or to both.

In the disseminated deposits the sulphate solutions penetrate deep into the zone of scattered sulphide grains (mainly pyrite and chalcopyrite) before all the copper is precipitated, and a thick zone of secondary sulphide is produced which is usually not rich.

The zones migrate downward, and oxidation of the secondary sulphide zone may produce a relatively rich zone of carbonates and oxides. The presence of the water table practically limits the downward movement of oxidation.

The net result of oxidation is the formation of four zones, any one of which, except the lowest or primary zone, may be lacking: (1) An upper zone from which copper has been leached; (2) a zone in which copper has been fixed as carbonate; (3) a zone in which copper leached from above has been deposited as secondary sulphide; and (4) the zone of primary sulphides.

In the northern regions glaciation has removed most of the top deposits which are affected by oxidation. In other regions both the carbonate and oxide zone and the zone of secondary sulphides have been productive, but they have varied in importance with the type of deposit. Much copper has been produced from deposits that have been changed and somewhat enriched by oxidation.

OXIDATION RESULTING FROM MINING OPERATIONS [14]

Lowering the water table by draining the mine permits circulation of water through ore that was previously under water. Where caving or shrinkage stoping is practiced, the increased permeability of the broken ore causes increased circulation and increased oxidation.

Postmine oxidation may result in difficulties in treatment and in losses. If it is recognized it may be possible partly to prevent it.

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LOCKE, AUGUSTUS. Leached Outcrops as a Guide to Copper Ores. Baltimore, 1926.

COPPER REGIONS AND DISTRICTS

UNITED STATES

The copper districts in the United States and parts of Canada and Mexico are shown in figure 6.

COPPER PROVINCE OF SOUTHWESTERN UNITED STATES

Southwestern New Mexico, central and southwestern Arizona, and adjacent parts of northern Mexico contain 10 or more copper districts whose outputs have each been large and many smaller districts; together these constitute one of the great copper provinces of the world. The geologic features of this province have been summarized by Tenney,[15] and much of the following description has been abstracted from his summary.

SEDIMENTARY ROCK

The oldest rocks consist of schist, largely of sedimentary origin, and gneisses into which have been intruded large bodies of granitic rock of varying composition. On this basement of earlier pre-Cambrian rocks, eroded to an almost plain surface, was deposited the later pre-Cambrian rock consisting of conglomerates, sandstone, shales, and limestones (the Apache group). After another period of erosion the Paleozoic rocks were deposited, which contain a large proportion of limestone. The early Mesozoic period is not represented by sedimentary rocks, but during Cretaceous time a thick series of conglomer-

[14] Gardner, E. D., and Sullivan, John D., Oxidation of Copper Sulphide Minerals in Copper Ore: Min. Jour. (Phoenix, Ariz.), Dec. 30, 1934.

[15] Tenney, J. B., Ore Deposits of the Southwest: 16th Internat. Geol. Cong. Guidebook 14, 1932, pp. 40–67; also, Copper Resources of the World; The Copper Deposits of Arizona. Internat. Geol. Cong., vol. 1, 1935, pp. 167–235.

ates, sandstones, and shales with subordinate amounts of limestone were formed. In Tertiary and Quaternary times, after another period of erosion, were deposited the thick Gila conglomerate and lake beds that filled the valleys to undetermined depth.

IGNEOUS ROCKS

There have been at least four periods of igneous activity: (1) Early pre-Cambrian, when there was both extrusion and intrusion on a large scale, probably several times; (2), late pre-Cambrian, when there was mild extrusion and possibly intrusion; (3), possibly late Jurassic or early Cretaceous, although this is not certain; and (4), late Cretaceous to the present time, when there has been a great outpouring of lavas of varying, although generally intermediate, composition that cover hundreds of square miles of the region and intrusion of granitic and porphyritic rocks, also mainly of intermediate composition, at many centers.

STRUCTURE

In early pre-Cambrian time folding, faulting, and intrusion occurred throughout the region and probably during more than one period. There were numerous structural disturbances during the Paleozoic and Mesozoic periods, but the next major structural revolution began in late Cretaceous time and continued through Tertiary time.

The whole region was uplifted, and the belt bordering the present Colorado Plateau was thrown into a series of mountain ridges roughly paralleling the plateau but in eastern Arizona swinging southward into Mexico. This upheaval raised the sedimentary rocks bordering the Colorado Plateau far above those within the plateau. This belt, called by Ransome the "Mountain province," was the locus of much igneous activity, and within it are most of the copper deposits. South and west of the Mountain province the rocks were broken, by faults, into blocks that were tilted to give the mountain ranges a northwesterly trend. The erosion of these ranges and the deposition of the debris from them in the depressions have built up the intermountain plains and produced the characteristic physiography known as the Basin and Range province.

MINERALIZATION

Mineralization was closely associated with the major structural revolutions and their accompanying igneous intrusions, the two of greatest importance being in pre-Cambrian and late Cretaceous-Tertiary times. The only large copper deposits known to be of pre-Cambrian age are those in the Jerome-Prescott region of Arizona. The other copper deposits probably belong to the late Cretaceous-Tertiary period, although some, such as the Bisbee deposits, may be earlier.

DISTRICTS OF SOUTHWEST PROVINCE

ARIZONA

Jerome district.[16]—The Jerome district is in north-central Arizona about 23 miles northeast of Prescott, from which it is easily reached over a good road.

[16] Summarized from Ransome, F. L., 16th Internat. Geol. Cong. Guidebook 14, 1932, pp. 20–22.
Tenney, J. B., Copper Resources of the World, vol. 1; Copper Deposits of Arizona: 16th Internat. Geol. Cong., 1935, pp. 179–189.

The basal rocks of the district are crystalline schists, which are largely metamorphosed rhyolites and rhyolitic porphyries. These are intruded by a stock of rather basic diorite. All these rocks, after prolonged erosion, were unconformably overlain by reddish sandstone, probably of Cambrian age, and this in turn was succeeded by Devonian and Carboniferous limestones. After uplift and erosion, these formations were covered by basaltic flows of Tertiary age; remnants of these remain, especially on top of the Black Hills.

The district is traversed by several faults, of which one, the Verde fault, has been active at different periods and has a total throw of at least 4,000 feet. The Jerome district contains two ore bodies of outstanding size and productivity—namely, that of the United Verde mine and that of the United Verde Extension mine. The ore bodies of the United Verde mine, chiefly chalcopyrite, occur as lenticular masses, in which the copper is distributed through a great pipelike body of quartz, pyrite, sphalerite, and very subordinate quantities of other sulphides. This mass is generally 500 to 800 feet in horizontal diameter, plunges at an angle of about 68° in a northwesterly direction, and has been mined to a depth of more than 3,000 feet with no essential change in character. The pipe lies in an embayment in the diorite, with that rock as the hanging wall. The quartz and sulphides have largely replaced the schistose and altered rhyolitic rocks into which the diorite is intrusive. Supergene enrichment of this pre-Cambrian ore body has been limited to the relatively short time that the top of the ore body has been exposed to erosion since the removal of the Cambrian and later formations which previously covered it.

The United Verde Extension ore body lies east of the United Verde mine, on the opposite or hanging-wall side of the great Verde fault. It is not exposed at the surface, being covered by about 850 feet of nearly horizontal Paleozoic beds and Tertiary basalt. The Extension ore body is similar in form and geologic relationships to the United Verde ore body but differs in two very significant respects. It is definitely cut off, at a depth of about 2,000 feet, by the Verde fault, and it underwent extensive oxidation and enrichment in pre-Cambrian time; consequently the main ore body was largely converted to chalcocite.

The facts mentioned, and others, indicate that the United Verde Extension ore body was formerly the top of the United Verde ore body; that it was sheared off and relatively downthrown a distance of 2,400 feet on the Verde fault in pre-Cambrian time; and that it reached its present relative position by a further throw of 1,600 feet in Tertiary or Quaternary time, as shown by the displacement of the basal bed of the Cambrian and, of course, of the higher formations, including the Tertiary basalt. (See fig. 7.)[17]

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TENNEY, J. B. Copper Resources of the World, vol. 1; The Copper Deposits of Arizona. 16th Internat. Geol. Cong., 1933, pp. 179–189.

[17] After Ransome, F. L., 16th Internat. Geol. Cong. Guidebook 14, 1932, pl. 4.

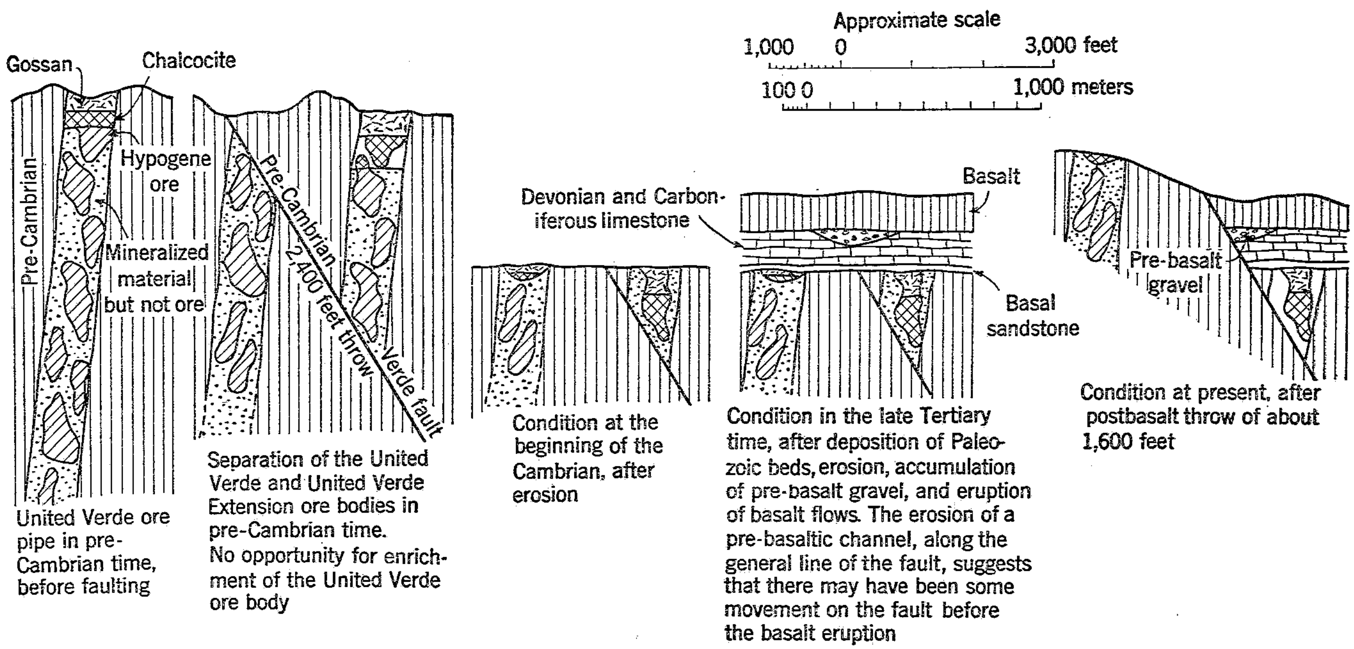

FIGURE 7.—Probable stages in the faulting of the United Verde and United Verde Extension ore bodies, Jerome district, Arizona. (After Ransome, F. L., 16th Internat. Geol. Cong. Guidebook 14, pl. 4.)

Ajo district.[18]—The Ajo district is in Pima County, Ariz., 43 miles south of Gila Bend on the Southern Pacific R. R. The oldest rocks in the immediate vicinity of the ore body are a series of lavas and associated tuffs, chiefly keratophyres and quartz keratophyres (locally called rhyolites), with subordinate quantities of andesite. Into this volcanic series was intruded a composite dikelike body or elongated stock comprising a discontinuous shell of quartz diorite and a core of porphyritic quartz monzonite. The exposed outcrop is about 2 miles by 1 mile.

The southwest part of the quartz monzonite and the adjacent volcanic rocks comprise the host rock of the ore deposits. The ore body is almost wholly in the monzonite, although adjacent volcanic rocks are mineralized. It is crudely elliptical (about 3,600 feet by 2,500 feet). Its average thickness is 425 feet and the maximum about 1,000 feet. Most of the ore is in a flat lens with a deeper northwestward-trending keel, but at the south end a tongue dips steeply southward to great depth.

The primary ore consists chiefly of chalcopyrite, with bornite and a little pyrite. The gangue is quartz and orthoclase, and the sulphides are distributed in veinlets and in grains through altered monzonite. A little tennantite, considerable magnetite and specularite, and a little sphalerite and molybdenite accompany the ore. The rock is highly orthoclastic, really pegmatized, along two main northwesterly zones, and the richest ore accompanies this intensely altered rock. The ore body is oxidized to a surprisingly level plane near the present water table at an average depth of 55 feet. The minerals of the oxidized ore were malachite with a little azurite, cuprite, tenorite, chrysocolla, hematite, and limonite.

The oxidized ore and the unoxidized ore have almost the same copper content; this indicates that the copper has moved very little. The age of the deposit is unknown.

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GILLULY, JAMES. Copper Resources of the World, vol. 1; Ajo District. 16th Internat. Geol. Cong., 1935, pp. 228–233.

——— Geology and Ore Deposits of the Ajo Quadrangle, Arizona. Arizona Bureau of Mines Bull. 141, 1937.

East-central Arizona region.—In east-central Arizona are closely grouped the Globe-Miami, Superior, Ray, and Christmas districts. The basement rock of this area is the pre-Cambrian Pinal schist intruded by granitic rocks. Resting on this is the Apache group, comprising shales, quartzites, and one limestone member (total thickness in Miami district, 1,300 feet) and on this, in turn, a smaller thickness of Paleozoic limestone. In the lower members of the sedimentary series are extensive sills of diabase of uncertain age. Into these earlier rocks were intruded, probably in Tertiary time, stocklike bodies of granitic to dioritic rocks. Ettlinger considers that the exposed stocks, around which the copper deposits are grouped, in the several districts

[18] Summarized from Gilluly, James, Copper Resources of the World, vol. 1; Ajo District: 16th Internat. Geol. Cong., Washington, 1935, pp. 228–233.

are upward projections from a single batholithic body underlying the whole region.[19]

Globe-Miami district.—The Globe-Miami district is at the northeast end of the elongated stock of Schultz granite which strikes north of east; the Superior district lies to the southwest along the strike of the stock. In the deeply eroded central portion of the stock no large ore deposits have been found.

The earlier developed deposits in the Globe end of the district are replacement veins in the Apache sedimentary rocks, the diabase and the Pinal schist. The primary mineralization comprises sericitization and silicification with pyrite, chalcopyrite, and, in places, specularite.

The disseminated deposits of the Miami end of the district are at the margin of the Schultz granite stock. Most of the ores occur in the Pinal schist near its contact with the granite, but some occur also in dikes and sills of the granite porphyry. The protore comprises disseminated pyrite and chalcopyrite in sericitized and silicified schist and granite.

The very low grade protore has been enriched by copper as chalcocite to form flat-lying bodies of ore. In parts of the district the upper part of this chalcocite ore has been further oxidized to carbonates and silicates, with little leaching of copper; thus it has formed large bodies of oxidized ore.

Since the ores were oxidized and enriched they have been buried beneath dacite flows and covered by Gila conglomerate, only part of which later has been eroded away. Extensive faulting after mineralization has lowered the area between the Miami and Globe ends of the district, and little is known of the possibilities of this down-faulted block.

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Superior district.—The Superior or Pioneer district is about 20 miles southwest of Miami and about 11 miles northwest of the Ray district. The entire sedimentary section of the area is present in this district and rests on the Pinal schist. Into the lower formations sills of diabase have been intruded which aggregate 2,500 feet in thickness. No granitic or monzonitic stock is exposed, but the west end of Shultz stock is covered by later lavas and may extend well toward Superior, and a stock of granitic rock is exposed 2 miles to the northwest in the Silver King area. The Magma ore body, the principal one of the district, closely follows a monzonitic dike. The deposit is a replacement vein in the nearly east-west Magma fault. Bornite is the most abundant primary ore mineral, although locally chalcopyrite and hypogene chalcocite are plentiful and sphalerite and galena are present. Some of the richest hypogene ore is in the diabase. The upper part of the ore body, which apexed below the 400-foot level, consisted largely of supergene chalcocite and was very rich. The deepest shaft is 4,000 feet, and the ore body has been developed to the 3,400-foot level.

[19] Ettlinger, I. A., Ore Deposits Support Hypotheses of Central Arizona Batholith: Am. Inst. Min. and Met. Eng. Tech. Pub. 63, 1928.

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Ray district.[20]—The Ray district is about 20 miles southwest of Globe. The ore bodies are similar in general character and origin to those at Miami but are genetically related to intrusive masses of quartz monzonite porphyry which are, superficially at least, much less extensive than the Schultz granite at Miami. Most of the ore is in the Pinal schist, but a little is found also in the porphyry. Geologic conditions are similar to those at Miami, but the ore bodies have been less affected by faulting.

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Christmas district.—The Christmas district about 25 miles south of Globe has made a relatively small production from replacement deposits in limestone near an intrusive body of quartz-mica diorite. These may be classed as contact deposits.

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Clifton-Morenci district.[21]—The Morenci district is in southeastern Arizona about 20 miles from the New Mexico line. The district formerly yielded copper from contact-metamorphic deposits, but more recent production has been from low-grade disseminated deposits. In this district low-grade ore of this type was first concentrated successfully in the United States.

Upon a base of pre-Cambrian granite and schist rest unconformably Paleozoic beds which have a total thickness of about 1,000 feet. These are unconformably overlain by Cretaceous beds. After invasion by granitic and monozonitic porphyries, in early Tertiary time, the region was covered by later Tertiary lavas, part of which have been removed by erosion.

The ore bodies fall into three main classes: (1) Irregular or roughly tabular bodies in limestone or shale, near porphyry; (2) lodes or veins; and (3) irregular deposits of low-grade disseminated ore in porphyry. The last are the bodies last worked, and the large reserves of the district are of this type. All the ore bodies have undergone supergene enrichment with the formation of chalcocite from a protore containing mainly pyrite, chalcopyrite, and sphalerite. Oxidation and enrichment probably began in Tertiary time, before the eruptions of the lava, which, for a time, covered the deposits.

[20] Summarized from Ransome, F. L., Ore Deposits of the Southwest: 16th Internat. Geol. Cong. Guidebook 14, 1932, p. 19.

[21] Summarized from Ransome, F. L., Ore Deposits of the Southwest: 16th Internat. Geol. Cong. Guidebook 14, 1932, p. 16.

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TENNEY, J. B. Copper Resources of the World, vol. 1; The Copper Deposits of Arizona. 16th Internat. Geol. Cong., 1935, pp. 213–221.

Bisbee district.—The Bisbee district, which since 1880 has produced metal, largely copper, with a value of nearly $800,000,000, has been the most productive district in Arizona.

The rocks of the district comprise the basal Pinal schist overlain by about 5,000 feet of Paleozoic sedimentary rocks, largely limestone. Resting on these with marked unconformity are Cretaceous and possibly earlier Mesozoic sedimentary rocks of about equal thickness, which are largely shales and sandstones but have a basal conglomerate and a prominent, central, limestone member. (See fig. 8.)[22]

Into the pre-Cretaceous rocks have been intruded stocks, sills, and dikes of granite porphyry; the largest stock is the Sacramento Hill stock. The earliest pronounced structure was the Dividend fault, striking northwesterly, on which there was strong pre-Cretaceous displacement with relative upthrow on the northeast. Almost all the Paleozoic sedimentary rocks were removed from this upthrow block before the Mesozoic rocks were laid down. Later, probably during the Laramide revolution, the district was further folded and broken into blocks by faulting. This was accompanied and followed by the intrusion of the granite porphyry and by the mineralization, both of which were guided by the faults. The Sacramento Hill stock is in the Dividend fault. Geologists who have worked in this district do not agree on the time of the granitic intrusion and accompanying mineralization. The early wave of mineralization produced a zone of contact silicates with hematite and magnetite in the limestone adjacent to the Sacramento Hill stock. Further from this stock, and perhaps associated with unexposed stocks, the early mineralization is represented by pipes of silica breccia with hematite replacing limestone.

Outside of the more intense contact-metamorphic zone abundant pyrite and quartz were deposited, followed by copper sulphides, mainly chalcopyrite, which partly replaced the pyrite and other minerals. Around the south and west side of Sacramento Hill a crescent-shaped zone of mineralization resulted, which pitched to the southeast with the dip of the formations and terminated at both ends in the Dividend fault.

Outside of the main crescentic belt the mineralization extends along roughly radial lines much farther from the Sacramento Hill stock. Within the main crescent most of the ores were copper. On the outer fringes, perhaps associated with independent centers, lead and silver become abundant, and the district has produced considerable lead.

The granite porphyry of Sacramento Hill was intensely mineralized with iron and copper sulphides, resulting in a deposit of the disseminated type which, by supergene enrichment, became ore.

Oxidation has extended to great depth (2,200 feet) in the Campbell ore body, but in the deeper levels it is not extensive. The shallower ores in the north end of the district were largely carbonates and oxides

[22] After Tenney, J. B., 16 Internat. Geol. Cong. Guidebook 14, pl. 13, p. 52.

with some native copper, but most of the deep ores to the southeast are primary sulphides.

In the more highly metamorphosed limestones there was considerable sulphide enrichment in the ores replacing limestone. In the less metamorphosed rocks the abundant limestone precipitated the copper as carbonate with little movement. The production from disseminated ores has been small compared with that from limestone-replacement ores.

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Other Arizona districts.—Numerous other copper districts in Arizona have produced copper or may produce it in the future.

In the Twin Buttes district, Pima County, contact deposits have yielded considerable copper, and in the Silver Bell district, Pima County, both contact deposits and veins in granitic intrusives have been productive.

The Bagdad district in southwestern Yavapai County contains large deposits of low-grade disseminated ore which as yet have not been productive on a large scale.

In the Turquoise district, Cochise County, the deposits are replacements in Paleozoic limestones which have been faulted and intruded by monzonite porphyry.

NEW MEXICO

The principal copper deposits of New Mexico are grouped in a small area in the southwestern part of the State in the same geological province as those of Arizona and Sonora, Mexico.

Santa Rita district.—The deposits of the Santa Rita district, the first to be mined in the western United States (early in the last century) have been by far the most productive.

The district is in an area of varied mineralization. A few miles to the north is the Hanover district, which has yielded iron and zinc ores. Ten miles to the northwest is the Pinos Altos district, and 15 miles to the west is the Silver City district; both have produced silver, lead, gold, and manganese.

The rocks of the district comprise the familiar pre-Cambrian basement of schist and granitic rocks overlain by about 2,700 feet of Paleozoic sedimentary rocks, most of which are limestone. These in turn are overlain by 880 feet of Cretaceous sediments with 80 feet of sandstone at the base and shale above. Above are some 3,000 feet of Tertiary lava flows and interbedded sands and gravels and, finally, 1,000 feet of recent sands and gravels with interbedded basalt flows.

Volcanic activity began in Cretaceous time and has continued at intervals nearly to the present, resulting in the immense lava fields to the north and west and in the intrusion of stocks, laccoliths, dikes, and sills. At Santa Rita an early sill of quartz diorite was followed by a stock of granodiorite porphyry and associated dikes. After the solidification of the granodiorite porphyry stock, its outer border and the adjacent intruded rock were intensely fractured.

Mineralizing solutions traversing this permeable fracture zone deposited the protore as an envelope around a relatively unmineralized core of granodiorite. Like the disseminated deposits at Miami and Ray, Ariz., this ore is partly in the intrusive rock and partly in the bordering intruded rock. The hypogene mineralization resulted in thorough sericitization and silicification of the fractured rock and deposition of pyrite and chalcopyrite. Oxidation, downward movement of the copper, and reprecipitation formed an enriched chalcocite zone which has yielded most of the ore. Part of chalcocite, however, was oxidized to native copper, copper oxide, and copper carbonates, which were the bases for early operations in the district. The oxidation and enrichment were accomplished in two stages; after these were well-advanced the district was covered by lavas, and oxidation was renewed when the lavas were eroded. In this respect the district resembles the Morenci district of Arizona. Both the top and bottom of the ore body are very irregular in contour.

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Burro Mountain district.—There has been relatively little production from the Tyrone district in the Burro Mountains, Grant County. The ore is closely associated with quartz-monzonite porphyry which intrudes pre-Cambrian granite. The veins are in fractures on the border of the granite and porphyry. The primary pyrite-chalcopyrite protore has been enriched by supergene chalcocite to form the ore.

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Lordsburg (Virginia) district.—The Lordsburg district in central Grant County also has produced relatively little copper, most of which came from the Eighty-Five mine, formerly operated by the Calumet & Arizona Copper Co. The highly siliceous ores were used as flux at the Douglas smelters. Andesitic flows and breccia have been cut by granodiorite intrusive bodies. The quartz-tourmaline-copper veins cut both rocks. Much of the ore mined has been oxidized.

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CALIFORNIA

GEOLOGY

The large copper deposits of California are associated with intrusions of granodioritic and allied rocks of late Jurassic and early Cretaceous age.

SHASTA COUNTY DISTRICT

The Shasta County copper district is in the Klamath Mountains at the head of the Sacramento Valley, a few miles north of Redding. The Sacramento River crosses the belt. To the east of the river are the Bully Hill and Afterthought districts and to the west the Iron Mountain and Little Backbone districts. The sedimentary rocks are of Devonian, Carboniferous, and Triassic age and are mainly shales or slates, with some limestone. These rest on altered andesite of pre-Devonian age.

The rocks were intruded, probably in early Cretaceous time, by siliceous soda alaskite porphyry followed by quartz diorite. The country rock of most of the copper deposits is soda alaskite porphyry. The ore minerals have replaced this rock along fissures and zones of shearing. In the western districts the larger deposits are flat-lying, tabular bodies of small vertical extent. Some deposits more like fissure veins and with a relatively high percentage of chalcopyrite have been worked.

In the eastern districts the character of ore bodies is more definitely that of fissure veins, and the ore occurs locally in the sedimentary rocks near the porphyry. Sphalerite is a notable constituent.

The sulphide lenses in the western district are massive pyrite with relatively little chalcopyrite and varying quantities of sphalerite. Quartz gangue is scarce. The vein deposits in the western district are siliceous and contain a relatively high percentage of chalcopyrite.

In deposits that have been exposed by erosion, such as the Iron Mountain and Bully Hill deposits, oxidation has produced a shallow but rich secondary sulphide zone. Many of the deposits have no outcrop.

The gold and silver content of the ores has been as much as 3.5 cents per pound of copper.

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FOOTHILL DISTRICT

The Foothill belt is a rather extensive district with its best development in Calaveras County.

The ores, comprising pyrrhotite, pyrite, chalcopyrite, and sphalerite, occur as lenticular bodies replacing schistose rocks. They vary in different parts of the district from iron and copper sulphide with low gold and silver content to ores carrying considerable lead, zinc, and precious metals.

The deposits are thought to be associated, in origin, with the Sierra Nevada batholithic rocks.

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PLUMAS COUNTY COPPER BELT

The copper deposits of the Engles, Superior, and Walker mines in the Taylorville region are closely associated with the granitic rocks of the Sierra Nevada batholith, which are quite complex in this region and intrude a series of metamorphosed volcanic rocks. The formation of the deposits followed closely the consolidation of the intrusive rocks.

The Engles deposit contains bornite and chalcocite in norite and quartz diorite, also in roof pendants of these intrusive bodies. The ore bodies of the Walker mine are near the contact of quartz diorite and highly metamorphosed, intruded rocks. Locally, tourmaline is abundant.

Various interpretations have been presented of the origin of the deposits, but Knopf and Anderson show clearly that it is of the replacement type and is near the upper-temperature limit for sulphide deposition.

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OREGON AND WASHINGTON

Oregon and Washington have yielded little copper. The Waldo district in Grant County, Oreg., and the Blue Mountain district in eastern Oregon have been most productive. The ore bodies of the Waldo district are pyrite with subordinate quantities of chalcopyrite and bornite; they occur in quartz porphyry near its contact with slate and as bodies of pyrrhotite and chalcopyrite in greenstone or serpentine.

Most of the production of Washington has been from the Danville district, Terry County, and the Chewelah district, Stevens County.

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NEVADA

Copper has been produced in many districts in Nevada. The earliest record is for 1873, but the output was small until 1908 when the Ely district began production, which has continued to the present. Other districts are the Yerington, Battle Mountain, and Jack Rabbit.

ELY (ROBINSON) DISTRICT

The Ely district is in the Egan Range, which consists of Paleozoic limestones and shales that have been intruded by monzonite porphyry, probably of Tertiary age although possibly earlier. Bateman believes there were two distinct intrusions. The intrusive bodies take various forms which are controlled by faults and folds; they are not typical stocks. The district was buried beneath volcanic rocks after the ores had been exposed and oxidized. These have been partly eroded.

The deposits are of two types—replacement deposits in the limestone and disseminated deposits in the altered monzonite porphyry. The deposits in the limestone are near the quartz monzonite and are associated with garnet and other contact silicates and usually with abundant jasperoid. These deposits have been partly oxidized, and production from them has been relatively small. The disseminated ores are principally in the monzonite porphyry, although they extend into the adjacent sedimentary rocks. The monzonite porphyry has been much shattered and altered to a rock composed of secondary orthoclase, quartz, sericite, and brown mica. The metallic minerals of the primary ore, pyrite and chalcopyrite, occur as veinlets and disseminated grains in the altered rock.

The ore has resulted partly from supergene enrichment, although Bateman states that supergene enrichment has been of minor importance and that the hypogene mineralization is of ore grade. The leached capping over part of the disseminated ore was thin enough to permit stripping and open-pit mining of the ore. Augustus Locke states that deeper mining has revealed that the intrusive bodies are expanded plugs of limited thickness tapering downward. After an interval of nonminable pyrite is passed, primary chalcopyrite ore of minable grade is encountered.

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YERINGTON DISTRICT

The main ore bodies of the Yerington district in Lyon County are chalcopyrite and pyrite in a gangue of pyroxene, andradite, or epidote. Magnetite and hematite are conspicuously absent. The ore averages from 2.75 to 6 percent copper, with traces of gold and silver.

A succession of igneous rocks invaded a series of Triassic andesites, keratophyres, and limestones, probably in late Jurassic time. The intrusive sequence began with medium-grained basic granodiorite which was followed by a coarse-grained quartz monzonite. These intrusions were followed by aplite and by quartz-monzonite porphyry dikes.

Two stages of metamorphism are shown. During the first the Triassic rocks were extensively altered to calcium silicate rocks, chiefly garnetites. Quartz monzonite porphyry dikes intruded into the garnetites. Faulting then dislocated the dikes, and along some of these faults metalliferous solutions rose and formed the copper deposits.

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IDAHO

Many districts of Idaho have produced some copper, but no large copper deposits have been developed. In the Coeur d'Alene district the Snowstorm mine has produced considerable copper ore. The deposit [23] consists of disseminated bornite, chalcocite, and chalcopyrite in beds of pre-Cambrian quartzite.

In the Loon Creek district considerable copper has been produced, largely from the Lost Packer mine,[24] which is developed on a fissure vein in schist. Chalcopyrite, the principal metallic mineral, is accompanied by pyrite and pyrrhotite and a gangue of quartz and siderite.

The Alder Creek district has produced copper from contact deposits that replace blocks of limestone enclosed in granite porphyry. The ore minerals are chalcopyrite, pyrite, and pyrrhotite in a garnet gangue.[25]

Contact copper deposits are known in the Seven Devils district, [26] but there has been little production from them.

UTAH

Many districts in Utah have produced copper, but in only a few is it the most important metal. By far the largest production has come from the Bingham district.

[23] Ransome, F. L., and Calkins, F. C., Geology and Ore Deposits of the Coeur d'Alene District, Idaho: Geol. Survey Prof. Paper 62, 1908, pp. 150–152.
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[24] Umpleby, J. B., Contributions to Economic Geology, 1912, pt. I; A Preliminary Account of the Ore Deposits of the Loon Creek District, Idaho: Geol. Survey Bull. 540, 1914, pp. 167–211.

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BINGHAM DISTRICT

The Bingham district has been a steady and large producer of copper since 1896 and has one of the largest reserves of developed ore in the country.

The earliest rocks exposed in the district are a thick series of quartzites with interbedded limestones of Pennsylvania age. In Tertiary time there were extrusions of lavas and intrusions of stocks, sills, and dikes of monzonite porphyry into the sedimentary rocks.

Two stocklike bodies known as the Bingham and Last Chance stocks are apparently connected by dikes. Mineralization has been far more intense in and around the Bingham stock. Most of the ores within the stock are copper ores. In the sedimentary rocks close to the stocks copper ores also prevail, but they grade outward into zinc-lead-silver ores that give an irregular but definite zoning of the metals. The general shape of the disseminated ore body, as described by Boutwell, is that of a cylinder standing nearly vertical or an oval drum standing on its head. This body of ore, as indicated by company tests, extends along a northeast-southwest axis about 6,000 feet with a width of nearly 4,000 feet and a vertical depth of 2,000 feet.

The ores of the Bingham stock are typical of the disseminated-copper deposits. The monzonite has been intensely shattered and altered to varying degrees. In what is known as "dark porphyry" the hornblende and augite of the monzonite have been altered to brown mica, and the plagioclase has been partly altered to sericite and partly to secondary orthoclase. The "light porphyry", which is more intensely altered, is composed essentially of quartz, secondary orthoclase, and sericite, together with the metallic minerals; where alterations have been most intense quartz is the main gangue mineral. The primary ore minerals are pyrite and chalcopyrite, which occur as grains and veinlets disseminated through the shattered and altered monzonite porphyry. Locally molybdenite is present. The primary ores are apparently richer in copper than those of several of the other disseminated deposits.

Oxidation has caused leaching of part of the copper content from the upper part of the ore body and reprecipitation of copper on the sulphides lower down to form an enriched sulphide zone. As already stated, the enrichment is less than in several other disseminated deposits, and part of the primary mineralization may be ore. The proportion of the ore minerals in recently mined ore follows: [27]

	Percent		*Percent*
Pyrite	2. 15	Chalcocite	0. 18
Chalcopyrite	1. 66	Total copper sulphides	2. 17
Bornite	1. 14	Total sulphides	4. 32
Covellite	. 19		

The thickness of the oxidized zone varies greatly but averages more than 100 feet. It is not all greatly leached. Some of the oxidized ore mined and treated in 1918 averaged 0.882 percent copper.

The thickness of the enriched sulphide zone is 500 to 600 feet, although commercial ore has been encountered at greater depth in diamond drilling.

[27] Head, R. E., Crawford, A. L., Thackwell, F. E., and Burgener, Glen, Detailed Statistical Microscopic Analyses of Ore and Mill Products of the Utah Copper Co.: Rept. of Investigations 3288, Bureau of Mines, 1930, 93 pp.

The deposits in the sedimentary rocks surrounding the Bingham stock are bed replacements associated with fissures in the limestone.

The primary ores contain abundant pyrite and varying amounts of chalcopyrite in a gangue of silicates and unreplaced limestone. The ores are thus allied to the pyrometosomatic deposits but are not of the massive garnet type. Oxidation has produced a shallow but rich secondary sulphide zone.

The copper ores extend several hundred feet beyond the monzonite stock and grade outward into zinc-lead-silver ores.

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MINOR DISTRICTS

The San Francisco and Beaver Lake districts in Beaver County, Lucin in Box Elder County, and Lutsagubet in Washington County have produced copper ore; there has also been important production, largely incidental to the production of other metals, from the Park City, Cottonwood, Tintic, Ophir, and Rush Valley districts. The largest production from the San Francisco district has been from the Cactus mine. The deposit is a breccia pipe with disseminated pyrite and chalcopyrite in a gangue of abundant carbonate. The deposit showed no zone of sulphide enrichment.

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COLORADO

The mines of Colorado have yielded considerable copper, but most of it has been incidental to the production of lead, zinc, and precious metals. The State contains no large districts or mines that have been valuable chiefly for copper.[28]

MONTANA

Copper has been produced from numerous districts in Montana, but the Butte district is the only one with a large output.

Part of the ore deposits at Butte are fissure fillings, but most of them are replacement veins; they are confined to granite and aplite

[28] Burbank, W. S., and Lovering, T. S., Copper Resources of the World, vol. 1; Copper-Bearing Ores of Colorado: 16th Internat. Geol. Cong., 1935, pp. 253–260.

in the Boulder batholith, which was intruded in late Cretaceous or early Tertiary time. Three main systems of mineralized fissures cut the granite in three directions, and three or four later systems are unmineralized. The copper minerals are chiefly primary chalcocite and bornite with considerable enargite. Sphalerite, galena, native gold, native silver, and silver-bearing tetrahedrite are other valuable minerals. Pyrite, one of the first minerals formed, is the most abundant sulphide. The ores of the district show a well-marked zonal arrangement; copper minerals form a central core; copper and zinc minerals an intermediate zone; and zinc, lead, and manganese minerals a peripheral zone. (See fig. 9.)[29]. Oxidized copper, zinc, and lead

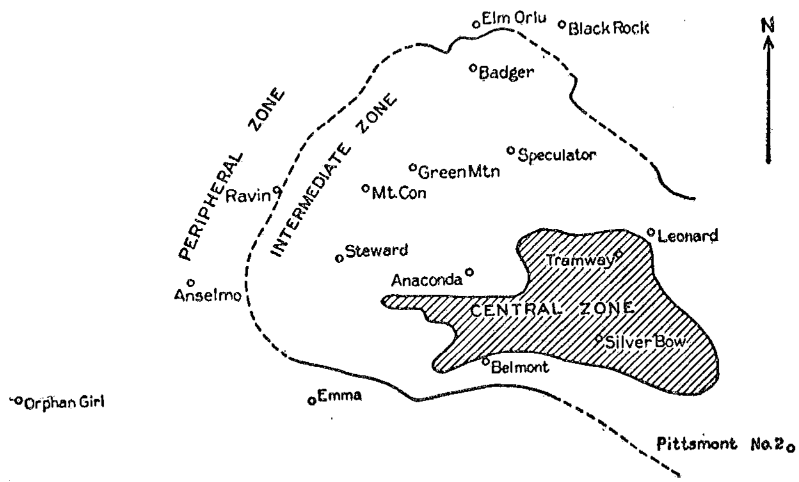

FIGURE 9.—Plan of Butte mines at altitude of 4,600 feet, showing general zonal arrangement of ore minerals. (Revised after Sales, R. H., Trans. Am. Inst. Min. Eng., vol. 46, 1914, p. 58, fig. 7.)

minerals have always been inconspicuous, but oxidized silver minerals were formerly mined. Superficial alteration has resulted in thorough leaching of the surface deposits to a depth of 100 to 500 feet and in the deposition of secondary "sooty chalcocite" for 100 to 500 feet below the leached zone. The main mass of ore, which extends from a depth of 500 feet to more than 4,000 feet, is believed to be of primary origin and to have been deposited by ascending magmatic solutions under conditions of intermediate thermal intensity.

Ore at Butte occurs as true fissure fillings, as deposits largely replacing country rock, and as deposits disseminated through the country rock. All these types may be included in a single vein, and the proportion of replacement and filling differs in different localities. In the central copper area, where horse-tail fracturing has occurred, disseminated deposits are abundant. The early east-west, or Anaconda vein system, shows the greatest tendency toward replacement of wall rock, and in places great zones of crushing, 100 feet or more across, may be so completely mineralized as to give the appearance of a single vein. Some observers state that as much as 60 to 70 percent of the mineralization of the Butte district has been accomplished by

[29] After Sales, R. H., Ore Deposits at Butte, Mont.: Trans. Am. Inst. Min. Eng., vol. 46, 1914, pp. 3-109, fig. 7.

replacement of the granite wall rock and included blocks. Many of the ore bodies of the northwest or Blue system consist of fissure fillings, but blocks of granite included in the veins may be largely replaced. In describing the character of the ore bodies, Sales states:

> The fissure-vein structure is the rule; * * * the ore bodies display rather well defined boundaries when broadly considered * * *. In the Leonard, West Colusa, Rarus, and Tramway mines the largest ore bodies are more in the nature of mineralized, highly fissured granite, having boundaries which are often commercial rather than geological. * * * 60 to 80 per cent by weight of the ore is altered granite, which usually, though not always, carries sufficient quantities of valuable minerals in seams, impregnations, or disseminations to constitute ore.

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ALASKA

Production of copper from Alaska in notable quantity began about 1903 and has been continuous since then. The principal districts, in order of production, have been the Copper River district, the Prince William Sound district, and the Ketchikan district.

COPPER RIVER DISTRICT

The rocks of the district comprise the Nikolai greenstone (altered amygdaloidal basalt flows) overlain by Triassic dolomitic limestone, which in turn is overlain by shales. These rocks are cut by monzonitic intrusives, but the intrusives are not closely associated with the copper.

The ores are principally chalcocite, with some bornite and covellite replacing limestone near the greenstone contact. This is one of the few large copper deposits known in which chalcocite, the principal ore mineral, has been regarded as a hypogene mineral.

The deposit outcrops prominently; in fact, much ore was taken from talus derived from it. Oxidation has been slight.

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PRINCE WILLIAM SOUND DISTRICT

The deposits of the Prince William Sound district, which for some years produced considerable ore, comprise pyrrhotite, chalcopyrite, chalmersite, and pyrite with minor quantities of other sulphides, occurring chiefly in shear zones in graywacke and slates.

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KETCHIKAN DISTRICT

The principal deposits are of contact or pyrometosomatic type; the ores are composed of chalcopyrite, pyrrhotite, magnetite, and pyrite in a silicate gangue.

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MICHIGAN

The copper district of Keweenaw Point (fig. 10)[30] in northern Michigan has been the second largest producer of copper in the world. Since 1845 the district has produced more than 8½ billion pounds of copper.

The rocks of the district comprise a thick series of basaltic flows, felsite conglomerates, and sandstones. The middle and most productive part of the series consists mainly of flows with subordinate conglomerates. Near the base of the series are intrusions of gabbro and a siliceous differentiate, similar to the Duluth gabbro of Minnesota.

The tops of the lava flows contain many gas cavities which are partly filled with minerals to form amygdaloids. In a small part of the flows the upper cellular part is a breccia which forms permeable zones that have determined the location of the ore shoots in the amygdaloidal lodes.

[30] After Broderick, T. M., and Hohl, C. D., Geology and Exploration in the Michigan Copper District: Min. Cong. Jour., 1931, p. 480.

During cooling and solidification the top part of the flows, especially the breccia tops, became rich in hematite.

The copper region is on the south rim of the Lake Superior syncline that underlies the lake. Transverse to the south limb of the Lake Superior syncline are synclines and anticlines that pitch down the dip of the major fold, and on these larger structures are small anticlines and synclines.

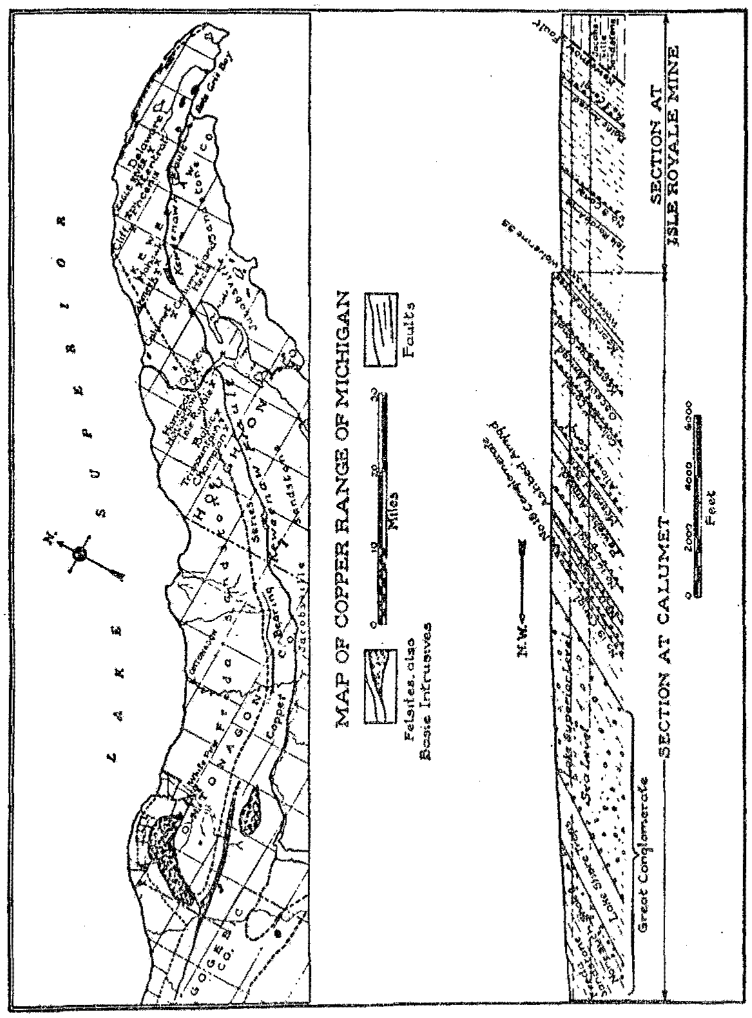

FIGURE 10.—Map and cross section of copper range. (After Broderick and Hohl, Min. Cong. Jour., October 1931, p. 480.)

The Keweenaw fault is a reverse fault of large throw which borders the copper-bearing series on the south.

The copper deposits are of two classes—lode deposits and fissure deposits. The lode deposits consist of conglomerate lodes, which are mineralized beds of felsite conglomerate interbedded with lava flows, and amygdaloid lodes, which are mineralized vesicular and brecciated tops of lava flows. The fissure deposits are veins along fractures, of which some are parallel and some transverse to the beds.

Only two conglomerates have been extensively explored—the Calumet and Hecla and the Allonez. The largest production has been from the Calumet and Hecla conglomerate. Through most of its length of 50 miles or more it is a thin sandy bed. At Calumet it is a conglomerate lens 5 to 30 feet thick which thickens and widens down the dip. The ore shoot is confined to this conglomerate lens. The copper content decreases quickly where the conglomerate grades into sandstone. The native copper replaces the fine matrix of the conglomerate and some of the pebbles.

This ore body, which is developed for 2 miles along the strike and nearly as far down the dip, is one of the great ore shoots of the world.

With the conglomerate lodes might be included the Nonesuch sandstone lode in the extreme southern part of the district, which has made a relatively small production.

The amygdaloidal lodes are in the fragmental type of flow tops, although the Pewabic lode is partly of a cellular type with unusually large coalescing openings. The more productive amygdaloidal lodes from the base upward are the Baltic, Isle Royale, Kearsarge, Osceola, Pewabic, and Ashbed.

The ore in all lodes occurs in shoots that were determined by permeability of the beds. Two conditions seem to have favored this: (1) A bed that is prevailingly impermeable but contains permeable portions extending far downward, like the Calumet and Hecla conglomerate, and (2) a bed that is prevailingly permeable but contains impermeable streaks which cause a concentration of solutions beneath them, like the Osceola amygdaloid.

The veins in the north end of the district are in cross fissures on the Keweenaw and Allonez anticlines. The mineralization of the fissure is near its intersections with strong amygdaloids and under the "slide" at the base of the Greenstone flow. Most of the fissures at the south end of the district are strike fissures which dip more steeply than the lodes. They also are mineralized near the intersection of lodes.

Some of the lodes have been developed for nearly 2 miles down the dip. From a consideration of the mineral zoning, based on the arsenic content of the copper and on associated minerals, T. M. Broderick has concluded that mineralization in the district shows a range of 20,000 feet.

The source of the ore-forming solutions is believed to be an underlying intrusive body allied to the Duluth gabbro. The solutions rose through permeable channels, and the copper was deposited as native metal, rather than as the more ordinary sulphide minerals, due to the oxidizing effect of the hematite-rich lodes.

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APPALACHIAN DEPOSITS

Copper deposits in the Appalachian region are distributed from Quebec to Georgia. The largest production has been from the Ducktown district, Tennessee.

The deposits are of several types. The pyrometasomatic iron deposits of Lebanon, Pa.,[31] have produced some copper as a byproduct of the treatment of iron ores.

Deposits associated with the Triassic trap rocks that are present from New Jersey to Virginia yielded some copper in the early days. Newhouse[32] regards these as allied to the magnetite deposits of Cornwall. By far the most important deposits are sulphide replacements of schistose rocks. Some of these are too low in copper to constitute copper ore but have been mined for sulphur, the copper being recovered as a byproduct. Others have yielded copper from a rich supergene sulphide zone, and still others have been mined for copper in the primary zone.

Deposits of this type are those of the Milan mine, Coos County, N. H.;[33] the Ely district, Vermont;[34] the Dumphries district, Virginia;[35] the "Great Garsen lead" extending from Virginia into North Carolina;[36] the Virgilina district of North Carolina and Virginia, which differs from the others as the copper occurs as bornite and hypogene chalcocite;[37] the Gold Hill district, North Carolina;[38] and the Gullowhee district, North Carolina.[39]

By far the most productive deposits of this type have been those of the Ducktown district, Tennessee.[40]

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[31] Spencer, A. C., Contributions to Economic Geology, part I; Magnetite Deposits of the Cornwall Type in Berks and Lebanon Counties, Pa.: Geol. Survey Bull. 315, 1907, pp. 185–189.
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[32] Newhouse, W. H., Mineral Zoning in the New Jersey-Pennsylvania-Virginia Triassic Area: Econ. Geol., vol. 28, 1933, pp. 613–633.

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[35] Watson, T. L., Mineral Resources of Virginia: Virginia Jamestown Exposition Comm., 1907, p. 494.

[36] Watson, T. L., Work cited, p. 511.

[37] Laney, F. B., The Gold Hill Mining District: North Carolina Geol. and Econ. Survey Bull. 21, 1910, pp. 9–137.

[38] Laney, F. B., The Geology and Ore Deposits of the Virgilina District of Virginia and North Carolina: Virginia Geol. Survey Bull. 14, 1917, 176 pp.

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[40] Emmons, W. H., Laney, F. B., and Keith, Arthur, Geology and Ore Deposits of the Ducktown District Tennessee: Geol. Survey Prof. Paper 139, 1926, 114 pp.
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Emmons gives the following summary of the geology of the Ducktown district:

The Ducktown mining district, in the southeast corner of Tennessee, is the largest producer of copper and sulphuric acid in the southern Appalachian region. The deposits were opened in 1847 and have been worked almost continuously since that date. Two companies are operating in the district—the Tennessee Copper Co. and the Ducktown Copper, Sulphur, & Iron Co.

The country rock consists of schist and graywacke intruded by gabbro. A great group of granitic batholiths which extend the length of the southern Appalachian Mountains lies a few miles to the east, and the deposits are believed to have been formed in connection with the intrusion of the granite, probably near the end of the Paleozoic era.

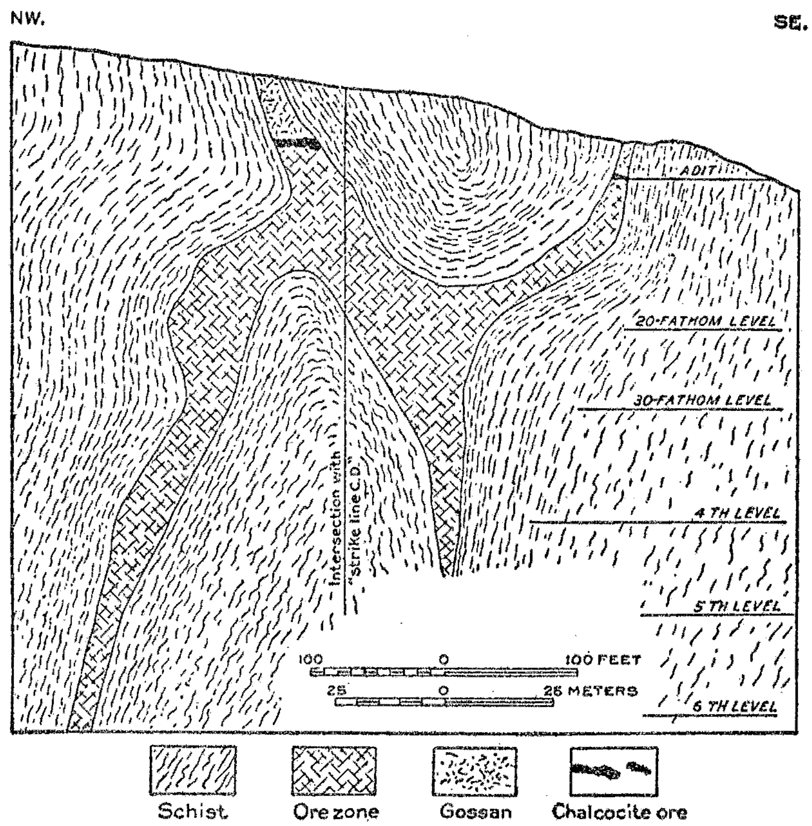

FIGURE 11.—Cross section of Mary mine, Ducktown district.

The deposits are great lenses of heavy sulphide ore (fig. 11).[41] They extend, with interruptions, for thousands of feet on the strike and are from 1 to 300 feet or more wide. Except where faulted they lie with the bedding of the schists and have the form of close folds; some of them are "saddle reefs." It is believed that they represent a limestone bed that was closely folded and extensively faulted and subsequently replaced by hot solutions that rose along faults and along the folded and faulted limestone bed.

At places the original bedded limestone is found in the ore zone, but nearly all of the limestone has been replaced by ore. The minerals include pyrrhotite, pyrite, chalcopyrite, magnetite, garnet, amphibole, and chlorite. The ore near the surface is altered to iron oxide or gossan, which extends to a depth of about

[41] After Emmons, W. H., The Ducktown Mining District, Tennessee: 16th Internat. Geol. Cong. Guidebook 2, 1932, p. 141.

100 feet; below the gossan at the water level is a zone of chalcocite from 3 to 8 feet wide which contains 5 to 25 percent of copper, and below this is the primary copper ore now mined, which carries about 1 to 3 percent of copper.

Ross, after studying and comparing the copper deposits of the southern Appalachian region, concluded that the carbonate including the calcite at Ducktown is of vein origin and not unreplaced limestone and that the sulphides have replaced schist.

DOMINION OF CANADA

The copper deposits of the Dominion of Canada may be considered in three groups: Those of Quebec, Ontario, and Manitoba in the Canadian shield, of pre-Cambrian age; some in eastern Quebec of Paleozoic age; and those of British Columbia on the west coast, of Mesozoic and Cenozoic age.

The pre-Cambrian deposits of the Canadian shield area are thought to be of at least two ages—early pre-Cambrian, which includes the copper-zinc deposits of Manitoba and the copper-gold deposits of Quebec, and late pre-Cambrian, which includes the copper-nickel deposits of Ontario and the native copper of the Lake Superior area.

The geological relations of the ore deposits of the Canadian shield and a summary of the geology of individual districts have been given by E. L. Bruce [42] and by Frederick J. Alcock.[43]

QUEBEC

Eastern Quebec contains numerous sulphide lenses replacing schists probably of Paleozoic age, similar to those of Vermont and New Hampshire. These yielded copper ore in the early days and were later mined for sulphide with copper as a byproduct of sulphuric-acid manufacture. Perhaps the oldest and best known is the Eustis mine.[44]

ROUYN DISTRICT

In recent years the Rouyn district has become the most important copper producer in Quebec. The deposits are probably closely allied to the neighboring gold deposits of Kirkland Lake and other Ontario districts. The rocks are Keewatin lavas and later Timiskaming beds which have been intruded by granitic rocks of varying composition and by diabase. The deposits are replacements in shear zones in the Keewatin lavas.

The ore minerals are pyrrhotite, pyrite, magnetite, sphalerite, and chalcopyrite in a gangue of quartz, chlorite, sericite, amphibole, and cordierite. Gold is an important constituent of the ore.

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[42] Bruce, E. L., Mineral Deposits of the Canadian Shield: Toronto, Macmillan Co., 1933, 428 pp.
[43] Alcock, Frederick J., Copper Resources of the World, vol. 1; Copper in Canada: 16th Internat. Geol. Cong., 1935, pp. 65–136.
[44] Canadian Mining Journal, Canada's Oldest Copper Mine: Vol. 52, 1931, pp. 571–576. Alcock, Frederick J., work cited, p. 75.

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SUDBURY, ONTARIO

The Sudbury deposits not only furnish the world's largest supply of nickel but also yield a large amount of copper. Opinions still differ as to the geology of these deposits, although the subject has been studied long and carefully.

The oldest rocks exposed are Keewatin greenstones and quartzites of Archean age, which are overlain by conglomerates, tuffs, and slates of upper Huronian age. Between these has been intruded a sheet of igneous rock known as the nickel eruptive. This, as described by Collins, is composed of three parts. The lower norite consists of about equal quantities of feldspar and pyroxene more or less changed to hornblende, primary hornblende, flakes of biotite, some grains of quartz, and infrequent particles of black iron ore and yellow sulphides. The norite gradually gives place to the transition zone, a dull salmon-pink feldspathic rock of syenitic aspect with 250 to 800 feet of outcrop. The transition rock in turn gives place to the upper micropegmatite, an "acid-edge" rock which is dull salmon-pink and is composed mainly of feldspar with scattered crystals of a dark mineral and grains of quartz.

The basal norite occupies 34.5 percent of the area of the "nickel eruptive", the transition rock 2.9 percent, and the micropegmatite 62.6 percent. In addition to these main divisions are the "offsets" or dike-shaped protrusions from the outer edge, or base, of the "nickel eruptive" into the underlying rocks.

Collins describes the Fay offset as an example. Its base at the nickel eruptive is about 1,200 feet wide but tapers quickly to 400 feet; thence for 6 miles its course is somewhat crooked, with occasional angular turns. It branches at one place, and protrusions ramify crookedly for a few feet or yards. It consists of a breccia of older rocks cemented by offset igneous rock. The offset rock ranges in texture from glassy, in dikelets and near contacts, to fine-grained and is of dioritic composition.

Collins had considered three possible modes of formation of the nickel eruptive, all of which have advocates (namely (1) that a single intrusion differentiated during cooling, (2) that the norite and micropegmatite are two separate intrusives, and (3) that the upper part of a single intrusive assimilated overlying sedimentary rocks). He concludes that the original magma of the nickel eruptive separated while in the liquid state into norite and micropegmatite magma and that near the end of the process some of the norite magma escaped into the cracks in the floor to form the basic offset.

The broader structure of the area is a spoon-shaped syncline about 36 miles long and 16 miles wide, on which are superimposed many minor structures. The ore deposits are grouped near the base of the intrusive and are most abundant along the south margin. Many of the large deposits are in the offset dikes; some are directly connected with the norite, others, such as the Frood deposit, have no known connection. The deposits occur as roughly tabular or pipelike deposits at the contact of the norite with the underlying rock and in the offset dikes. The ore, much of which is massive sulphide, comprises pyrrhotite, pentlandite, and chalcopyrite with some magnetite, pyrite, and sphalerite and, in places, sperrylite.

Theories of the origin of the Sudbury ores range from gravity

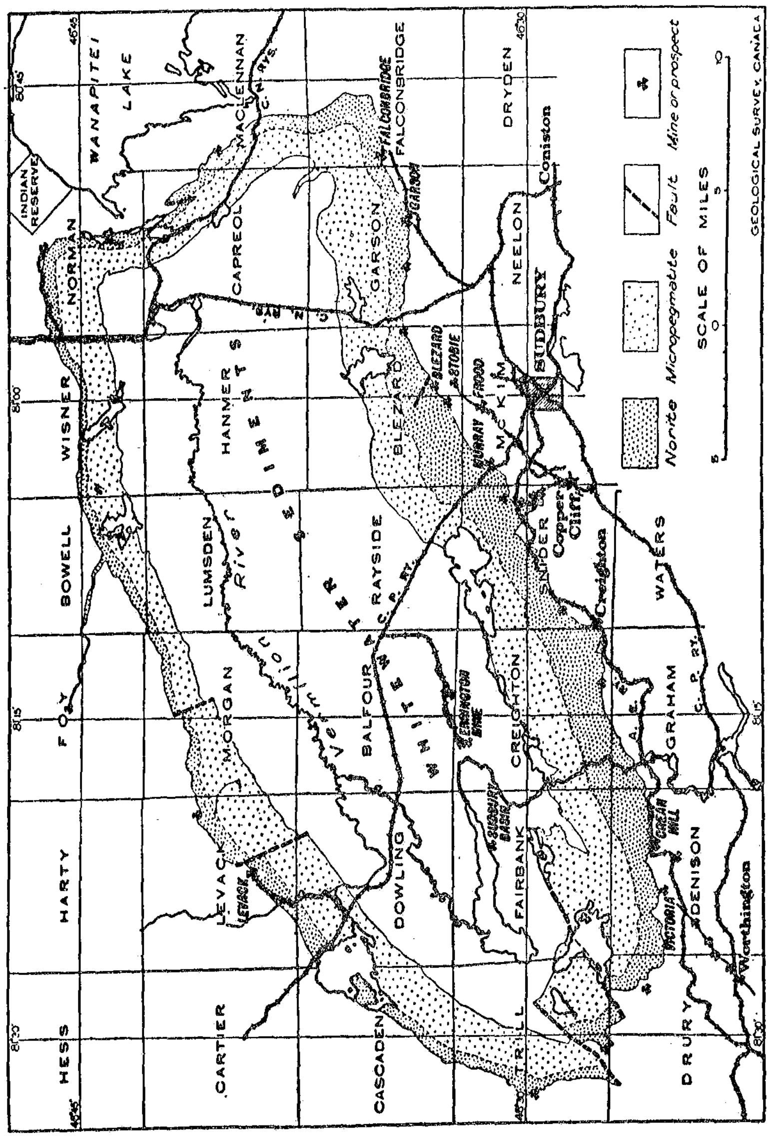

FIGURE 12.—Geology of the Sudbury Basin, Ontario.

separation of liquid sulphide to hydrothermal replacement. As competent students of the deposits are unable to agree as to their origin, the question must remain open, regardless of how strongly the relations seem to suggest ordinary replacement. The geology of the Sudbury Basin is shown in figure 12.[45] A vertical section through the Frood ore body is shown in figure 13.[46]

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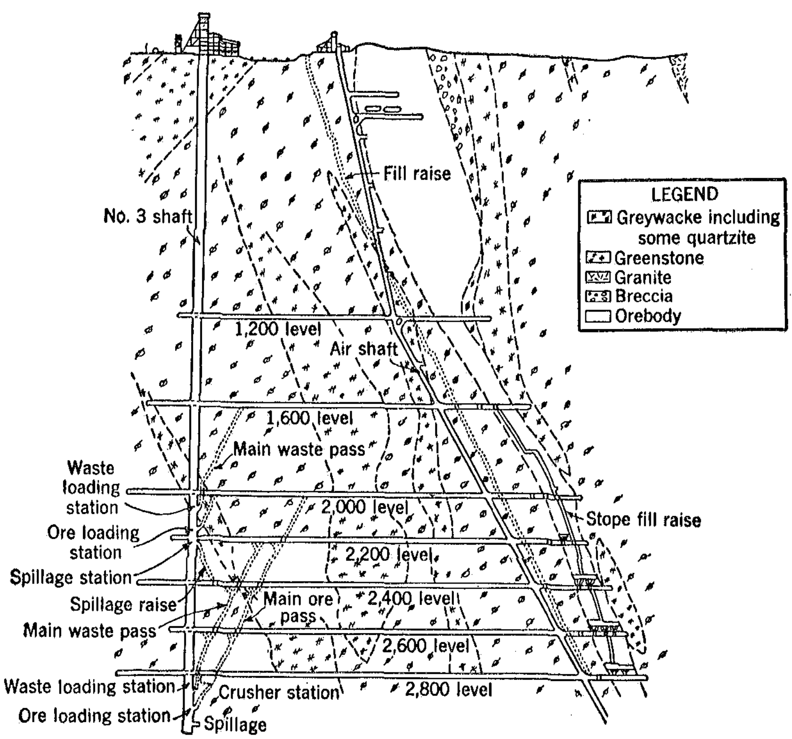

Figure 13.—Vertical section through Frood ore body, Sudbury, Ontario.

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[46] After Mutz., Herman J., Mining the Frood Ore Body at Depth: Eng. and Min. Jour., Nov. 10, 1930, vol. 130, p. 450.

MANITOBA

In recent years there has been much prospecting and development in western Manitoba, and three mines—the Mandy, the Flin Flon, and the Sherritt-Gordon—have produced copper, zinc, and precious metals. The Mandy mine is 3½ miles southeast and the Sherritt-Gordon 40 miles northeast of Flin Flon.

FLIN FLON MINE

The rocks around Flin Flon comprise a volcanic series including both basic and acidic extrusives and intrusives overlain by a sedimentary series of arkoses and conglomerates. These rocks were intruded by basic dikes and small irregular bodies followed by batholithic intrusions and dikes of granitic rock. Later, basic dikes cut the granitic rocks. All are of pre-Cambrian age.

The ore body is a fairly regular lens in greenstone that has been derived from basic lavas. The principal ore minerals are pyrite, sphalerite, and chalcopyrite. The ore is of two classes—massive sulphide, which forms the central portion of the lens, and disseminated ore or schist impregnated with ore minerals, which partly surrounds the massive sulphide ore but is most abundant on the footwall. The disseminated ore is much richer in copper than the massive ore, and both contain gold and silver.

MANDY MINE

The ore body of the Mandy mine is also a lens in a band of schist with greenstone on either side. The location of the lens is thought to be controlled by a drag fold. The central part of the lens consists of high-grade chalcopyrite surrounded by sphalerite and pyrite.

SHERRITT-GORDON MINE

The rocks at the Sherritt-Gordon mine comprise a series of lavas and sedimentary rocks overlain by sedimentary gneisses, of which the whole is cut by basic and granitic intrusives. The ore body is in a narrow zone of fractures and sheared quartzite gneiss that has been impregnated and replaced by pyrrhotite, sphalerite, chalcopyrite, and chalmersite.

ORIGIN OF DEPOSITS

All three deposits are regarded as replacements along fracture or shear zones. The source of the mineralizing fluids is generally thought to be the granitic magma, although the basic magmas are recognized as possible sources.

Numerous deposits similar in geologic occurrence to those mentioned are known to exist in the Manitoba copper-zinc district and have been explored and developed to varying degrees.

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BRITISH COLUMBIA

British Columbia until recently has been the most important copper area of Canada. The geological history of the region is summarized by James as follows:[47]

> The Coast Range batholith extends for 1,000 miles along the west coast of Canada and in Alaska. Along the flanks of this complex of intrusives and at intervals within the outlines of the main mass, thick series of volcanic rocks and variable amounts of normal sedimentary material are found. These are earlier than the batholith and have been disturbed and metamorphosed during the batholithic intrusion. Some of these rocks are of Paleozoic age, but the great bulk of the prebatholithic volcanic rock is early Mesozoic age. They are dominantly basic volcanic rocks consisting of flows, agglomerates, tuffs, and sills, but normal sedimentary rocks are represented in variable amounts. Following this period of extensive volcanicity is the period of the batholithic intrusion, which generally has been dated as Upper Jurassic. The history of the main Coast Range batholith region has been chiefly one of erosion since this intrusion, and the core of the range has supplied great quantities of material to the basins on either side. By the end of Cretaceous time the range was probably reduced to a peneplain, only to be reëlevated during the Laramede revolution. Small granitic stocks along the east flank of the main batholith are generally assigned to this period of mountain building. Great thickness of Tertiary rocks accumulated, in places largely volcanic. On Vancouver Island the Tertiary beds were highly folded at the beginning of Oligocene time and intruded by basic and granitic stocks.

Schofield has given the distribution of mineralization around the batholith as follows:[48]

> There are two main mineral belts in British Columbia separated from each other by an elongated and curved area of granite batholiths belonging mainly to the early part of the Mesozoic area. This mass includes the Coast Range batholith and the majority of the batholiths occurring in the southern part of British Columbia. The belt which follows along the Pacific coast, including the island fringe on the western size of the Coast Range batholith, may be called the Pacific mineral belt; that along the east side of the same batholith, the Interior mineral belt. It will be remarked that the two belts differ in the mineralogical composition of their ore bodies. The ore deposits of the Pacific belt are sought mainly for their copper content; those of the Interior belt are sought mainly for their gold, silver, and lead content. It is only necessary to mention Anyox, Marble Bay, Quatsino Sound, Sunlach, and Britannia of the Pacific belt and Salmon River, Premier mine, Bear River, Alice Arm, Dolly Verdin mine, Hazelton (except Rocher de'Boule), and the deposits of the Ootsa Lake and Whitesail Lake areas to be convinced of the truth of the above statements.
>
> The Pacific and Interior mineral belts strike in a northwesterly direction parallel to the coast, but in the neighborhood of Vancouver they turn eastward, conforming in general with the contact of the batholiths of southern British Columbia. To avoid confusion it is suggested that the northerly belt be called the Kootenay belt and the southern one the Boundary belt. The metal characteristics of the Pacific belt predominate also in the easterly extension of the Boundary belt; the same is true of the Interior belt in the easterly extension of the Kootenay belt. The two belts merge into one in the area bordering the southern part of Kootenay Lake. The Boundary belt includes Highland Valley, Copper Mountain, Phoenix, Deadwood, and Rossland, and the Kootenay belt embraces the Ainsworth, East Kootenay (Sullivan, North Star, and St. Eugene), Slocan, Lardeau, Revelstake, and Stump Lake mineralized areas.

[47] James, H. T., Britannia Beach Map Area 13 C: Canada Geol. Survey Mem. 158, 1929, p. 6.
[48] Schofield, S. J., Ore Deposits of British Columbia: Canada Geol. Survey Mem. 132, 1922, pp. 63-64.

Numerous copper mines and prospects occur throughout the Pacific and Boundary belts. The geology of some of the more important districts is outlined in the following paragraphs.[49]

ANYOX DISTRICT (PORTLAND CANAL AREA)

The Portland Canal area is on the east side of the Coast Range batholith and includes a variety of deposits. The Anyox center has yielded chiefly copper, silver, and gold. The Hidden Creek mine has been by far the most productive.

The rocks of the district are mainly the Anyox amphibolite, which intrudes argillites of Hazelten (Jurassic) age. The Hidden Creek ore bodies (fig. 14) [50] are mainly but not entirely on or near the contact of

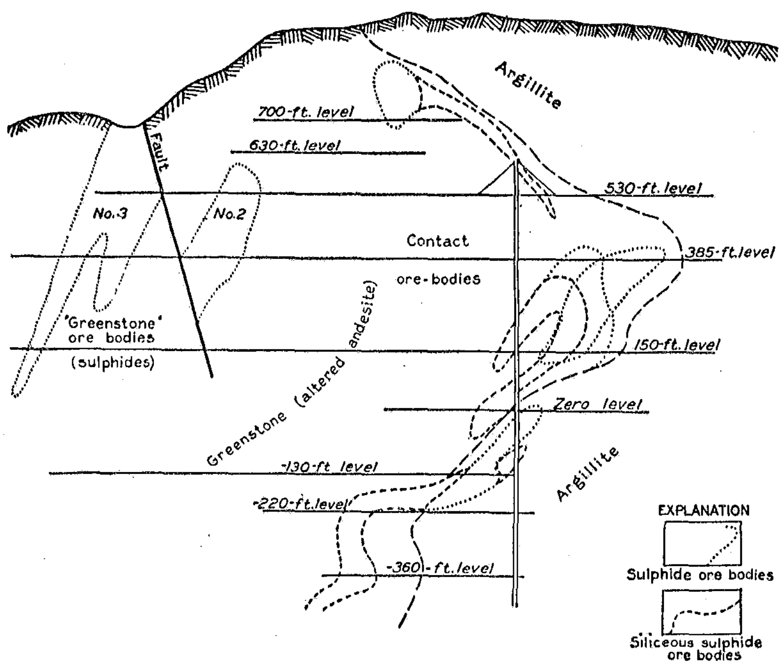

FIGURE 14.—Generalized vertical section, Hidden Creek mine, Anyox, British Columbia. (After British Columbia Bureau of Mines, 1930.)

the argillite and amphibolite. They are irregular replacement bodies, which are as much as 1,000 feet long and deep and 200 feet wide, and are accompanied by strong silicification. The ore ranges from solid sulphide to partly replaced amphibolite and sulphide in silicified rock.

The metallic minerals are chalcopyrite, pyrrhotite, pyrite, magnetite, arsenopyrite, and sphalerite. The gangue is amphibolite, quartz, calcite, epidote, and biotite. The average ore contains 1.5 percent copper and about one-third ounce of silver and 10 cents in gold per ton.

[49] Alcock, F. J., Copper Resources of the World, vol. 1, Copper in Canada; 16th Internat. Geol. Cong. 1935, pp. 113–136.

[50] From Alcock, F. J., Copper Resources of the World, vol. 1; Copper in Canada: 16th Internat. Geol. Cong., p. 122.

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BRITANNIA BEACH AREA (HOWE SOUND DISTRICT)

The Howe Sound district is on the west side of the Coast Range batholith in the "copper belt."

The rocks represent the batholithic intrusives and roof pendants of the intruded rocks. The earliest rocks, probably of Triassic age, are sedimentary and volcanic. The sedimentary rocks range from quartzite to slate, with which are interbedded tuffs and basic sills or flows; above this layer is a great series of prevailingly volcanic rocks, including tuffs, flows, and sills, of which the latter are chiefly basic. These earlier rocks were intruded first by the Britannia sills of varying intermediate composition, then by the main Coast Range batholith of quartz diorite and granodiorite, then by basic and acid dikes, and still later by basalt dikes.

The large deposits are in the Britannia shear zone—a belt of highly sheared rock about 5 miles long and 2,000 feet wide which strikes northwest and dips 70° south. The western 7,500 feet of the zone is confined to one member of the Britannia sills, the mine porphyry. In this section the zone narrows to a blunt edge surrounded by slate, which plunges west at an angle of about 75°. The deposits are in this section. They occur as veins or series of veins, mainly near the footwall of the shear zone.

The ore minerals are pyrite, chalcopyrite, and sphalerite, which are common to all of the deposits. Borite and anhydrite are present in some, and galena and tetrahedrite are present in places. A generalized longitudinal section of the Britannia mine is shown in figure 15.[51]

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BOUNDARY DISTRICT

The Boundary district, including the area around Phoenix and Greenwood, was one of the most important copper districts in Canada during the early years of the twentieth century.

The rocks comprise a series of Paleozoic volcanics with limestones, shales, and argillites. These were intruded by rocks associated with the Coast Range batholith.

After the deposition of Tertiary sediments and volcanics, stocks, sills, and dikes of Miocene age were intruded. The cores of pyrometosomatic or contact type occur as replacements in the limestone. The metallic minerals are chalcopyrite, pyrite, hematite, and magnetite, with abundant andradite garnet, actinolite, and epidote. The ores

[51] From Alcock, F. J., Copper Resources of the World, vol. 1; Copper in Canada; 16th Internat. Geol. Cong., 1935, p. 116.

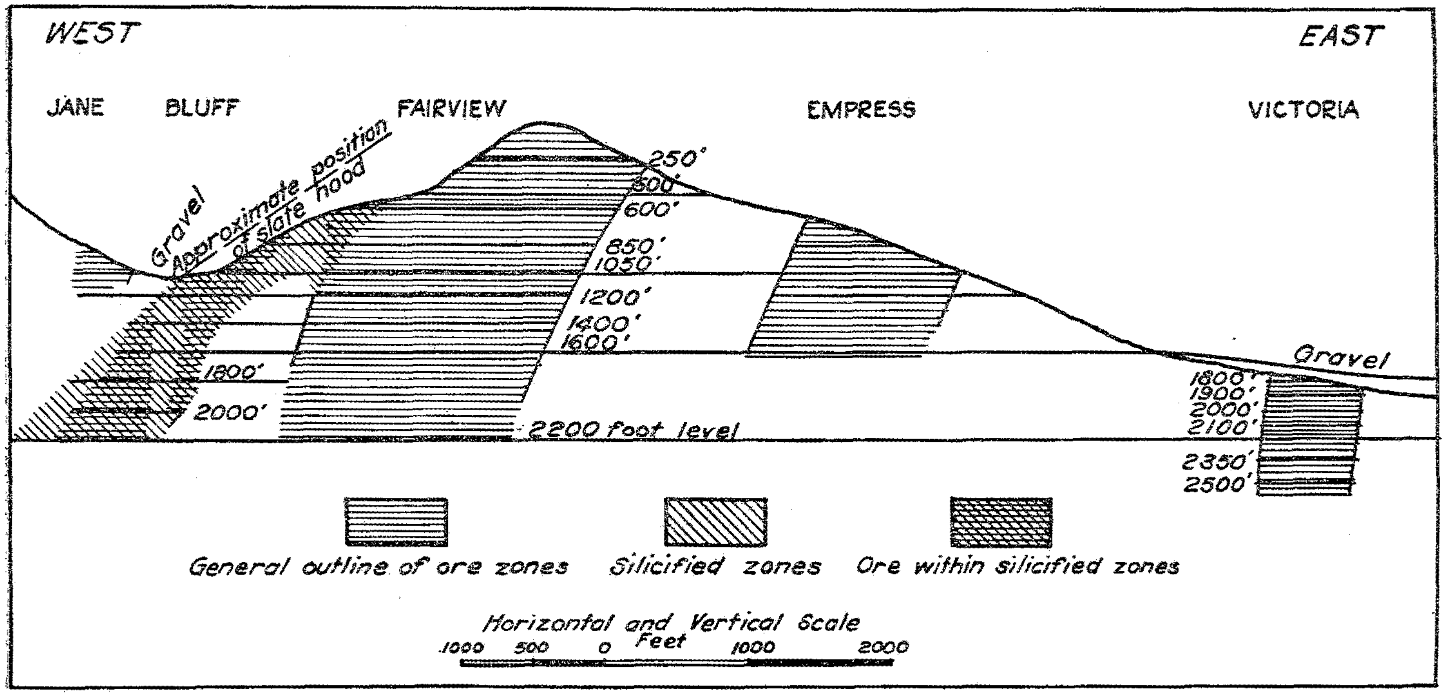

FIGURE 15.—Generalized longitudinal section, Britannia mines, British Columbia. (After Canada Geol. Survey Mem. 158.)

contained from 1.2 to 1.6 percent copper and about $1 in gold and silver per ton. The shoots vary in size; the Knob Hill-Ironside shoot was 2,000 feet long, 900 feet wide, and 126 feet thick.

Le Roy [52] assigns the ores to Coast Range batholith age, although the deposits are not closely associated with the stocks.

SIMILKAMEEN DISTRICT (PRINCETON, COPPER MOUNTAIN) [53]

The Allenby or Copper Mountain mine is located at Allenby, 13 miles south of Princeton.

Paleozoic sedimentary rocks are intruded by a stock of monzonite composition and by later dikes. The ore occurs near the contact of the monzonite and intruded rocks in fractures in both rocks. The ore consists of bornite, chalcopyrite, pyrite, magnetite with galena, and sphalerite in a complex silicate gangue. Ore also occurs in zones of fracture in the intrusive rock.

TRAIL (ROSSLAND) DISTRICT

Paleozoic rocks containing much volcanic material were intruded by the Coast Range batholithic rocks. After uplift and erosion at the close of the Mesozoic period, Tertiary volcanism resulted in additional extrusion and intrusion. The ore deposits occur in fissure or shear zones, mainly in the intrusive rocks or between two intrusive bodies.

The ores range from massive sulphide, mainly pyrrhotite and chalcopyrite with some pyrite, to gold-quartz veins with little sulphide.

Drysdale [54] attributes the mineralization to early Cretaceous and Tertiary time which followed the two main periods of intrusion. The gold is thought to have been introduced largely in Tertiary time. Much of the ore had its largest value in gold, but the district has been an important producer of copper.

MEXICO

Copper occurs in many districts in Mexico,[55] but three districts have produced large quantities—the Cananea and Nacozari districts of Sonora and the Boleo district of Lower California.

SONORA

The districts of northern Sonora are closely allied geologically to those of the southwestern United States; the Cananea district is only 50 miles south of Bisbee, Ariz.

CANANEA DISTRICT

The following description is condensed from Perry.[56]

Paleozoic quartzites and limestones are capped by volcanic extrusives, and the whole is invaded by dioritic to monzonitic intrusives.

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[52] Le Roy, O. E., The Geology and Ore Deposits of Phoenix Boundary District, British Columbia: Canada Geol. Survey Mem. 21, 1912, 110 pp.

[53] Young, G. A., Geology and Economic Minerals of Canada: Canada Geol. Survey, Econ. Geol. Ser. 1, 1926, p. 170.

[54] Drysdale, C. W., Geology and Ore Deposits of Rossland, British Columbia: Canada Geol. Survey, Mem. 77, 1915, no. 64, Geol. Series, 317 pp.

[55] Santillan, Manuel, Copper Resources of the World, vol. 1; El Cobre en Mexico: 16th Internat. Geol. Cong., 1935, pp. 379–409.

[56] Perry, V. D., Applied Geology at Cananea, Sonora; Ore Deposits of the Western States: Am. Inst. Min. and Met. Eng., 1933, pp. 701–709; Copper Resources of the World, vol. 1; Copper Deposits of the Cananea District, Sonora, Mexico: 16th Internat. Geol. Cong., 1935, pp. 413–418.

The mineralized zone follows the trend of the intrusive bodies, which is that of a regional axis of uplift along which Paleozoic sedimentary rocks are raised relative to the volcanic rocks that flank the uplift.

The ore deposits are of several types:

1. Limestone replacements along bedding, in which the ore occurs as irregular masses in partly or completely garnetized limestone.

2. Limestone replacements along contacts. Steeply dipping contacts where impervious rocks overlie the limestone are favorable for replacement deposits in limestone.

3. Disseminated deposits. Fractured areas in intrusive or volcanic rock have been mineralized and raised to ore grade by supergene enrichment. Such deposits surround plugs and overlie breccia pipes.

4. Breccia pipes. The most important ore bodies are in pipes. To the present known depth, 1,600 feet, they are isolated, sharply defined breccia pipes, usually oval or circular in plan, and not connected with through-going fissures. They were localized, however, by fractures.

The pipes are of three types: Capote, Duluth, and Colorada. The Capote pipe is nearly vertical, is oval in plan, and has the ore scattered as irregular masses through it. The pipelike character which prevails in hard, brittle rock spreads out in soft overlying limestone; this gives the whole a mushroom shape of which the top is a limestone replacement and the stem a jumble of brecciated rock cemented and replaced by ore. The Duluth pipe is an oval ore ring 1,200 by 250 feet in brecciated rock encircling a weakly brecciated and slightly mineralized interior mass. It has been developed to a depth of 1,400 feet.

The Colorada pipe occurs along the contact of and within a plug of quartz porphyry. The following stages are recognized in the formation of the pipe. The earliest pipe was preporphyry; starting at the 1,300-foot level a secondary pipe developed in the southwest lobe of the earlier one. Porphyry magma intruded into the secondary pipe separated it at the 1,100-foot level into an upper and a lower part. Ore fluids deposited quartz and sulphides beneath the porphyry dam; the dam was eventually pierced, and the mineralization of the upper part took place as a ring or envelope with an apex 500 feet below the present surface; intense alteration of the rock of the core permitted slumping and brecciation of the surrounding rim of ore.

The massive sulphides, as determined by Kelly, comprise chiefly chalcopyrite, bornite, chalcocite, covellite, and molybdenite. Luzonite and tennantite are abundant, especially in the brecciated ores. Pyrite is abundant peripherally.

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NACOZARI DISTRICT

The rocks at the Pilares mine, according to Locke, are flat-bedded latite tuffs 300 to 500 feet thick, which rest on andesitic tuff. No pre-ore intrusive has been positively identified.

A pipelike section, oval in plan and 1,000 by 2,000 feet in area, of these rocks has been lowered so that the rocks in the central portion are several hundred feet lower than the corresponding rocks outside the pipe. The central part of this cylinder is only moderately broken, but the margin is intensely brecciated; the envelope of breccia is some 50 feet thick along the side walls and much thicker around the ends. The ore minerals, mainly chalcopyrite and pyrite, cement and replace this breccia. Lesser deposits occur within the cylinder. Specularite is locally abundant. Secondary enrichment is unimportant below 200 feet. The pipe has been developed to a depth of 1,800 feet.

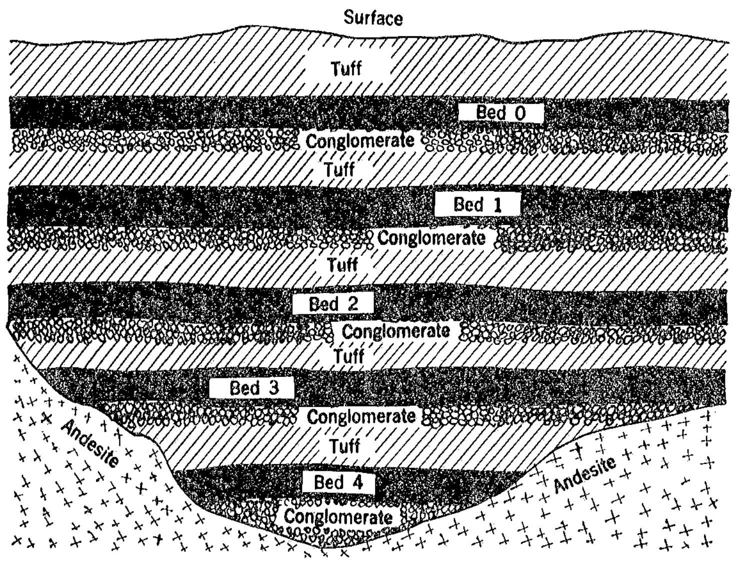

FIGURE 16.—Section of Boleo ore deposit.

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BAJA CALIFORNIA

BOLEO

According to Huttl, the ores at Boleo occur in five separate flat beds, or mantos, which are composed of clayey tuff between layers of coarse conglomerate and tuffs, all of which rest on an andesite floor. (See fig. 16.) [57] Beds 0, 2, and 4, with a copper content of less than

[57] After Huttl, J. B., The Boleo Enterprise: Eng. and Min. Jour., vol. 132, 1931, p. 348.

1.5 percent, are not exploited. Ore mined from beds 1 and 3 comprises various copper minerals; chalcocite and covellite are the chief constituents, and cobalt, zinc, manganese, and nickel minerals are present. The ore bed is soft and resembles shale.

Near the outcrop the ores presented a great variety of oxides and carbonates, but present ores are largely sulphides.

Locke suggests that the deposits may have been formed by solutions traveling upward in crosscutting paths and that the ore bodies may have spread along porous zones from these paths.

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PUEBLA

TEZIUTLAN DISTRICT

The Teziutlan district is near the northern edge of the State of Puebla, Mexico. According to Ernest H. Herivel, the oldest rocks of the district are metamorphosed sedimentary and igneous rocks of pre-Cambrian age. These have been intruded by dioritic rocks and dikes of andesite and bostonite.

The ore occurs as flat-lying lenticular bodies in the metamorphic series. Four lenses have been developed. The ore is massive sulphide, which is composed of chalcopyrite, sphalerite, pyrite, and galena in a gangue of quartz and some mica schist.

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CUBA

The most productive copper deposits of Cuba are at Matahambre in the Province of Pinal del Rio 100 miles west of Havana.

The ores occur in shear zones in shale with interbedded sandstone. No igneous rocks are exposed nearby. The ore is chalcopyrite and pyrite and averages about 5 percent copper.

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PART 3. PROSPECTING, EXPLORATION, DEVELOPMENT, AND MINING

In this paper the term "prospecting" is limited to the search for and discovery of ore bodies at the surface. "Exploration" as used here is applied to the search for new ore bodies and extensions of known ore bodies by surface or underground work. "Development" is applied to the operations involved in preparing a mine for the extraction of ore. "Mining" is limited to the actual extraction of the ore and the necessary preparatory work in stopes.

The various terms, however, cannot be distinctly defined; work done in prospecting may explore an ore body, or adits and shafts excavated for exploration may later be a part of the development plan. Some mining companies make no distinction between exploration and development. Moreover, preparatory work done in stopes may be classed as development at one mine and similar work elsewhere be included under "stoping."

PROSPECTING

Although copper occurs in rocks of all ages prospecting for copper, as for other metals, is more likely to be successful in areas where the rocks and structures favor the deposition of ore bodies of commercial size. Virtually all the principal copper mines are in areas where intensive igneous action has taken place. Sedimentary beds or large masses of granite or allied rocks that contain copper deposits usually have been intruded by later igneous rocks.

Regions in which copper is known to occur are naturally more favorable for prospecting that those in which copper has never been produced. Figure 6 shows the copper-mining districts of North America. The principal copper belt is in the desert area of southern Arizona and adjoining parts of New Mexico and Sonora, usually called the Southwest. A series of districts extends from southern British Columbia to the Copper River in Alaska, and other districts show a general grouping.

The most recent important discoveries of copper in new areas have been in Manitoba and Ontario, where the surface is overlain by swamps and muskeg; a high-grade deposit of copper that did not outcrop also was found in Nevada in 1933.

COPPER MINERALS

Copper is found in nature in numerous minerals and in various combinations with other elements. Only about a dozen copper minerals, however, are commercially important; about six minerals are the source of well over 95 percent of the copper mined in North America. Enough of the minor minerals occasionally are found to be a source of copper in some ores, but usually they are associated with other copper minerals.

Most of the copper of the World comes from the sulphides, of which chalcopyrite is the most widely distributed; in North America, at least, it is, commercially, the most important. Chalcocite is second to chalcopyrite and up to about 25 years ago was the source of more than half of the copper produced. Bornite is important in a few districts. Enargite is an ore mineral at Butte. Enough covellite is also found in some of the mines of Butte to be considered an ore mineral. Tetrahedrite and tennantite are not important minerals of copper but frequently are rich in gold and silver.

Malachite, azurite, and to a smaller extent chrysocolla are mined for copper in the oxidized zones of copper deposits. At present (1935) these minerals are the principal source of copper from the Inspiration and Miami mines.

Native copper is the only copper mineral mined in the Lake Superior district of Michigan. Minor quantities are widely distributed in gossan or the oxidized zones in many districts, but it is commercially important only in Michigan.

Copper deposits rarely consist of a single copper mineral; generally they comprise several copper minerals mixed with a predominance of worthless gangue material.

The ore minerals of copper and their theoretical composition, percentage of copper, hardness, and specific gravity are shown in table 11.

TABLE 11.—*Ore minerals of copper deposits* [1]

Ore mineral	Composition	Copper, percent, when pure	Hardness	Specific gravity
Chalcopyrite	$CuFeS_2$	34. 5	3. 5–4. 0	4. 1–4. 3
Chalcocite	Cu_2S	79. 8	2. 5–3. 0	5. 5–5. 8
Native copper	Cu	100. 0	2. 5–3. 0	8. 8–8. 9
Azurite	$2CuCO_3Cu(OH)_2$	55. 2	3. 5–4. 0	3. 8–3. 9
Malachite	$CuCO_3Cu(OH)_2$	57. 4	3. 5–4. 0	3. 9–4. 0
Chrysocolla	$CuSiO_32H_2O$	36. 0	2. 0–4. 0	2. 2–2. 4
Bornite	Cu_3FeS_3	55. 58	3. 0–3. 5	4. 9–5. 4
Enargite	$3Cu_2SAs_2S_5$	48. 3	3. 0	4. 4
Covellite	CuS	66. 5	2. 5	4. 5
Cuprite	Cu_2O	88. 8	3. 5–4. 0	5. 9–6. 2
Tetrahedrite	$Cu_8Sb_2S_7$	52. 1	3. 0–4. 5	4. 4–5. 1
Tennantite	$Cu_8As_2S_7$	57. 5	3. 0–4. 5	4. 4–4. 5
Tenorite	CuO	79. 84	3. 0	5. 8–6. 3
Atacamite	$CuCl_23Cu(OH)_2$	59. 45	3. 0–3. 5	3. 8
Chalcanthite	$CuSO_45H_2O$	25. 4	3. 5	2. 1–2. 3

[1] Compiled from Dana, Edward S., A System of Mineralogy: John Wiley & Sons, Inc., 1920, 1,500 pp.

Copper is identified in its minerals by dissolving a small amount of the pulverized mineral (as held on the end of a knife blade) in a glass container with hydrochloric (muriatic) acid or if necessary with hydrochloric and nitric acids. The solution is then neutralized with ammonia; a deep blue color indicates the presence of copper. A clean piece of iron placed in the acid solution will have copper deposited on it. A wire moistened and dipped into the powdered mixture will impart a green color to a nonluminous flame if volatile compounds of copper are present; if the mineral is moistened with hydrochloric acid the flame is colored azure-blue.

The principal copper minerals may be identified as follows:[1]

Chalcopyrite—copper pyrites:
Luster—metallic.
Color—bright brass-yellow. Iridescent (changing rainbow colors) tarnish.
Streak—greenish black.
Fracture—brittle, uneven.

May be distinguished from pyrite by the color and streak; pyrite usually is pale yellow to almost white in color and has a black streak. Some chalcopyrite looks like gold, but it can be distinguished from the precious metal by its brittleness; gold is malleable and can be cut with a knife.

Chalcocite:
Luster—metallic or dull when tarnished.
Color—dark lead-gray, frequently tarnished dull black or green.
Streak—lead-gray.

When pure it cuts easily, leaving a polished surface. Smooth conchoidal fracture when compact. Resembles tetrahedrite and tennantite, which, however, have uneven fractures.

Native copper:
Luster—metallic.
Color—copper-red. Easily tarnishes bronze-green to black.
Streak—copper-red.
Fracture—Hackley.
Very malleable, can be cut.

Azurite—blue carbonate of copper:
Luster—vitreous to dull.
Color—dark blue to azure-blue.
Streak—blue.
Fracture—conchoidal, brittle.
Effervesces vigorously in hydrochloric (muriatic) acid.

Malachite—green carbonate of copper:
Luster—silky to dull.
Color—bright emerald green. Often nearly black on exposed surfaces.
Streak—green, paler than the color.
Effervesces vigorously in hydrochloric acid.

Chrysocolla:
Luster—vitreous to dull.
Color—blue to bluish green, black or brown when impure, often spotted with brown.
Streak—white when pure.
Usually adheres strongly to a dry tongue.

Bornite—peacock copper:
Luster—metallic.
Color—dark reddish brown, "pinchbeck" brown on a fresh fracture, speedily tarnishes blue, purple, or iridescent.
Streak—grayish black.
Fracture—uneven to conchoidal, brittle. Often associated or intimately mixed with chalcocite.
The color and tarnish are distinctive.

Enargite:
Luster—bright metallic.
Color—iron-black to dark gray.
Streak—black.
Cleavage—perfect prismatic.
Fracture—brittle.
Only mineral giving above color, streak, and cleavage.

Covellite—indigo copper, blue copper:
Luster—submetallic, resinous or dull.
Color—indigo-blue or darker.
Streak—black (shining).
Cleavage—perfect basal. Thin leaves are flexible.
Turns purple when moistened with water, which is distinctive of a mineral of this character.

[1] Butler, G. Montague, Blow-Pipe Analysis and Crystallography: Handbook of Mineralogy, 1918, John Wiley & Sons, Inc., 155 pp.

Cuprite—ruby copper:
- Luster—adamantine or submetallic to dull.
- Color—various shades of red, sometimes nearly black and often carmine.
- Streak—brownish red, shining.

Tetrahedrite and tennantite—gray copper:
- Luster—metallic.
- Color—steel-gray to iron-black. The latter is apt to be light and may be brownish to reddish.
- Streak—gray, brown, or reddish, the last two being more characteristic of tennantite than of tetrahedrite.
- Cleavage—none.
- Fracture—uneven, brittle. For further identification see chalcocite.

Tenorite (black oxide of copper):
- Luster—usually dull.
- Color—black to iron-gray.
- Streak—black.
- May soil the fingers when earthy.
- It is a common alteration product of other copper minerals, which it coats or with which it is associated.

Atacamite:
- Luster—adamantine to vitreous.
- Color—various shades of green.
- Streak—apple-green.
- Cleavage—perfect.
- Usually transparent to translucent.
- Gives azure-blue flame without the presence of hydrochloric acid.

Chalcanthite—blue vitriol:
- Luster—deep blue, ashy blue, or greenish blue.
- Color—deep blue, sky blue, or greenish blue.
- Streak—white.
- Taste—nauseating and metallic.
- Dissolves readily in water to a blue solution and deposits copper from the solution on clean iron.

Copper minerals cannot always be identified in the field. An analysis is sometimes necessary to make a definite determination. The grade of an ore can only be determined by assay.

The copper minerals found most frequently at the surface are the green and blue copper carbonates and the green silicates. These colors are distinctive; however, a rock with a very small percentage of copper may be colored green. Cuprite, tenorite, native copper, atacamite, chalcanthite, and occasionally other rare copper minerals are found in outcrops; these minerals, together with the carbonates and silicate minerals, are usually formed by the weathering of copper sulphides. Chalcopyrite and, to a smaller extent, bornite may exist in outcrops or in float, but when found at the surface in arid districts usually they are in the form of "kernels" surrounded by alteration products.

Copper sulphides are generally associated with pyrite and other iron sulphide minerals. The iron sulphides oxidize to red, black, or yellow iron oxide; red and yellow, however, predominate.

DEPOSITS

The formations and structures of the copper-mining districts have been discussed in the section on geology. The types of deposits at the individual mines are given later in tables under Mining Methods.

The same general principles are followed in prospecting for copper as in the search for other metals. Vein or lode outcrops are sought, and when found they are examined for evidences of valuable mineral.

Deposits are frequently found by tracing fragments of lode material called "float" to their source.

If an outcrop is leached, underground work will be necessary to prove the value of a discovery. Hidden lodes are sometimes found by careful study of the surface geology. Moreover, a study of the surface oxidation of a lode may indicate the best level for finding hidden ore.

Climatic conditions in the copper districts in North America vary widely. In general, weathering or oxidation of copper minerals in an outcrop takes place above the water level; the zone of secondary enrichment, when one exists, is just below the water level.

Weathering in the arid or semiarid districts reaches a greater depth than in humid regions, hence prospecting must be carried to greater depths to determine the value of a discovery. In the Southwest copper sulphide minerals are seldom found near the surface; the depth may be as much as several hundred feet. At the Kennecott mine in Alaska, on the other hand, chalcocite was shipped from a rock slide, and in Manitoba massive sulphides have been found at the rock surface in swampy regions.

Although surface weathering may have removed all the copper from outcrops of copper deposits, usually some indication of copper in the form of green stain can be found at or near the surface. Following copper stains underground has led to the discovery of many copper ore bodies.

Erosion may have removed all of an ore body except the roots, or, conversely, the present surface may be the same as that when the ore was deposited. Erosion may have kept pace with the weathering of an outcrop, or a shifting water level may have masked the customary effects of oxidation.

Glaciation in the northern districts may have removed all vein outcrops and any oxidized portions of ore bodies coming to the surface.

From the viewpoint of the miner, copper deposits are of four types: (1) Disseminated, (2) replacements in metamorphic rocks or limestone, (3) veins, and (4) beds.

DISSEMINATED

At the disseminated mines (commonly called the "porphyries") the ore deposits are relatively flat lying and are overlain with leached capping (except for the Ajo deposit). The existence of the disseminated deposits at the Utah Copper, Copper Flat, Consolidated Coppermines, Ruth, Chino, Ray, Morenci, and Sacramento Hill mines was first indicated by underground work done in the search for vein deposits. The Ruth ore body, however, was found by sinking a shaft on a meager showing of mineral. The size, grade, and importance of the deposits were determined later by churn drilling supplemented by underground work. The original Inspiration and Miami ore bodies were found by drilling; carbonate ore, however, outcropped at one place on the Live Oak group of claims, which is now a part of the holdings of the Inspiration Consolidated Copper Co. The Ajo ore body outcropped at the surface in the form of a copper-stained hill. It remained unworked until a successful method of leaching the carbonate ore was devised. Sulphide ores are now being mined from below the oxidized horizon.

The ore being mined in the autumn of 1935 at the Inspiration and Miami mines comprised mixed carbonates and sulphides. It is of higher grade than the remaining straight sulphide ore.

Although all of the disseminated deposits now being worked, with the exception of Ajo, were found in established districts, it is possible that other similar deposits exist elsewhere. Moreover, the possibility of finding new disseminated deposits in known copper districts has not been exhausted.

The ore-bearing formation at the Utah Copper, New Cornelia, Copper Flat, Ruth, Consolidated Coppermines, and Morenci is monzonite porphyry. At Chino it is granodiorite and quartz diorite and at Sacramento Hill granite porphyry. The Inspiration and Miami deposits are in schists and sills of granite porphyry. The Ray mine is in schist and porphyry.

At Miami and Ray the best schist ore retains little of its original schistosity. Likewise, at the mines in porphyry the spots showing the most complete destruction of original rock texture are commonly of the highest grade.

The copper minerals at the porphyry mines are disseminated throughout the rock mass. These ore bodies are intensely fractured, and even small fragments of broken ore show evidence of having broken to preexisting planes of weakness. Some of the ore, however, has been silicified since fracturing to such an extent that it breaks into sizable fragments.

One of the products of the oxidation of pyrite in surface ores is sulphuric acid (see p. 48). This acid dissolves the copper sulphides associated with the pyrite; the copper may be carried away entirely but frequently is deposited just below the capping to form a zone of secondary enrichment. At some mines only the enriched portion of a disseminated deposit can be worked at a profit.

As the copper in a capping dissolves its place is taken by iron in the solvent. The dissolved iron may be precipitated, replacing either sulphides or gangue minerals or both. According to Locke [2] the character of the iron precipitates formed in the capping of disseminated deposits during oxidation is a definite indication upon which to predict the existence of hidden copper. The iron tends to precipitate in the place of the sulphides when the ore is rich in copper and away from the sulphides when it is high in pyrite. Indigenous iron oxide is one of the principal indications that copper sulphide was formerly present. It reproduces the exact pattern of the sulphide, assumes the same veinlet and speck shapes, and is similarly related to quartz-lined microvugs and to sericitic rock. It may be a weak, irregular, finely porous body (which suggests the presence of chalcocite in the past); a fine or coarse boxwork which makes a porous pseudomorph of the sulphide (indicating chalcopyrite or bornite); or rigid linings of the insides of voids from which sulphides have been removed (which may mean a mixture of pyrite and chalcocite). Often the transported iron is in the form of microscopic spherulites, disks, or plates in feldspar, sericite, or kaolin. It may occur as holos, crusts, or veinlets and as the earthy or clayey variety. It often forms a paint on the rock and usually is far more conspicuous than the indigenous kind. The formation of iron-bearing quartz veinlets in leached capping some-

[2] Locke, Augustus, Leached Outcrops as Guides to Copper Ore: Williams & Wilkins Co., Baltimore, Md., 1926, 175 pp.

times has indicated pyrite and sometimes copper-bearing sulphide below.

At Ajo the copper sulphides were oxidized to a depth of about 50 feet, but the copper did not migrate. The deposit contained relatively little pyrite at the surface to produce acid by weathering.

The acid resulting from the oxidation of pyrite after the copper is consumed attacks the gangue minerals. The principal reaction is the kaolinization of the silicate rock minerals. Although the presence of any particular gangue does not invariably indicate ore, most of the disseminated-copper mineralizations are accompanied by characteristic finely ground mixtures of kaolin, sericite, quartz, and orthoclase that survive unimpaired in the outcrop. Therefore, an outcrop lacking in these gangue minerals gives less promise of ore.

A gangue of sericite or quartz has little ability to prevent the leaching of copper. A capping containing calcite, however, would tend to prevent the migration of copper, and little leaching would be expected. The cappings over the large disseminated ore bodies are quite free of calcite or other carbonate gangue minerals.

The ores and capping of the disseminated deposits are relatively porous except for case hardening in desert regions, therefore a nonporous capping seldom indicates copper below. The removal of sulphides by weathering may make the capping more porous than the ore.

Relict sulphide minerals in the capping correlate the products of oxidation but may be entirely lacking at the surface. Their presence indicates sulphide minerals below. Although the disseminated ores usually produce enough acid during oxidation to carry away the copper, few cappings of ore bodies are totally devoid of copper. Traces remain behind in a brown, undetermined, and inconspicuous form or as a visible silicate, carbonate, or sulphate. Copper once in an ore almost never wholly escapes, therefore a capping with no trace of copper lends little promise of overlying a body of copper ore.

Areas of iron-stained fractured rock which show evidence of leaching are worthy of investigation, particularly in arid or semiarid regions where copper is known to exist. Detailed geological studies of capping usually will indicate the most promising areas for further prospecting by drilling or underground work.

REPLACEMENTS

Replacement deposits are of two general types—those in schistose rocks and those in limestone. The deposits are formed by sulphides replacing the country rock. The ore bodies at a mine may comprise a single large deposit or a series of smaller ones. Not all ore bodies outcrop, but in most districts at least one comes to the surface. Such deposits may consist of pyrite with a small percentage of copper, as at Ducktown, or principally of chalcocite, as in the main ore body of the United Verde Extension at Jerome.

The oxidized outcrops of replacement deposits comprise masses of iron oxide called "gossan", which usually is harder than the surrounding rock and extends above the surface. Added strength may be imparted to the gossan by silica deposited with the sulphides. Signs of copper usually are evident in the gossan overlying copper ore bodies, but occasionally all copper at the surface has been leached out. Not all gossans, however, overlie commercial deposits of copper; the copper in the deposit below may be too low grade for the deposit to be ore, or copper may be entirely absent. All such outcrops are investigated and explored to such a depth that the character of the deposit below the leached zone can be determined. It may be assumed that nearly all large gossans overlying massive copper deposits have been found and investigated in all parts of the continent except in regions not easily accessible to prospectors.

In the schistose rocks the copper may be leached and carried away, but in limestone the carbonate neutralizes the acid formed by the oxidation of pyrite, and the dissolved copper is reprecipitated in place. Therefore, an outcrop of a deposit in limestone, with an adjacent wall of limestone or a calcite gangue, presumably would still retain its copper. Deeper exploration with the hope of finding high-grade material would not be justified.

A part of the material is carried away during the oxidation of massive sulphides, thereby reducing the original volume. If cavities thus formed are of sufficient size, weak walls will collapse and the overlying ground slump into the opening. At Bisbee, steep, marginal cracks above oxidized ore bodies ascend to the surface and outline an irregularly cracked body of ground which has undergone a slight subsidence. Where the cracks traverse limestone they are cemented with calcite and indicate at the surface the position of oxidized ore to a depth of as much as 700 feet.[3]

Unlike copper, the gold in outcrops does not migrate. It is not dissolved by the products of oxidation of pyrite or by ordinary weathering. As other elements are removed by weathering there is a mechanical concentration of the gold in gossans. Some gossans are worked for their gold content where the underlying sulphides are relatively low in this metal. The Highland Boy mine at Bingham, among others, was first worked as a gold mine. At depth copper was found, and the property has been a producer of this metal for many years. Likewise the first copper veins of Butte were worked for the gold and silver in the oxidized zones. As most copper ores contain some associated gold, this element in a gossan encourages deeper exploration.

The finding of replacement deposits that do not outcrop generally is a function of the geological department at a mine. This work usually comes under the head of exploration. Exploratory work may be directed from geologic deduction or may follow a predetermined geometric plan or both. Favorable areas in the limestone division of the Copper Queen mine at Bisbee were divided into 100-foot squares at 100-foot vertical intervals. Some ore bodies found later within these blocks were not cut by the first exploratory workings.

Geophysical prospecting has been successful in locating bodies of copper sulphide below the surface. This method of prospecting generally is used where the geological formation is known to be favorable and the occurrence of bodies of sulphide is suspected from geological evidence. Geophysical prospecting requires skilled crews and is not in the field of the ordinary prospector.

[3] Locke, Augustus, work cited.

VEINS

A considerable part of a vein outcrop may be barren; mineral indications will occur only where an ore shoot has been exposed at the surface. Vein deposits of copper are of various sizes and shapes and may comprise massive sulphides or copper minerals disseminated in siliceous gangue.

The outcrop of a sulphide ore shoot in an arid region would consist of gossan, as in replacement deposits; the weathering of the top of a siliceous ore body would be similar in some respects to that found in the capping over the porphyry disseminated deposits.

Most of the vein structure may be preserved at the surface after oxidation has changed the general character of the vein filling. This is shown by fracturing within the walls of the vein, by a banded structure, by quartz or siliceous seams and veinlets, by vugs and cavities within the vein, or by combinations of the foregoing. Where the ore shoot comprised principally sulphides, contraction due to oxidation may have caused the hanging wall of the vein to slump at the surface, thereby indicating less than the true width of the vein.

After copper float has been traced to its source, the next step is to determine whether or not the lode or vein is worthy of exploration. Narrow veins in tight formations with a filling of broken country rock showing copper stain, seldom have proved valuable unless they contained gold, silver, or some other valuable metal in addition to the copper. A strong vein in a favorable formation with a copper-stained siliceous filling usually should be explored to ascertain whether or not it contains commercial ore bodies of the disseminated type below the leached zone.

Lode outcrops will extend above the surface of the ground if the vein material is more resistant to weathering than the enclosing wall rocks. Conversely, if the vein material is more easily eroded than the wall rocks the top of the vein may be indicated by a depression.

Outcrops of veins may be obscured by wash or other detrital matter. In searching for a hidden vein, prospectors note the following features which may be caused by the existence of a lode.[4]

1. A natural trench or ditch that does not run directly down the slope of the hill or mountain.
2. A sudden change of slope.
3. A sharp notch crossing a ridge that has a rather uniform altitude on both sides of the notch.
4. Several springs in a line.
5. A sudden change in the kind or quantity of vegetation, which may indicate a contact or, if the change in vegetation is found over a narrow strip of ground, a lode beneath.

Although many other causes may be responsible for these structural features some trenching would be justified if float were found immediately below and not above any one of them.

BEDS

The copper-bearing amygdaloids and conglomerate beds of Michigan are unique both as to type of deposit and form of the copper.

[4] Butler, G. M., Some Hints on Prospecting for Gold; Arizona Lode Gold Mines and Gold Mining: Bull. Univ. of Arizona, Tucson, Ariz., vol. 6, p. 251.

(See p. 68). In places they contained great masses of native copper. Mass copper was found at the surface and led to the discovery of the lodes. Native copper float has been found in other districts, particularly at Kennecott, but in such places the copper was the result of oxidation of copper sulphides.

Copper is mined from a bed at Boleo, Baja California, and on a relatively small scale at the contact of limestone and quartzite beds at Bingham and other western mining districts. At Bingham the copper occurred in the beds contiguous to fissures that cut the formation.

The outcrops of bedded deposits are similar to those in veins except that they may not show the same evidence of fracturing.

FACTORS GOVERNING THE VALUE OF A DEPOSIT

A relatively small percentage of the showings of copper discovered at the surface is developed into commercial deposits. The principal factors governing the value of a deposit are the size of the ore bodies and the grade of the ore. Other important factors are the price of copper, location of the deposit, type of ore and gangue minerals, and regional mining and milling costs. Relatively small bodies of high-grade ore can be mined and shipped at a profit, but ore of milling grade must exist in large enough quantities to justify the erection of a treatment plant if the venture is to prove successful.

The present average grade of the ore at the large porphyry copper mines is a little more than 1 percent copper. The Miami Copper Co. at Miami, Ariz., has mined ore successfully that returned only 12 pounds of copper to the ton when the price of copper was high, but the mining and milling plants were already installed. In general, a large deposit of disseminated copper ore (several million tons) would be valuable if it contained 1 percent or more copper. A vein deposit, however, which contained 1 percent copper could be worked at a profit only under the most favorable conditions and when the price of copper was high. A wide vein of 2-percent copper ore under average circumstances probably could be worked profitably; with copper at 10 cents per pound, however, the minimum grade for profitable operation probably would be about 3 percent. Low-grade deposits usually are milled and must be worked on a large scale to be commercial.

Under average conditions a 6-foot vein should contain a minimum of 3-percent ore and a 2-foot vein, 5-percent ore, to be worked profitably. Where the ore contains appreciable amounts of gold and silver, or other marketable minerals, the minimum grade of copper can be reduced to correspond with the value of the other metals recovered.

Until the past few years copper ore in which most of the gangue was iron sulphide had to be smelted. Iron can now be dropped in the milling circuit by selective flotation and a copper concentrate made. In some sulphide ore, however, the gold and silver occurs with the iron; in such ore, concentration might not be feasible.

Copper ore in which the copper minerals are oxidized must either be smelted or leached. When it is leached any gold or silver the ore may contain is not recovered. Most large bodies of oxidized ore contain some sulphide in addition to the oxidized copper minerals. The sulphide, chalcocite, can be successfully leached, but chalcopyrite and bornite cannot be recovered commercially by this method. For example, a large disseminated deposit containing ½ percent copper as an oxide plus ½ percent as chalcopyrite would be of no value at present.

A relatively low grade deposit near a smelter or a treatment plant might be worked at a profit, whereas the same deposit, if at a remote location, would have no value.

EXPLORATION

Copper deposits are explored by surface cuts or trenching, by underground workings, and by drilling. Trenching is also used in prospecting for vein outcrops and in studying surface geology; frequently important information may be gained more readily in this manner than by costlier underground work. The simplest method of exploring a lode deposit that outcrops is to follow it downward on the dip or to drift on it on the strike where the topography permits. During the early stage of exploration and development too much stress cannot be laid on the desirability of following the ore. Long crosscut adits are seldom justified in exploration. The arguments that such an adit will be useful as a working level and that production can begin as soon as the ore is cut are unsound unless enough ore has already been developed to justify the expenditure.

The so-called porphyry deposits, which yield over half of the copper produced on the continent, were largely explored and sampled by churn drills. The thickness and outlines of the deposits were mainly determined in this manner before mining was begun. Further drilling, however, has shown extensions of the ore bodies as originally outlined in nearly all cases. Sometimes drilling was supplemented by underground working before actual mine development was begun. One of the richest single copper deposits found in recent years, the Colorada at Cananea, was discovered by deep churn drilling. Diamond drilling is used extensively in copper mines in searching for new deposits and charting extensions of known ore bodies. This method of drilling is also used at some mines to determine accurately the size and shape of an ore body before stopes are laid out. Long-hold drilling using heavy air drills with independent rotation and sectional drill steel has been practiced for the last few years. This form of drilling is used mainly for the same purposes as diamond drilling. Drilling methods, the advantages and disadvantages of each method, and costs have been described by Jackson and Knaebel.[5] Drilling is used for sampling the ground penetrated, as well as for finding the position of ore deposits. Sampling and estimation of tonnages or ore deposits also have been covered by Jackson and Knaebel. Drifting and shaft sinking are treated in subsequent sections of this paper.

Exploration is under the direction of the geological and engineering departments at most large copper mines. At small mines, where the ore occurs in a single vein, a foreman with a good "nose for ore" may successfully direct exploration. Where the geology is complex or the operations are extensive, experience has shown that a trained geologist is required to interpret geologic signs properly. At many properties the geologist has been able to find ore bodies that were overlooked in earlier operations and to find other ore shoots that the operating force would have missed without his assistance.

[5] Jackson, C. F., and Knaebel, J. B., Sampling and Estimation of Ore Deposits: Bull. 356, Bureau of Mines, 1933, p. 150.

An important part of the work at most underground copper mines is the search for new ore bodies and the determination of the extent and position of the extensions of known deposits. At operating mines this work usually is classed as development.

GENERAL DEVELOPMENT

Most underground copper mines have grown from small beginnings. Usually it has been impossible to make general plans for development or stoping until the properties have passed the prospect stage. As the scope of operations has expanded, shafts or adits at many mines, although adequate for the early operations, have proved to be bottle necks that limit production. Moreover, new working adits or shafts cannot always be laid out to the best advantage because they must fit existing underground conditions.

A mine operator who knows the position, size, and extent of his ore body before laying out underground development work is fortunate. With such knowledge, the mine can be equipped and laid out at the start for the most economical operation. The low costs at some of the underground porphyry mines would have been impossible without careful preliminary planning.

Access to most underground copper mines in mountainous country at their beginning has been through adits. As the depth of mines has increased, shafts have been necessary in most mines. The Britannia mine in British Columbia is one in which no ore or waste is hoisted; the Coronado mine of the Arizona Copper Co. in Arizona was worked chiefly or entirely through adits. In some mines interior shafts have been sunk from adit levels to work the lower reaches of mines; main working shafts, however, usually are extended to the surface where practical. At mines with interior shafts it is frequently found economical as depth is attained to excavate new working shafts from the surface. Whether hoisting is done to a haulage level or to the surface will depend upon the topography and location of the mine plant.

Deposits worked by underground methods in flat or rolling country are, of course, developed and mined from shafts. New working shafts are often found necessary as the scope of operations increases.

HAULAGE ADITS

Adits used for main working levels usually are larger in cross section than drifts because of the greater traffic through them. Their cross sections range from 7 by 7 feet at some of the small properties to 10 by 12 feet at the United Verde mine, Jerome, Ariz., and the Pilares mine at Pilares, Sonora. Grades usually are one-fourth to one-half of 1 percent outward, which favors loaded ore cars and facilitates drainage.

An early prospect drift can be enlarged without trouble to permit larger cars or even for a double track, but the grade cannot be changed once it is established. Most miners tend to run adits up hill; unless there is constant supervision, instead of one-fourth or one-half percent, an adit is more likely to have a grade of 1 or 2 percent. The steeper grades are uneconomical and dangerous where heavy equipment is used for haulage. An established gradient can be checked inexpensively by using a spirit level and straight-edge in laying the track as a heading is extended. A 0.48-inch block under the outer end of a 16-foot straight-edge will equal a grade of one-fourth of 1 percent.

Drilling and blasting in adits has been discussed by Gardner.[6]

DRIFTS AND CROSSCUTS

Usually crosscuts must be driven from the shaft to the vein or ore body. At some mines the shafts are located so fortunately that these crosscuts are little longer than the requirement for a good shaft station. They are then driven the full width of the station until they intersect the vein, where drifts are turned off in either direction. Other conditions generally prevail, however, and in most mines crosscuts over 100 feet long are required to reach the ore. To obtain minimum average tramming distance at the United Verde mine, the main crosscut on each level is directed toward the center of the ore body as projected downward from the last level. In the Colorada mine, Cananea, Sonora, where the shaft is about 300 feet north of the large, elliptical ore body, a service crosscut is driven south from the shaft and through the ore body; the development laterals are then turned off from it in both directions parallel to the long axis of the ellipse. In addition, on the alternate levels, which are to be haulage levels, a main haulage drift is run close to and back of the shaft and continued in a complete loop around the ore body to intersect the ends of the laterals.

A haulage loop around the shaft, although desirable, is not always justifiable, as it involves considerable extra footage. At the Magma mine the haulage drifts are extended over the shaft pockets and past the shaft itself, far enough to form tail drifts. Where the ore is hoisted in cages no tail room is necessary, but the shaft station must be laid out to provide switching room for loaded and empty cars. Where the hoisting shaft is inclined, the station usually extends over the shaft far enough to provide vertical clearance for a loading pocket. In some small mines, which have an inclined shaft in the lode, cars are dumped directly into skips.

Crosscuts from the shaft to the ore body may be driven larger than ordinary drifts to permit free passage of men and trains or even to allow double tracking; usually, however, they are the same size as standard drifts.

The cross section of exploration drifts ranges from 4 by 6 to about 7 by 7 feet. As shorter rounds are drilled by hand than by air drills, it is practical to carry a smaller heading with hand drilling. Drifts in producing mines usually range from 5 by 6 to 7 by 8 feet in section, depending upon the size of ore cars, the traffic through the drift, and the difficulty of holding the ground. At Pilares, for example, the main motor-haulage drifts are 6 by 8 feet in section; other drifts and crosscuts are 5 by 7 feet.

The most common cross section for haulage drifts is 5 by 7 feet; haulage crosscuts usually are of the same size as drifts, though sometimes larger. The authors believe that it is good practice to make the more traveled drifts and crosscuts, such as those between the shaft and ore body, larger than 5 by 7 feet, particularly from the standpoint of safety. Moreover, a relatively large cross section may be needed for adequate ventilation. In heavy ground small drifts crush less readily than large ones, but if some crushing is inevitable before the

[6] Gardner, E. D., Drilling and Blasting in Metal Mine Drifts and Crosscuts: Bull. 311, Bureau of Mines, 1928. 165 pp.

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useful life of the opening is finished allowance should be made for squeezing and the section chosen should be large enough to permit passage of cars without frequent chopping of posts or caps.

Apparently the location, size, and method of support of drifts bear little or no relation to the general stoping method used. The first essential development opening, after a shaft has been sunk and a crosscut driven to the lode, is an extraction drift. In vertical or steep-dipping ore bodies the most convenient location for this drift is in the ore midway between the walls. If the ore dips more flatly and is not more than about 15 feet wide, the extraction drift is usually located at the footwall. This permits the most efficient placing of ore chutes under the stopes. In a wide ore body more than one extraction drift may be required, as at Cananea where the extraction drifts or laterals are driven lengthwise through the ore body at 40-foot intervals. As the stopes and pillars are laid out at right angles to these drifts, extraction chutes are spaced at 40-foot intervals along the center lines of the stopes, giving a maximum shoveling distance in the stopes of 25 to 30 feet. At Pilares a single drift in the center of an ore body is sometimes sufficient for extraction purposes. At Matahambre, Cuba, a drift approximately in the center of the vein is driven for extraction. However, the character of the ground or other factors may force the location of an extraction drift elsewhere than in the center of the ore. Very often the maintenance of a drift close to one wall of an ore body either in the ore or in the country rock is much cheaper than one in the center of the vein.

Frequently the extraction drift, due to its closeness to stoping operations, is difficult to maintain and must be supplemented for haulage by another drift. Moreover, if more than two or three stopes are being worked along a vein the use of a single drift for tramming may cause congestion. Therefore, main haulage drifts are run in the footwall parallel to the vein and some distance from it as at the Magma mine, Superior, Ariz. At the United Verde mine the sulphide ore is so hard and drifting in it so expensive that extraction drifts are driven in country rock close to the edge of the massive sulphide ore body. If the entire width of the stope cannot be reached economically from a drift in this position, then drifts in ore become necessary in spite of their high cost.

Drifts in the hanging wall are useless for extraction, except in steeply dipping veins. Magma mine uses an extraction drift in the ore on the foot wall, a haulage drift 25 feet away from the ore in the foot wall, and a waste drift in the hanging wall close to the ore for stope filling. In this mine, as at Matahambre, the main haulage drift is run first, then crosscuts are driven from it through the ore body. At the Campbell mine of the Calumet and Arizona Mining Co. the main haulage drift is driven in the country rock of the hanging wall close to the ore, and extraction crosscuts are driven from it through the ore body on the lines between stope and pillar sections. Contrary to the general practice, no drift is run in the ore at this mine.

COSTS OF DRIFTING AND CROSSCUTTING

In the early stages of exploration or development, work may be done by hand. The direct cost of drifting with hand drills is about the same as with power drills, except in very hard rock; the progress, however, is only about one-third as fast.

The total cost per foot of development work under the same natural conditions varies widely. When men are working for themselves or on contract, the direct cost at small mines may be the total cost of running a drift or crosscut from the surface. When work is done on company account supervision must be provided and other overhead expense incurred. Frequently the indirect costs are nearly as much, or more, than the direct costs, especially when only a few headings are being run. For example, the direct cost of a single 5- by 7-foot drift, run from a shaft in 1934 by the American Smelting & Refining Co. at Bingham, Utah, was $15 and the total cost $24 per foot. A heavy pumping expense, tramming the spoil 3,100 feet from the collar of the shaft, and all overhead of the operation were charged against the drift.

The indirect charges are relatively small at mines producing regularly when current development work only is done.

The direct cost of drifting and crosscutting ranges from $3.50 to $15 depending upon natural conditions. The direct cost of haulage adits ranges from $8 to $30 per foot.

Table 12 gives detailed costs of the haulage adit for level 14 at Morenci in 1930. The timber section was 10 feet 6 inches wide at the bottom and 8 feet 6 inches high.

Table 13 gives costs of drifting at eight representative producing copper mines.

TABLE 12.—*Detailed costs of fourteenth level haulage tunnel at Morenci*

	Per foot
Labor:	
Breaking	$3.50
Mucking, by hand	5.10
Timbering	2.40
Spiling	1.05
Ditch, temporary	.50
Ditch, permanent	.75
Track, temporary	.45
Track, semipermanent	.70
Track, permanent	1.00
Trolley beams	.15
Ventube	.15
Total labor	15.75

	Per foo
Supplies:	
Timber	$5.25
Explosives	2.00
Track and supplies, permanent	2.15
Trolley, trolley hangers, bonds, etc	.50
Ventube	.70
Total supplies	10.60
Miscellaneous:	
Tramming	1.50
Hoisting	2.00
Waste disposal	1.00
Ventilation, power	1.00
Mine department expense	1.50
Drills and tools	3.00
Total miscellaneous	10.00
Grand total	[1]36.35

[1] Total cost per foot of completed haulage-tunnel timbered section, February 1930. For mechanical mucking, use $2 per foot for mucking and add $2 per foot for repairs to shovel and $0.40 per foot for double track, making a total difference in the above cost of minus $0.70.

TABLE 13.—*Cost of drifting at 8 copper mines*

Mine	Location	Kind of rock or ore	Size of heading	Number of holes to round	Costs					
					Year	Labor	Supplies	Total direct	Total	Remarks
Miami	Miami, Ariz.	Soft schist	10 by 10		1928	$9.12			$19.95	Timbered.
Do	do	do	4 by 6		1928	2.85			10.70	Do.
Ray	Ray, Ariz.	do	4 by 5		1928	2.22	$1.43	$3.65		Raw.
Do	do	do	6 by 8		1928	4.46	3.83	8.29		Timbered.
Do	do	do	8½ by 9	13 to 15	1928	7.43	7.36	14.79		Do.
Morenci	Morenci, Ariz.	Porphyry	5 by 7		1931	3.82	2.34	6.16		General, 20 percent timbered.
Do	do	do	do		1931	3.24	2.47	5.71		Stope, 37 percent timbered.
Engels	Engelmine, Calif.	Hard diorite	8 by 8½		1929				30.93	Haulage adit.
Do	do	do	6 by 7		1929				13.09	Drifts and crosscuts.
Verde Central	Jerome, Ariz.	Hard schist	5 by 7		1927	6.19	5.51	11.70		Raw.
Eighty-Five	Valedon, N. Mex.	Porphyry	do	13 to 22	1929	4.29	4.33	8.62		Crosscut.
Do	do	do	do	do	1929	5.98	5.13	11.11		Drift.
Teziutlan	Teziutlan, Puebla, Mex.	Metamorphic	do	17	1930 1931	5.49	7.39	12.88	17.48	10 percent timbered.
Sherritt-Gordon	Cold Lake, Manitoba	Sulphide and schist	7 by 8	31	1931	4.81	7.53	12.34	13.92	Untimbered.
Do	do	do	7 by 15	42	1931	8.21	10.89	19.10	21.46	Do.
Do	do	Schist	4 by 6½	20	1931	3.61	4.40	8.01	8.85	Do.

SHAFTS

Frequently the location and extent of an ore body are not known until the working shaft is sunk, and the shaft may not be placed in the most advantageous position. Where possible a permanent working shaft is sunk in a rock formation that stands well and is free of faults. It should be at such a distance from the ore bodies that it will not be disturbed by ground movement caused by subsequent mining operations yet should be near enough so that crosscuts need not be driven unnecessarily long to reach the ore. Enough room should be available near the collar of the shaft for the necessary surface plant.

The location of mine shafts is important in the development of all deep mines but is often given insufficient consideration. The method of mining enters into this problem insofar as it affects ground movement or subsidence. Knowledge of the mechanics of ground movement is inadequate to permit a confident prediction of what will happen in any given case; this is especially true at a new mine where the characteristics of the ore and country rock are not well understood. With a caving system of mining, such as block-caving or top-slicing, subsidence will commonly take place on an angle as flat as 40° to 45° from the lowest point of caving. Crane [7] has reported movement on angles as low as 35° in some copper and iron mines of Michigan. Careful filling of individual stopes will usually prevent the initial movement that leads to subsidence. Although some settling is likely to occur where large areas are undercut, proper filling will restrict this movement to a minimum.

Shafts are usually sunk in the footwall of a vein, as in this position they are less likely to be damaged by subsidence resulting from stoping. If the vein has a decided dip from the vertical, however, the increasing distance to it with each new level makes the location of a vertical shaft in the footwall impracticable. Where shafts are sunk on or through the vein, pillars must be left to protect them. Frequently these pillars are of inadequate size, especially if they are of high-grade ore; as the pillars crush, maintenance of the shafts becomes costly, and frequently they must be abandoned.

The present tendency in mining is to hoist ore in skips; however, cages may be more satisfactory at mines producing two or more classes of ore and requiring all development waste for filling.

INCLINATION

Inclined shafts are recommended for exploring vein deposits on the dip, but once the ore bodies are outlined vertical shafts are usually preferable.

An inclined shaft has the advantage of requiring less crosscutting to reach a vein at different levels, and it may be the most practical kind for mining relatively flat veins or beds. Vertical shafts, however, have so many advantages that they are now used at most mines. Skips are more easily loaded in vertical shafts, and safety devices are more easily installed and surer in their action. Moreover, hoisting is speedier in vertical shafts. A vertical shaft is easier to maintain in bad ground than an inclined shaft, as side pressure only must be withstood. Skips or cages running on tracks in a shaft cause vibration

[7] Crane, W. R., Subsidence and Ground Movement in the Copper and Iron Mines of the Upper Peninsula, Michigan: Bull. 295, Bureau of Mines, 1928, 66 pp.

of the timbers which tends to loosen the blocking of the sets. Inclined shafts should not be steeper than 80°, as skips tend to jump the track at a greater pitch.

Both track and guides are used in steeply inclined shafts. In some districts vertical shafts have been continued downward as inclined shafts; however, in new shafts this practice has been largely discontinued.

SIZE AND SHAPE

The size and shape of shafts have been discussed by Gardner and Johnson.[8] They state:

> The cross section of a shaft is governed by the use to which the shaft is to be put and the material through which it will be sunk. Its size and the number of compartments should be such that the mine tonnage of ore or waste can be hoisted without difficulty in the time available and that men and supplies can be handled promptly. The time lost by men waiting to be hoisted or lowered at the beginning and end of a shift, waiting for cages to go from one level to another, or waiting for supplies before they can start working may be considerable in a year. Generally where such conditions exist the mine has outgrown the hoisting equipment. Ventilation is another important factor, particularly in hot mines. The shaft should be large enough to carry the amount of air required for adequate ventilation.
>
> The present tendency in shaft practice at large metal mines is to provide compartments for skips holding up to 12 tons, depending on the tonnage handled, and one or two service compartments with cages large enough to hold one to four trucks of timber or other supplies on each of one or two decks. A considerable saving is effected by providing cages on which loaded trucks can be run, thus avoiding the handling of supplies by hand at the shaft. Whether such a cage compartment is justified depends, of course, on the size of the operation. An example of a shaft of large cross section is the No. 7 shaft of the United Verde Copper Co., Jerome, Ariz. (fig. 17), now being sunk for hoisting 3,000 tons of ore per day and handling the men and supplies necessary to mine this tonnage. The ultimate depth of this shaft will be 5,000 feet.
>
> The Frood No. 3 shaft of the Internation Nickel Co., Sudbury, Ontario,[9] which is designed to handle 6,000 tons of ore per day, the men, and supplies, to a depth of 3,000 feet, is about the same size as the United Verde No. 7 shaft but is lined with timber. It is 16 by 28 feet 2 inches outside of timber and contains two 6- by 14-foot cage compartments, two 6- by 6-foot skip compartments, a service cage compartment, and the usual ladder, pipe, and counterweight compartments.
>
> The shaft of the Miami Copper Co., Miami, Ariz., through which 8,000 to 18,000 tons of ore per day have been hoisted since 1921, is of concrete construction, 13 by 16 feet 4 inches, inside dimensions. It contains one 6-foot 6-inch by 13-foot cage compartment, two 5-foot 6-inch by 6-foot skip compartments, manway, pipe, and counterweight compartments.
>
> Where a large tonnage is mined from a relatively shallow depth twin shafts may be used to advantage, as at the Inspiration Consolidated Copper Co.[10] A normal tonnage of 900 tons per hour has been hoisted through these shafts, which have two 5-foot 6-inch by 6-foot skip compartments and a service compartment of the same size. A single shaft at the same property is capable of hoisting 600 tons per hour. This latter shaft has two 5-foot 6-inch by 5-foot 11-inch skip compartments and a 6-foot 9-inch by 12-foot 11-inch cage compartment.
>
> Where operations of a single company extend over a large area, as at Butte, a number of shafts of relatively small cross section may be used. Moreover, where heavy ground is to be expected a number of shafts of relatively small section may be used rather than fewer shafts of large section because of the difficulty of keeping the large shafts open.

[8] Gardner, E. D. and Johnson, J. Fred, Shaft-Sinking Practices and Costs: Bull. 357, Bureau of Mines, 1931, 110 pp.
[9] Brock, A. F., Sinking Frood No. 3 Shaft: Eng. and Min. Jour., vol. 130, Nov. 10, 1930, 443 pp.
[10] Stoddard, Alfred C., Mining Practice and Methods at Inspiration Consolidated Copper Co., Inspiration, Ariz.: Inf. Circ. 6169, Bureau of Mines, 1929, 23 pp.

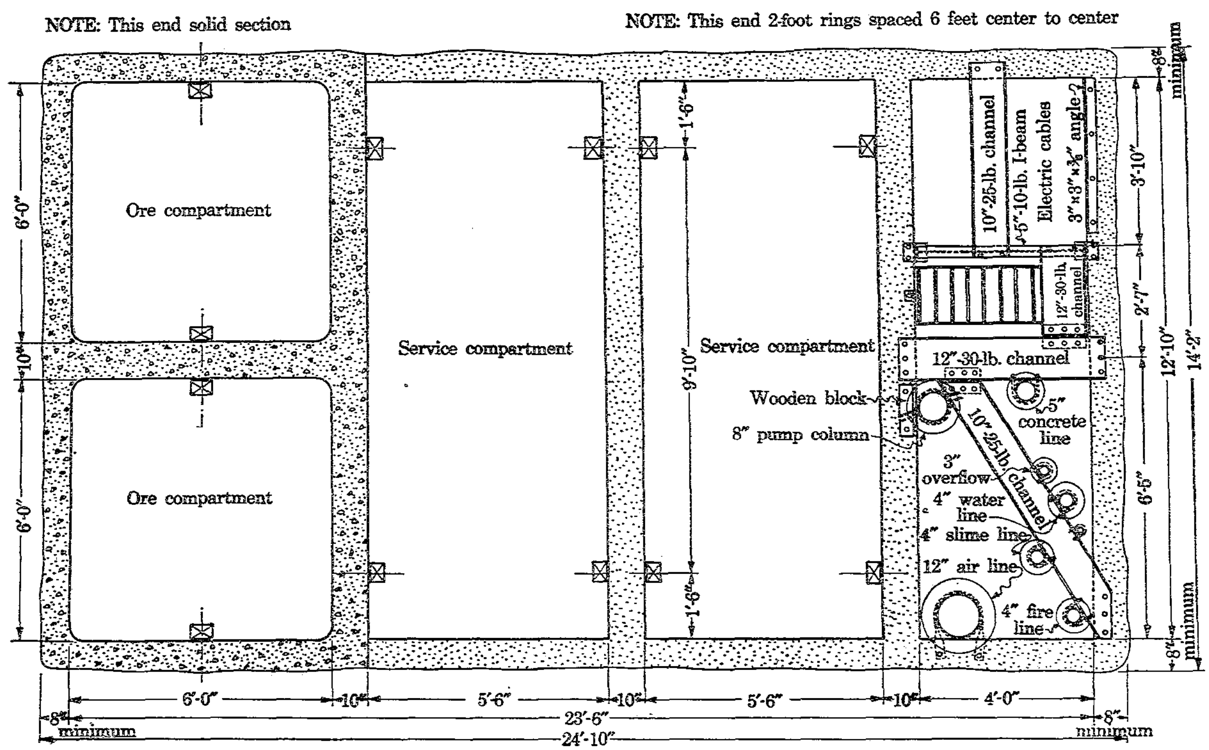


FIGURE 17.—Cross section of United Verde No. 7 shaft.

Shafts for large tonnages, particularly if sunk some time ago, generally contain four or five compartments in a row.

Although the majority of the main working shafts at copper mines are timbered, those sunk most recently have been concreted, and there is a tendency to concrete existing timbered working shafts. Figure 18 shows the framing of sets for a three-compartment shaft at Magma.

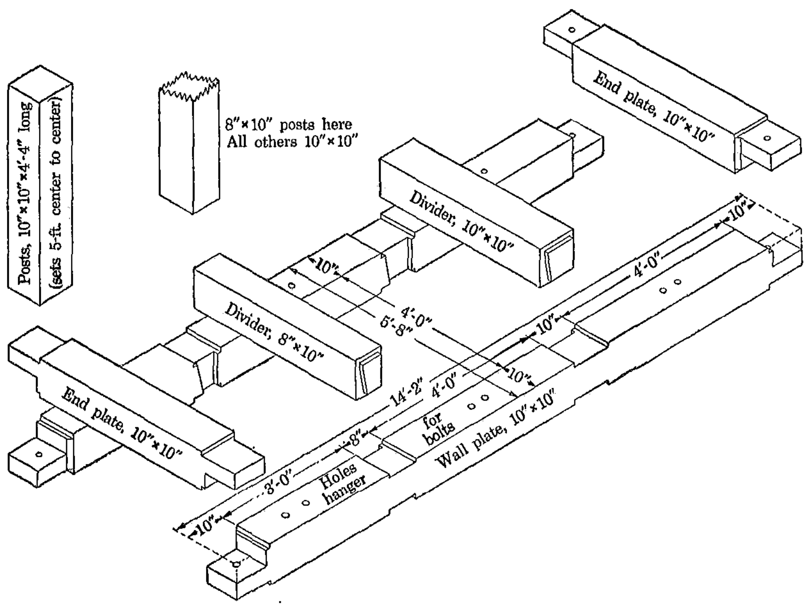


FIGURE 18.—Three-compartment shaft set, Magma mine. (All daps 1 inch. Corners fastened with wooden dowel pins.)

COSTS OF SINKING

Bureau of Mines Bulletin 357 [11] contains detailed discussions of shaft-sinking methods and costs. Table 14 is abstracted from table 4 of the above publication.

The cost of concreting six shafts [12] ranged from $31.85 to $196.61 per foot of shaft. The average cost per cubic yard of concrete was $25.

An interior shaft was sunk in limestone by the American Smelting & Refining Co. at Bingham, Utah, in 1933, a distance of 1,565 feet,[13] at a total cost, including cutting stations, of $117 per foot. The work was done from an adit at a point 3,100 feet from the portal. The rock section was 8 by 16 feet. Sinking operations were hindered after the first 250 feet by heavy flows of water; a maximum of 400 gallons per minute was handled.

[11] See footnote 8.
[12] See footnote 8.
[13] Johnson, J. Fred, Sinking a Shaft and Solving a Pumping Problem: Min. and Met., December 1934, pp. 438–486.

TABLE 14.—*Cost of sinking working shafts at typical copper mines*

Mine	Cananea	Cananea	Magma	Magma	United Verde	Calumet and Arizona	Eighty-Five	Tennessee Copper	Bisbee Queen
Shaft	Capote	Colorada	No. 5	No. 6	No. 5	Saginaw	Jim Crow	McPherson	Do
Location	Sonora, Mexico	Sonora, Mexico	Arizona	Arizona	Arizona	Arizona	New Mexico	Tennessee	Arizona
Year sunk	1925	-	1925–28	1930	1930	1930	1929–30	-	-
Formation	-	-	Sedimentaries, diabase	Dacite	Quartz porphyry	Limestone	Andesite diorite	-	-
Rock section	8-6 by 28	8 by 22	8 by 21	7-6 by 16-6	7-8 by 19-4	9 by 16	6-6 by 16	10 by 19	7 by 17
Footage:									
From	1,400	-	0	0	1,530	915	0	-	0
To	1,612	-	2,531	1,226	3,509	1,752	1,720	-	823
Costs:									
Labor	$61.34	$43.46	$71.26	$57.30	$57.80	$37.22	$39.00	$41.02	$38.61
Supervision	-	-	-	-	10.70	1.82	-	-	-
Supplies	38.85	27.26	28.01	20.75	29.97	22.66	15.17	18.72	20.49
Power	4.78	2.44	5.23	1.23	2.62	1.69	2.45	4.09	[1] 10.05
Total direct	104.97	74.18	104.50	79.28	101.09	63.37	56.02	65.54	69.15
Indirect	17.69	-	61.94	28.34	11.59	29.19	-	-	-
Total	122.66	-	166.44	107.62	112.68	92.56	-	-	-

[1] Includes general costs.

The No. 3 or main working shaft of the Sherritt-Gordon, Cold Lake, Manitoba, was sunk during 1929 and 1930 at an inclination of 51° in schist to a depth of 600 feet. It has four 5-by 6-foot hoisting compartments and a 6- by 6-foot pipe-and-manway compartment.[14]

Table 15 shows the cost of sinking.

TABLE 15.—*Cost of sinking No. 3 shaft at Sherritt-Gordon*

	Per foot	
Breaking:		
Shaft labor (drilling and blasting)	$28.08	
Drill repairs and drilling supplies	4.97	
Explosives	14.24	
Steel and steel sharpening	2.59	
		$49.88
Mucking:		
Shaft labor	35.00	
Surface labor	3.71	
		38.71
Timbering:		
Shaft labor	12.51	
Framing labor	2.61	
Supplies:		
Timber	6.64	
Miscellaneous	2.02	
		23.78
Hoisting		15.17
Supplies:		
Rails	$7.35	
Cables (electric)	3.57	
Pipe lines	5.18	
Miscellaneous	1.86	
		17.96
Compressed air		1.30
Shops: Machine, blacksmith, and carpenter		2.62
Preparatory: Putting in water rings, cleaning out sump, putting pentice under skip compartments, etc		9.61
Mine, general		5.00
Total cost per foot, completely equipped		164.03

[14] Brown, E. L., Mining Methods and Costs at the Sherritt-Gordon Mine: Trans. Canadian Inst. Min. and Met., vol. 36, 1933, p. 476.

RAISES

Raising is an important part of the development at most underground mines. Raises are run for various purposes and under many conditions. They may comprise 4- by 4-foot holes for ore chutes or four-compartment main working entries to upper parts of a mine. After a level is started from a shaft, raises are run to the level above to provide ventilation. Considerable raising is generally required in stope development and stoping routine with most methods of underground mining.

Raises used for ore passes are generally untimbered; after the raise is completed the timber is stripped from it.

According to Mosier and Sherman,[15] experience in the Morenci district has shown that ore transfer raises that run at an angle of 70°

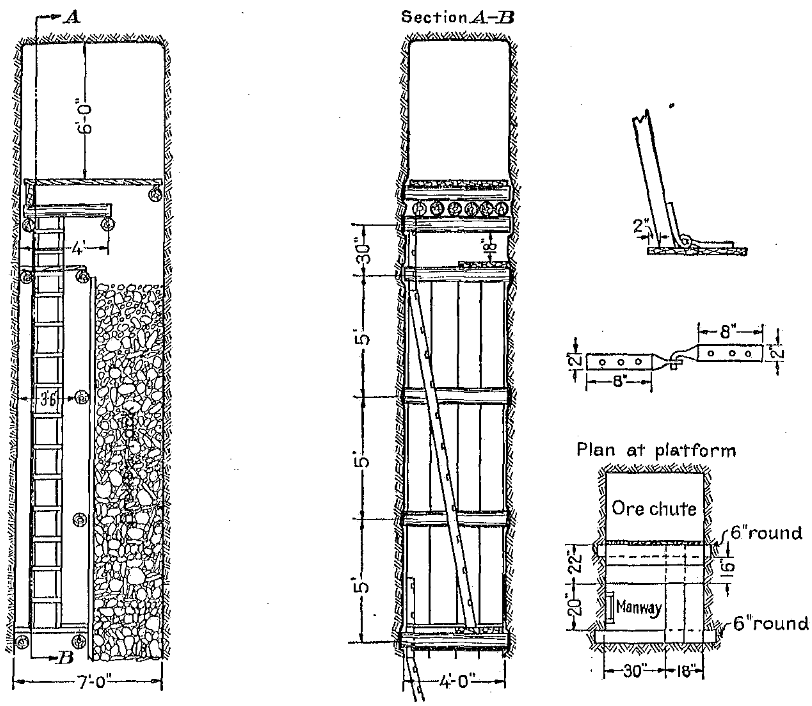


FIGURE 19.—Standard manway stull raise at Morenci.

are the most satisfactory; at this inclination the ore neither builds upon the bottom side of the raise nor packs in the chutes.

Short raises for ore chutes are usually driven with no timber except stulls to support a drilling platform. Although raises up to 100 feet in height are driven in this manner, it is not considered good practice, principally on account of the hazard to the workmen. The simplest type of timber raise is one in which a line of stulls is carried up to provide a base for a partition between a manway and a chute compartment. Figure 19 shows a standard stulled raise.

[15] Mosier, McHenry, and Sherman, Gerald, Mining Practice at Morenci Branch, Phelps Dodge Corporation, Morenci, Ariz.: Inf. Circ. 6107, 1929, 34 pp.

Most raises are timbered with framed sets; the most common form has two compartments with the sets placed on 5- to 7-foot centers and with lagging on the outside of the manway and the inside of the chute compartment.

Cribbed raises are commonly used in heavy ground and for ore chutes in cut-and-fill and shrinkage stopes. Figure 20 shows a cribbed raise used at the Magma mine.

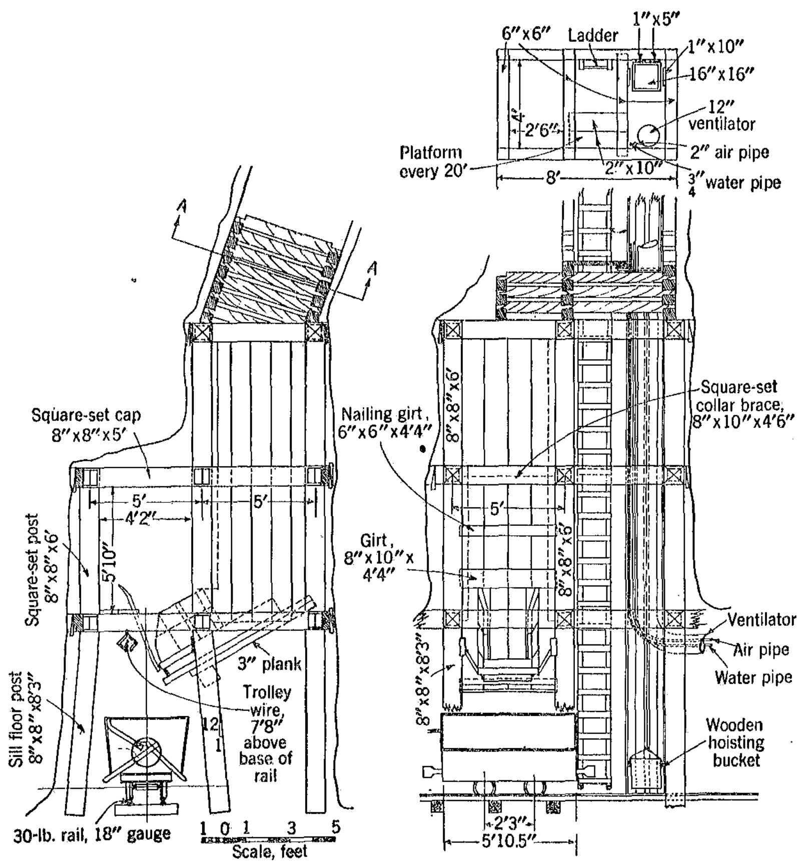


FIGURE 20.—Cribbed raises, Magma mine.

Raise rounds usually are drilled with stopers. The type of round is generally the same as in shafts, except that it is drilled upward instead of downward. The round is usually drilled so that most of the spoil is thrown to the chute side.

All raise rounds should be blasted electrically; this is required by law in most mining districts. Just before blasting, the bulkhead on the chute side is removed and placed on top of the one over the manway. After a blast access to the face is obtained by removing the top of the partition between the two compartments. Good practice dictates keeping the chute compartment full of broken rock or ore to within 6 feet of the top of the timbering.

At many mines pilot raises are run in shaft sinking. After the connections are made the raises are enlarged to full shaft size; the timbering begins at the point of connection and is carried downward.

At Morenci [16] three-compartment raises are run for distances of more than 200 feet. Figure 21 shows the size of the compartments and the timbering details. One compartment is used for holding the broken rock, the middle one is a manway, and the third is used for a light cage to handle men and supplies used in the raise. Stations are cut every 100 feet. As soon as a raise has progressed 30 feet above a substation the sheave for the hoist is raised and hoisting is done to that level.

To reduce the excessive weight of broken rock at the bottom of the chute in a long raise, rock is transferred through auxiliary chutes spaced 100 feet apart.

All raise rounds are blasted electrically; five delays are used. Before loading the holes the miner doing the work obtains the only key to the lock of the blasting switch. After the round is loaded the man in charge connects the lead wires to the blasting line 25 feet from the bulkhead, where a second man is stationed. These men then descend to an interrupter switch at the bottom of the raise. After the approaches to the raise are guarded, the interrupter switch is closed and the round blasted through the main blasting switch.

The back of the raise is never more than 12 feet above the timber. After a round is blasted a 5-foot set is put in place; on top of this a blasting set is installed, then the next round is drilled. The broken rock in the raise is kept within 6 feet of the bulkhead. Just before blasting the trammers draw enough rock from the chute to provide space for the material to be broken by the blast.

COST OF RAISING

The cost of raising varies with the height of the raise, cross section, type of timbering, and hardness, strength, and temperature of the rock.

Usually routine raising in stope development costs about the same as drifting in the same rock. The extra cost incurred in getting men and supplies to the working place and the more expensive timbering offset the cost of shoveling in drifting. In long raises, however, the cost of raising is considerably higher than that of drifting. Costs of raising at five typical mines are shown in table 16.

TABLE 16.—*Raising data at 5 copper mines*

Mine	Location	Kind of rock or ore	Size of heading	Number of compartments	How timbered	Year	Costs			
							Labor	Supplies	Total direct	Total
Miami	Miami, Ariz.	Soft schist	4 by 6	1	Cribbed	1928	$3.45			$8.97
Morenci	Morenci, Ariz.	Soft porphyry.	do	1	Raw	1928	1.56	$1.02	$2.58	
Do	do	do	4 by 7	2	Stulled	1928	3.14	1.09	5.23	
Engels	Engelmine, Calif.	Hard diorite.	6 by 8	2	Sets (20 percent).	1929				18.34
Teziutlan	Teziutlan, Puebla, Mex.	Metamorphic.	5 by 8	2	Stulls and lagging.	1930 1931	4.05	3.52	9.57	14.06
Sherritt-Gordon.	Cold Lake, Manitoba.	Sulphide	5 by 5	1	Raw	1930 1931	4.23	3.78	8.01	9.04

[16] Mosier, McHenry, and Sherman, Gerald, work cited (see footnote 15).

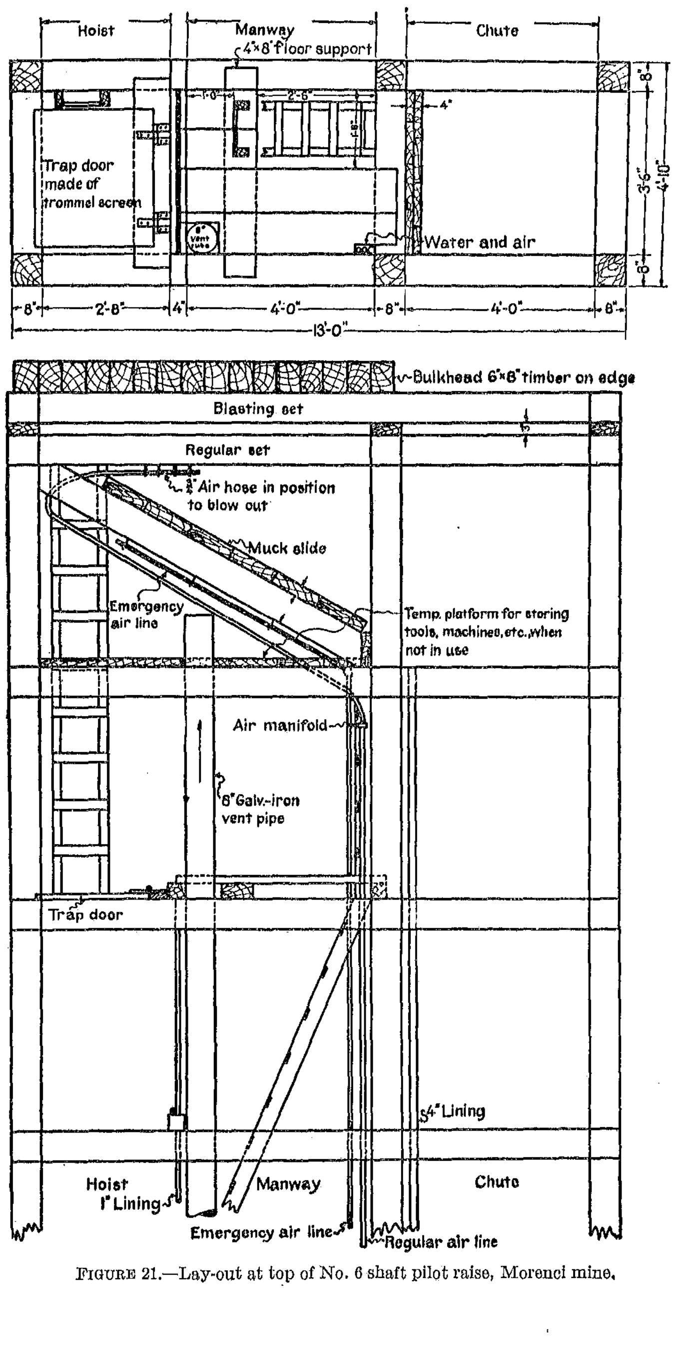


FIGURE 21.—Lay-out at top of No. 6 shaft pilot raise, Morenci mine.

LEVEL INTERVALS

The best interval to employ at any underground mine depends mainly upon geologic conditions at the mine and the mining method used. The cost of haulage and development drifts, shaft stations and loading pockets, and chutes under stopes must be charged against the ore mined. As the level interval is increased, fewer levels are required. The resulting saving, however, must be balanced against the greater cost per foot of driving long raises, the higher cost of maintaining long raises in heavy ground and long ore chutes when the ore is abrasive, and the greater difficulty of ventilating stopes. Where unusually long crosscuts are necessary intervals may be relatively longer and sublevels may be driven. Where the ore bodies are irregular in size and form, levels or sublevels must be relatively close together to prospect the ground and develop the ore properly. Moreover, in such mines short intervals are desirable to obviate the necessity of driving raises through waste to reach the ore.

In large irregular ore bodies and in wide veins weight is likely to develop on the level below or in the stopes before a block of ground is mined out. In such ore bodies a short level interval may be desirable so that a stope may be completed as quickly as possible and long raises need not be maintained in heavy ground. Where the ore bodies are regular, relatively long intervals may sometimes be desirable, even if excessive repair work is required to maintain ore passes and raises.

In the early days of mining, levels were usually run on 100-foot intervals. It has been found, however, that in many places longer intervals are more economical. The interval at some mines has been increased to 150, 200, 250, and finally to 300 feet. In most mines, however, the last two distances are not practical.

A saving in vertical development work is made through the use of sublevels when the level interval is 200 feet or more, as at the Magma mine where alternate levels are fully developed for haulage and ore extraction; the intermediate levels serve for travel, getting supplies into stopes, and ventilation. At this mine rock temperatures are high, and long raises are very expensive to drive. The factors affecting change in level interval have been discussed in detail by Vanderburg.[17] The intervals at various mines are given later in this paper in tables which show comparative data for different mines using the same mining method.

The preceding discussion does not apply to mines that use the block undercut method of mining. The working levels used with this method are not comparable to the level intervals that have been discussed.

RATIO OF DEVELOPMENT TO TONNAGE

The footage of development necessary at any mine will depend largely upon the type of deposit being worked. Table 17, compiled from company annual reports, shows the ratio of the footage of development to tonnage of ore mined at 20 copper mines. The footage of diamond-drill holes is also shown at some of the mines.

[17] Vanderburg, W. O., Factors Governing the Selection of the Proper Level Interval in Underground Mines: Inf. Circ. 6613, Bureau of Mines, 1932, 17 pp.

TABLE 17.—*Ratio of footage of development to tonnage of ore mined at 20 mines*

Company	Mine	Year	Tons of ore hoisted	Development work, feet	Tons per foot	Diamond drilling, feet [1]
Phelps Dodge Corporation.	Copper Queen branch [2]	1923	498,143	36,226	13.8	
		1924	506,302	42,000	12.0	
		1925	618,055	47,800	12.9	
		1926	478,688	39,830	12.0	6,954
		1927	414,954	27,104	15.3	
		1928	360,166	33,830	10.6	
		1929	476,043	[3] 40,748	11.7	7,673
		1930	280,861	22,146	12.7	16,452
		1931	[4] 175,000	14,861	11.8	5,907
		1932	260,284	20,000	13.0	13,600
		1933	324,000	27,900	11.6	6,885
		1934	[4] 512,000	32,046	16.0	10,418
		Average	408,708	32,041	12.8	
	Morenci branch	1923	641,757	26,743	24.0	
		1924	1,088,090	70,182	15.5	
		1925	1,528,277	102,001	15.0	
		1926	1,550,358	69,529	22.3	
		1927	1,463,591	50,637	28.9	
		1928	1,550,804	34,147	45.4	587
		1929	1,738,084	39,716	43.8	Large
		1930	1,321,268	31,627	41.8	2,180
		1931	[4] 1,304,478	19,177	68.0	
		1932	[4] 751,483	0		
		Average	[5] 1,218,671	49,306	27.5	
	Eighty-Five	1928	93,672	9,355	10.0	
		1929	58,722	5,909	9.9	
		Average	76,197	7,632	10.0	
	Verde Central	1928	([6])	8,364		6,182
		1929	93,050	1,487	62.5–9.4	
		1930	92,876	1,367	[7] 67.9	
		Average	92,963	3,739	16.6	
	United Verde branch	To 1931	15,682,947	[8] 341,417	45.9	258,000
Phelps Dodge Corporation (Montezuma Copper Co.).	Pilares	1925	807,113	38,843	20.8	14,728
		1926	764,551	33,793	22.6	10,616
		1927	759,953	23,109	32.9	11,401
		1928	813,416	29,495	27.6	16,218
		1929	835,036	32,836	25.4	18,870
		1930	645,504	33,309	19.4	15,036
		Average	770,929	31,897	24.2	
Phelps Dodge Corporation (Calumet and Arizona Mining Co.).	Calumet and Arizona	1927	482,130	57,224	8.4	
		1928	511,095	61,800	8.3	1,200
		1929	597,497	61,519	9.7	1,429
		1930	427,436	56,112	7.6	
		Average	504,539	59,163	8.5	
Magma Copper Co.	Magma	1929	269,579	18,785	14.3	
		1930	251,872	18,005	14.0	
		1931	231,862	12,109	19.1	
		1932	149,462	8,936	16.7	
		1933	152,866	9,001	17.0	
		1934	264,094	12,801	20.6	
		Average	219,956	13,273	16.6	
		1914–34	3,167,000	220,807	14.3	
Anaconda Mining Co. (Greene Cananea Copper Co.).	Cananea	1927	686,899	44,898	15.3	[9] 5,500
Arizona Commercial	Arizona Commercial	1925	75,499	4,013	18.8	
		1926	65,689	3,484	18.8	
		Average	70,594	3,748	18.8	

[1] Not included in development footage.
[2] Limestone-copper or copper-lead areas only. Excludes porphyry production and footage.
[3] Excludes 10,887 feet of exploration work in Warren Shaft area.
[4] Estimated.
[5] Excludes 1932.
[6] Mill built during year and stoping started. No production.
[7] Latter figure obtained by charging both years' development against 1929 ore.
[8] Mill, C. E., Ground Movement and Subsidence at United Verde Mine: Am. Inst. Min. and Met. Eng. Tech. Pub. 551, 1934.
[9] Approximate. Also 10,800 feet of churn drilling.

TABLE 17.—*Ratio of footage of development to tonnage of ore mined at 20 mines*—Con.

Company	Mine	Year	Tons of ore hoisted	Development work, feet	Tons per foot	Diamond drilling, feet
Miami Copper Co.	Miami	1929	5, 017, 983	131, 360	38. 2	
		1930	6, 124, 993	77, 430	79. 1	3, 422
		1931	4, 438, 808	49, 881	89. 0	
		1932	1, 417, 810	17, 969	78. 9	
		1933 [10]				
		1934	356, 000			
		Average	3, 471, 112	55, 328	62. 7	
Inspiration Consolidated Copper Co.	Inspiration	1928	5, 744, 527	164, 937	34. 8	[11] 6, 618
		1929	5, 799, 587	136, 329	42. 5	[12] 7, 988
		1930	3, 041, 854	68, 286	44. 5	1, 411
		1931	2, 625, 331	75, 742	34. 7	
		1932	463, 709	11, 351	40. 8	
		Average	3, 535, 002	91, 329	38. 7	
Isle Royale Copper [13]	Isle Royale	1926	564, 692	12, 223	46. 2	
		1927	593, 594	12, 881	46. 1	
		1928	560, 598	12, 936	43. 3	
		1929	669, 049	14, 793	45. 2	
		Average	596, 983	13, 208	45. 2	
Anaconda Copper Co.	Butte mines	1926	2, 570, 450	181, 680	14. 1	
		1927	2, 956, 003	161, 940	18. 3	
		1928	3, 013, 495	182, 688	16. 5	
		Average	2, 846, 649	175, 436	16. 2	
International Nickel, Ltd.	Frood [14]	1929	199, 852	50, 460	4. 0	
		1930	902, 531	50, 803	17. 8	
		1931	1, 068, 978	12, 074	88. 5	
		1932	513, 590	10, 337	49. 7	
		1933	952, 725	9, 924	96. 0	
		1934	1, 868, 186	19, 937	93. 7	
		Average	917, 644	25, 591	35. 9	
		Through 1934	5, 505, 862	167, 376	32. 9	
Granby Consolidated Mining & Smelting Co., Ltd.	Anyox	1931	1, 479, 905			
	Bonanza	1931	96, 984			
	Anyox plus Bonanza		1, 576, 889	11, 061	142. 6	16, 739
	Anyox	1929	1, 694, 873	15, 336	110. 5	4, 477
	Allenby	1929	919, 752	21, 814	42. 1	17, 796
	Anyox plus Allenby		2, 614, 625	37, 150	70. 4	22, 273
Noranda Mines, Ltd.	Noranda	1929	422, 331	10, 320	40. 9	31, 291
		1930	849, 308	16, 935	50. 2	50, 965
		1931	1, 012, 005	20, 728	48. 8	45, 948
		1932	1, 218, 295	16, 031	76. 0	48, 650
		1933	1, 541, 524	16, 664	92. 5	52, 639
		1934	1, 777, 021	17, 962	98. 9	49, 947
		Average	1, 136, 746	16, 440	69. 1	
Walker Mining Co.	Walker	1929	513, 526	17, 950	28. 6	
		1930	518, 509	15, 374	33. 7	
		1931	432, 294	6, 997	61. 8	
		Average	488, 109	13, 440	36. 3	
Engels Copper	Engels	1928	303, 301	10, 998	27. 6	
		1929	395, 042	2, 649	149. 1	5, 789
		1930	176, 680	2, 955	59. 8	7, 261
		Average	291, 674	5, 534	52. 7	
Copper Range Co.	Champion and Baltic	1928	345, 312	18, 174	19. 0	
		1929	319, 998	25, 113	12. 7	
		1930	304, 089	19, 178	15. 8	
		1931	404, 830	13, 776	29. 4	
		1932	291, 265	8, 792	33. 1	
		1933	203, 940	11, 266	18. 1	
		1934	241, 175	10, 955	22. 0	
		Average	301, 515	15, 322	19. 7	

[10] Closed down. [11] Also 1,332 feet of churn drilling. [12] Also 400 feet of churn drilling.

[13] Figures are for tons hoisted, of which 20 to 23 percent was discarded in the rock house.

[14] Mine in development stage; from September 1926 to end of 1929, 64,291 feet of development work proved 134,670,000 tons of ore. Of this, 43,562,000 tons below the 1,400 level averages 2.39 percent of nickel and 3.62 percent of copper. The cost of the 1929 development work in the Frood mine was $3,935,094.54, or $78 per foot.

MINING METHODS

CLASSIFICATION OF METHODS

Copper ores are mined by both surface and underground methods. The outcrops of many underground mines were first mined by open-cut methods, and eventually underground methods will be employed for extracting remnants of ore bodies being exploited at present open-cut mines.

Copper is leached from its ores in place at a few copper mines; this practice, however, cannot be classed strictly as a mining method.

SURFACE METHODS

The principal surface method of mining copper is by open pits in which the ore is excavated and transported by machinery. Practices vary at different mines, but there are no general subdivisions of this method of mining copper.

The open-cut method in which ore is broken at the surface and drawn to underground workings for transportation to the surface is called "glory-hole" or "mill-hole" stoping. Usually this is a transitionary method between open-pit mining and underground stoping.

UNDERGROUND STOPING

Underground stoping methods may be classified in numerous ways. Logical classifications which depend upon the manner of support, method of breaking the ground, or sequence of attack have been made and advocated in the literature on mining. Underground mining methods have been classified as follows by the Bureau of Mines. Although the classification is not strictly logical, it has the advantage of being the one usually followed by mine operators.

1. Open stope (including room-and-pillar and sublevel stoping).
2. Shrinkage.
3. Cut-and-fill.
4. Square-set.
5. Block caving.
6. Sublevel caving.
7. Top slicing.

One method of stoping may grade into another, or a combination of methods may be used in the same ore body. Many variations in mining practices exist throughout the industry. Methods and practices at typical mines are discussed later in this paper.

HISTORY OF MINING METHODS

The open-stope method of mining was used by early miners on the continent to follow mineral deposits underground. The method is well-suited to many types of deposits and has been used in the Michigan copper mines to great depths. Up to 1860 most of the mineral produced in North America was from open stopes. Stulls were used for casual support; in some districts additional support was afforded by pillars. As mining was expanded by the discovery of western districts and deeper levels were opened up, the open-stope method proved inadequate for handling ground that required systematic or immediate support. Attempts to mine large ore bodies with weak walls resulted in disastrous cave-ins and loss of life.

This was the situation in 1860 when stoping began in the famous Bonanza ore body of the Ophir mine on the Comstock lode, Virginia City, Nev. This ore body was phenominally rich and eventually proved to be 400 feet high, 90 feet wide, and 320 feet long. The walls and ore would stand over only short spans without support. The best engineering talent of the time was taxed by the problem; it was solved by the development of the square-set mining system by Philip Deidesheimer. The art of mining received a tremendous impetus by this invention. Most mining methods have been evolved slowly and are the result of the combined efforts of a number of minds; Deidesheimer's feat was all the more remarkable in that his system was immediately successful.

The system comprised the placing of framed timbers in rectangular sets as the ore was removed. The sets could be extended readily to any required height and over any given area and formed a series of horizontal floors, built up from the bottom sets like the successive stories of a house. The spaces between the timbers were filled with waste rock or with wooden braces to form a solid cube whenever maximum firmness was desired. After the invention of square-setting the ore bodies along the line of the Comstock lode were extracted with comparative ease and security.

The advantages of the square-set system in mining large ore bodies were quickly recognized, and the idea spread throughout the world. The system received its widest application in the Western States owing to the unusual size and richness of the ore bodies subsequently discovered and the plentiful supply of timber available.

As square-set mining became widely known, there was a tendency for years to use the method regardless of its suitability. With the increasing need of economy and the development of other methods, square-setting has been superseded by cut-and-fill, top-slicing, or caving methods in many mines where ground conditions permit.

Shrinkage, cut-and-fill, top-slicing, and sublevel-caving methods were gradually developed to fit particular conditions underground and started to come into general use some years after the adoption of square-setting.

A new problem in underground mining was presented for the exploitation of the large, low-grade, disseminated copper deposits of the West. A very low cost was necessary for the deposits to be worked at a profit. The earlier of these so-called porphyries were worked by open-pit methods, but the thickness of the overburden made this method uneconomical at some mines. The problem was solved by the development of the undercut-block-caving system of mining, which was based on earlier caving methods on the iron ranges of Wisconsin and Michigan. The method has been improved continually since 1906, when it was first installed at the Ohio Copper mine at Bingham, Utah. The development of the undercut-block-caving method is probably the most important advance in underground copper mining methods since the invention of square-setting.

The comparatively recent development of the sublevel variation of open stoping has also been an important step in copper mining; the method is well adapted to large ore bodies within strong walls.

New developments in mining methods in one mineral industry are readily adopted by others; the type of deposit rather than the metal or mineral it contains governs the practices followed.

As the years have passed the principal mining methods have been improved and mining practices refined. The need for lower costs has resulted in operations that have been better planned; advancing systems have been changed to retreating and other major changes made. There has been a steady increase in tonnage of ore mined per manshift.

Improvements in mining equipment and machinery have kept pace with progress in mining practices and have made possible reduced costs over a wide range of conditions. New and better mechanical equipment and improved technique have accounted for most of the progress in open-cut mining. During the past generation underground copper mines have been generally electrified; this has reduced ventilation and pumping costs underground.

The drilling speed of present-day air drills is several times that of machines of 20 years ago. Moreover, they are now lighter and compare well in sturdiness. Present high-speed electric pumps have materially reduced the cost of drainage since the day of the steam pump.

The wider use of efficient scrapers in stopes has reduced shoveling costs; moreover, fewer chutes need be maintained, and other economies are effected by their use. There has been a general tendency to keep hand-shoveling to a minimum.

The trend has been toward equipping mines to handle larger fragments of ore. Improved chute construction, larger cars, and improved skips and skip-loading devices have permitted handling larger boulders, thereby reducing secondary blasting and breakage in stopes. The installation of crushers underground at some mines where the ore breaks in fragments too large to be handled at the skip-loading pockets has permitted the use of wider grizzlies and reduced secondary breaking cost.

The substitution of skips for cages has reduced costs at some mines. Moreover, the adoption of larger hoists and skips at other mines has not only reduced hoisting charges but has made possible the adoption of cheaper stoping methods. Improved primary blasting practices have had a part in reducing the amount of secondary breakage necessary at some mines.

Haulage costs have been reduced by the use of a wider track gage, better tracks, larger cars, and improved locomotives. Storage-battery locomotives have proved useful as an auxiliary to trolley systems or as the principal motive power at many mines. Hand and animal tramming have been largely eliminated on the main haulage levels. The installation of conveyors at one mine, the Boleo in Baja California, resulted in a marked reduction in mining costs.

Better ventilation in deep mines has increased the efficiency of the workmen. New working levels are now laid out with air courses and provision for controlling the air currents. Undercut and other workings in block-caving mines are designed for efficient ventilation.

Accident prevention is now generally recognized as a major operating problem. This attitude has been responsible for a steady reduction in accident compensation at most large copper mines. Moreover, fewer accidents favorably affect other costs.

Labor management has been improved since early days. More care is taken in selecting men for particular jobs; training courses are given at some mines to increase the efficiency of the workmen. Bonus and contract systems are based on more seasoned plans than formerly with correspondingly better results.

SELECTION OF A MINING METHOD

In general, the mining method selected for exploiting a given ore body is governed mainly by the type of deposit. From a miner's viewpoint, as previously stated, copper deposits may be placed in four general classes—the disseminated deposits or so-called porphyries, massive sulphide or replacement deposits, veins, and bedded deposits.

The ore bodies at the porphyry mines are generally flat-lying and relatively near the surface. Where the overburden is relatively thin an open-cut method of mining is used. Where the cost of stripping would offset the lower costs attainable by open-cut work the deposits are mined by an undercut block-caving method. The sulphide copper minerals in these ore bodies are disseminated throughout a siliceous gangue, which permits caving the ore without danger of ignition. Moreover, the deposits are fractured; this allows the ore to cave and break into small fragments when it is undercut. Table 18 shows the relation of thickness of capping to thickness of ore at 10 copper mines.

TABLE 18.—*Relation of thickness of capping to ore at 10 copper mines*

Mine	Location	Average thickness of capping, feet	Average thickness of ore, feet
Caving method:			
Ray	Ray, Ariz.	250 (40 to 600)	120
Inspiration	Miami, Ariz.	300 (0 to 500)	200
Ruth	Ruth, Nev.	410 (110 to 540)	120
Miami	Miami, Ariz.	320 (250 to 500)	325
Consolidated Coppermines	Kimberly, Nev.	500	175
Open-cut method:			
Utah Copper	Bingham Canyon, Utah	115	556
Copper Flat	Ruth, Nev.	190	224
New Cornelia	Ajo, Ariz.	[1]	250
Chino	Santa Rita, N. Mex.	[2]	
Sacramento Hill	Bisbee, Ariz.	280	227

[1] Original carbonate ore body outcropped, but some stripping was required along the edges. The sulphide part of the deposit dips under cover; the ultimate ratio of stripping will be 1 of ore to 0.83 of waste.
[2] Ratio of stripping—1 of ore to 1.76 of waste.

In massive sulphide deposits the physical characteristics of the ore and wall rock are usually the prevailing factors in the selection of the method of mining. At Ducktown, Tenn., where the ore is hard and the wall rock strong, an open-stope method is successfully used. In the Campbell ore body of the Copper Queen mine at Bisbee, Ariz., the ore and walls have less strength than at Ducktown but are strong enough to permit a semishrinkage method. At the United Verde mine natural conditions are less favorable than in the Campbell ore body, and a cut-and-fill method supplemented by square-setting is employed. Where ore bodies lie near or at the surface they may be mined by open-cut or glory-hole methods, irrespective of their physical characteristics.

The size, dip, and shape of the ore body, together with the structural strength of the ore and wall rocks, determine the stoping method to be used for mining veins or bedded deposits. Where the rock will stand without support an open-stope method probably should be used. If greater weights must be sustained, shrinkage, cut-and-fill, or square-set methods are employed, the choice depending upon the degree of support required. In heavy ground top slicing may be used. Because of their shape, deposits in veins or beds usually are not amenable to extraction by a caving method, such as is used in prophyry ore bodies.

The first consideration in the selection of a mining method is, of course, safety of operation. Aside from humanitarian reasons, a method that permits a low operating cost but causes a high accident rate may not be as economical as a method that entails a higher direct cost but keeps accidents at a minimum. The main cause of accidents in stopes is falls of rock; therefore a method should be chosen in which adequate provision is made to support the walls and back where such support is needed.

The grade of the ore and the selling price of the metal are considered in selecting a mining method. In high-grade deposits the loss of mineral due to use of a low-cost method may be more than the added cost per ton of mining by a method that permits a higher recovery of the ore.

Heavy precipitation and excessively low temperatures cause operating difficulties in open-cut mines; these may be the deciding factors in choosing between a surface and an underground method. Topography also is important in deciding between an open-cut and an underground method. The high cost of transportation on a site where tracks must be spiraled down into a pit may make an underground method more economical than an open pit.

Where the surface must be maintained, a method that permits subsidence cannot be used. Moreover, to prevent ground movement in a shaft or other workings near or over an ore body, a stoping method such as square-setting which allows a minimum of settling may be necessary.

Shrinkage or caving methods which do not permit selective mining would not be applicable to ore bodies with two classes of ore that must be mined separately or to deposits that contain relatively large inclusions of waste. In some ores waste inclusions may make mining by an open-pit method more economical than by a caving method.

Some copper minerals oxidize when left standing in a mine after being broken. Oxidized minerals are difficult to save in a flotation concentrator; moreover the tarnish on particles of sulphides due to oxidation increases the loss of sulphide minerals in the tailings. At some mines the more expensive cut-and-fill method is used instead of shrinkage stoping on account of the oxidization of ore in a shrinkage stope before it is ultimately withdrawn from the stope. In other mines an open-cut method may have an advantage over a caving system due to oxidization of the ore after it is caved and before it is drawn from a caving stope. An ore that packs in the stope after breaking is not amenable to mining by a shrinkage method.

Ore can be handled at lower cost in large-scale than in small-scale work. Large-capacity mechanical units and expert direction common in large operations, however, would not be justified in handling a small daily tonnage. A glory-hole method might be the most economical one for small mines, while open-cut mining would be chosen where a large daily tonnage was to be handled. Moreover, here, the size of the deposit would affect the method chosen.

An undercut block-caving method is applicable only to relatively large scale work; once a block is undercut the ore must be drawn at a definite rate to prevent packing. Moreover, for the method to be economical, relatively large blocks must be mined as a unit.

Timber is a large item in the cost of mining by the square-set method. Where timber is expensive, another method might better be used that would not be as advantageous in other respects.

Labor-saving devices can be used to better advantage with some methods than with others. Where the cost of labor, supplies, and power is relatively high, a method that requires more hand labor but has other advantages may be the most economical one. Under such conditions a glory-hole method may give lower over-all costs than open-pit mining.

Some methods are more adaptable to intermittent or varying production than others. Shutting down stopes that are being worked by an undercut block-caving method is a serious matter. Starting such a stope after it has been shut down for any length of time is difficult; moreover, the stope may be lost entirely. Open-cut mining has a decided advantage in this respect. When operations are likely to be intermittent, underground methods that permit a minimum of ground to be left open at a time have an advantage over methods that require extensive stope development. Square-setting and open-stope mining have an advantage over other methods where consideration must be given to the varying prices of metals. By these methods high-grade sections of an ore body may be worked and low-grade ones left in a fair condition for subsequent extraction when metal prices are higher. In mining by other methods low-grade sections cannot be left as conveniently and when left may be much more expensive to mine than if taken at first.

MINING METHODS AT TYPICAL MINES

The copper industry in the United States and northern Mexico went through a period of expansion during 1927–29, which culminated in 1929, the year of maximum production. In Canada, however, expansion continued through 1930 and 1931. Although some Canadian copper mines were closed during 1934, the production of copper in the country exceeded that of 1929. Beginning in 1930 most of the companies in the United States and Mexico began to curtail production, and one by one a large proportion of the mines closed. Many worked part time, principally to keep men employed.

The copper industry in the United States showed some improvement in 1934, the output being 27 percent of the 1925–29 average as against 21 percent in 1933. Two Arizona mines, the New Cornelia and Shattuck-Denn (part capacity), resumed operation during the year. The industry showed further improvement during 1935; production was practically double that for 1934. The Miami (part capacity), Inspiration, United Verde open pit, and Walker mines started operations, and most of the operating mines increased production in 1935. One new mine, the Mountain City Copper in Nevada, came into production during the year.

Although wages and supplies were abnormally high in 1929, costs per ton of ore mined were moderate on account of the large tonnages handled. During 1930 labor efficiency increased, and supply costs were lowered; on the whole costs were lower during the year than in 1929 despite the reduced tonnages mined. Further economies were made in 1931 and 1932 at some mines. On account of curtailments and shut-downs in 1931 to 1935, the years 1929 and 1930 are considered in this paper as giving a better cross section of methods and costs at copper mines than later periods. Moreover, at the time of writing, operating data since 1930 were largely lacking.

The total normal daily capacity of all copper mines in North America, 1930–35, was about 250,000 tons of ore. The principal producers of copper are the so-called porphyries. The ore bodies at these mines are low grade, and relatively large tonnages are handled daily. Either an open-cut or a block-caving method is used. About one-half of the total production of copper ore per year in the United States is mined by open-cut methods; over a quarter is obtained by caving.

The massive-sulphide, vein, and bedded-copper deposits in North America are mined by the following methods, listed in the order of their importance: Open stope, square-set, shrinkage, cut-and-fill, top slicing, glory holing, sublevel caving, and leaching in place. Aside from the porphyries most underground mines employ two or more methods of mining.

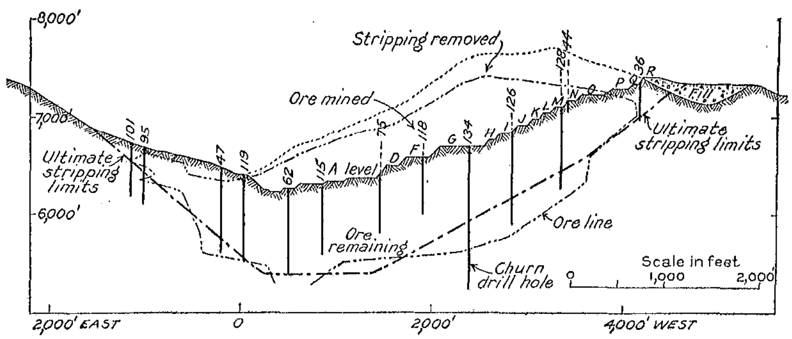


FIGURE 23.—Typical east-west section, Utah Copper ore body.

OPEN-CUT MINING

Table 19 lists the open-pit copper mines, together with production and geological data. The first to be developed was the Utah Copper in 1907 and the last, the Flin Flon in 1930. Open-pit mining at the Sacramento Hill mine was finished in September 1929.

UTAH COPPER

The Utah Copper Co. operates a low-grade, open-cut copper mine (fig. 22) at Bingham, Utah, 30 miles in a southwesterly direction from Salt Lake City. The ore is treated at two flotation concentrators, the Magna and the Arthur, which have a combined capacity of 60,000 tons daily at maximum metallurgical efficiency. The concentrators are at Magna, about 18 miles from the mine. The company owns its own standard-gage railroad, known as the Bingham & Garfield Ry., which is between the mine and the mills. Concentrates are smelted at the Garfield smelter of the American Smelting & Refining Co., situated 4 miles from the mills.

The Utah Copper ore body has an over-all length on its long axis of about 6,000 feet, a maximum width of 4,000 feet, and a vertical depth of about 2,000 feet. A typical east-west section is shown in figure 23. An average of about 115 feet of capping or completely leached porphyry covered the ore (see table 19). About 121,000,000 cubic yards of this capping had been removed to the end of 1934. The ore is relatively soft and breaks easily along fracture planes into sizes that can be loaded readily by power shovels with a minimum of secondary blasting. The ore and waste are loaded separately by electric power shovels with 4½-cubic yard dippers and caterpillar traction from 22 benches on the west side of the mine and 9 benches on the east side (1935). The top bench of the mine is 1,500 feet above the lowest one at the mine railroad yards.

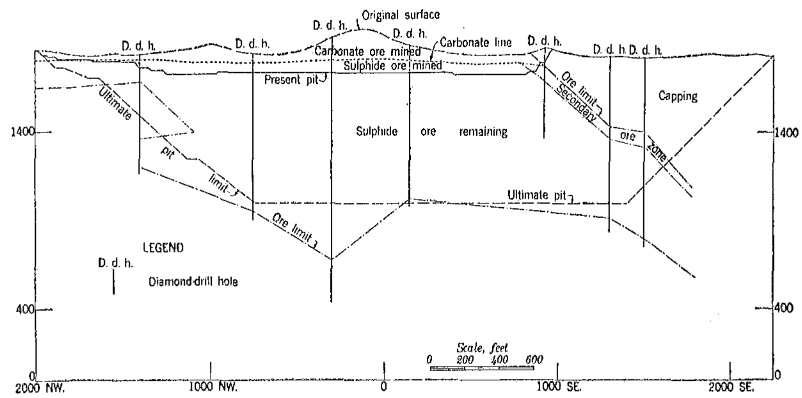


FIGURE 24.—Typical section of ore body, looking northeast, New Cornelia mine.

NEW CORNELIA

The New Cornelia mine of the Phelps Dodge Corporation is at Ajo, Ariz. The open pit is elliptical and is about 7,500 feet long and 3,500 feet wide. The ore body comprises disseminated-copper minerals in silicified monzonite and averages about 425 feet in thickness. (See fig. 24.) As little or no barren material covered the original ore body, relatively little stripping has been necessary; 3,667,000 cubic yards, mostly from the edges of the pit, had been removed to the end of 1934. An extension of the ore body dips under cover, and relatively more stripping will be required in the future.

The ore is mined in six 30-foot benches by 10 steam shovels with 4-cubic yard dippers and 2 electric shovels with 4½-cubic yard dippers. The ore near the surface was oxidized; it was treated in a 5,000-ton leaching plant, but leaching was discontinued in 1930. A flotation mill with a daily capacity of 5,000 tons was built in 1924. The capacity of the concentrator has been increased a number of times; its present capacity (1935) is 16,000 tons per day. The concentrator and leaching plant are about 1 mile from the mine. Concentrates are shipped to the Phelps Dodge smelter at Douglas, Ariz.

TABLE 19.—*General data for North American open-cut copper mines*

	Utah Copper [1]	New Cornelia [2]	Chino [3]	Copper Flat [4]	Sacramento Hill [4]	United Verde [5] open pit (lower pit)	Flin Flon [6]
Location	Bingham, Utah	Ajo, Ariz.	Santa Rita, N. Mex.	Ruth, Nev.	Bisbee, Ariz.	Jerome, Ariz.	Flin Flon, Manitoba
Normal daily production of ore ... tons	[7] 40,000	15,000	12,000	[8] 12,000	3,500	[9] 3,000	[10] 3,000
Normal daily removal of overburden ... cubic yards	23,000	12,000	13,000	6,000	1,500	[9] 3,000	([11])
Yearly production of ore:							
1929 ... tons	17,724,000	3,586,000	3,973,000	3,970,000	[12] 824,000	1,056,000	[13] 0
1930 ... do	9,552,000	2,370,000	[14] 2,514,000	2,165,000	0	[15] 500,000	206,000
1931 ... do	8,148,000	1,646,000	[15] 2,620,000	1,820,000	0	30,000	707,000
1932 ... do	3,169,000	342,000	[15] 1,165,000	1,009,000	0	0	881,000
1933 ... do	3,521,000	0	[15] 1,069,000	----------	0	0	968,000
1934 ... do	4,087,000	1,488,000	[15,16] 1,000,000	----------	0	[17] 0	884,000
Yearly production of copper:							
1929 ... pounds	296,625,000	70,000,000	[15] 85,000,000	[15] 94,000,000	34,691,000	[15] 70,000,000	0
1930 ... do	161,139,000	50,474,000	[15] 56,000,000	[15] 51,000,000	0	[15] 40,000,000	1,723,000
1931 ... do	142,695,000	41,393,000	[15] 56,300,000	----------	----------	----------	18,668,000
1932 ... do	60,013,000	10,373,000	[15] 26,500,000	----------	----------	0	22,363,000
1933 ... do	69,463,000	0	[15] 25,200,000	----------	----------	0	23,583,000
1934 ... do	78,787,000	33,085,000	[15] 21,800,000	----------	----------	0	19,814,000
Total ore mined to end of 1930 ... tons	203,400,000	32,709,000	----------	[15,18] 60,000,000	[15] 8,500,000	----------	206,000
Total ore mined to end of 1934 ... do	222,283,000	36,185,000	----------	----------	----------	[9] 5,323,000	3,646,000
Total overburden removed to end of 1930 ... cubic yards	110,000,000	3,174,000	[15] 50,000,000	[19] 44,145,000	[15] 11,000,000	[15] 8,700,000	871,000
Total overburden removed to end of 1934 ... do	121,225,000	3,667,000	----------	----------	----------	[9] 14,580,000	1,610,000
Total copper produced to end of 1930 ... pounds	3,484,000,000	718,000,000	1,260,000,000	[15] 1,500,000,000	340,000,000	[20] 1,960,000,000	1,723,000
Total copper produced to end of 1934 ... do	3,829,000,000	803,000,000	[15] 1,389,800,000	----------	----------	[20] 1,980,000,000	86,150,000
Ore body:							
Kind of rock	Monzonite	Monzonite	Granodiorite, quartz diorite.	Monzonite	Granite porphyry.	Schist, massive sulphides, and oxides.	Massive sulphides, schist, and quartz porphyry.
Character	Easily broken	Hard	Hard and tough	Easily broken	Hard	Medium	Hard
Length ... feet	[21] 6,000	3,500	----------	4,400	1,400	1,000	1,500
Width ... do	[21] 4,000	2,000	----------	1,500	[21] 1,200	800	300
Thickness ... do	[21] 2,000	425	----------	224	[22] 227	400	[22] 280
Grade of ore (percent of copper)	[23] 1.07	[23] 1.0	1.27	[23] 1.18	[14] 2.30	2.0, 5.0	[24] 2.10
Overburden thickness ... feet	115	[25] 0	([26])	190	280	([27])	100
Weight of ore and waste ... tons per cubic yard	2.08	2.16	2.00	2.16	----------	[28] 2.33	[29] 3.0
Benches:							
Number	[30] 22	(	8	11	3	5	5
Height ... feet	50–80	25–35	50	50	30	[31] 25–35	50 and 30
Width ... do	50–200	100–350	----------	100+	----------	([32])	100 to 300
Over-all angle of slope ... degrees	26	45	45	40–45	45	45	60 and 70

[1] Soderberg, A., Mining Methods and Costs at the Utah Copper Co., Bingham Canyon, Utah: Inf. Circ. 6234, Bureau of Mines, February 1930. Communication, November 1935, Moffatt, D. D., general manager, Utah Copper Co.

[2] Ingham, G. R. and Barr, A. T., Mining Methods and Costs at New Cornelia Branch, Phelps Dodge Corporation: Inf. Circ. 6666, Bureau of Mines, October 1932. Communication, February 1936, Curley, M., manager, New Cornelia Branch, Phelps Dodge Corporation.

[3] Thorne, H. A., Mining Practices at the Chino Mine, Nevada Consolidated Copper Co., Santa Rita, N. Mex.: Inf. Circ. 6412, Bureau of Mines, March 1931. Communication, January 1936, Tempest, R. B., general manager, Nevada Consolidated Copper Corporation, Chino Mines Division.

[4] Gardner, E. D., Drilling and Blasting in Open-cut Copper Mines: Bull. 273, Bureau of Mines, 1927.

[5] Alenius, E. M. J., Methods and Costs of Stripping and Mining at United Verde Open Pit, Jerome, Ariz: Inf. Circ. 6218, Bureau of Mines, February 1930. Communication, January 1936, Saben, W. M., manager, United Verde Branch, Phelps Dodge Corporation.

[6] Roche, M. A., Open-cut Blasting at Flin Flon: Canadian Min. Jour., vol. 54, June 1933, p. 219; Canadian Inst. Min. and Met. Eng., vol. 36, 1936, pp. 371–377. Communication, August 1936, Phelan, R. E., general manager, Hudson Bay Mining & Smelting Co., Ltd.

[7] Mill capacity, 60,000 tons.

[8] 1 shift, 1935.

[9] Estimated, end 1935.

[10] 1932–35; first half 1936, 2,100 tons and total of mines 4,528 daily.

[11] Overburden consisted of 1,000,000 tons of mud and clay; 750,000 tons removed by dredging during first 3 seasons at cost of 15 cents per ton.

[12] Open-cut mining completed September 1929; excludes 70,000 tons to leaching dumps.

[13] Started production in 1930.

[14] 1929.

[15] Estimated.

[16] Closed September 1934.

[17] 1935, production 700,000 tons.

[18] Including Ruth underground mine.

[19] In June 1929.

[20] Estimated total for underground and surface mine.

[21] Maximum.

[22] To be mined open-cut.

[23] 1934.

[24] Ore also contains 3.86 percent of zinc and 0.08 ounce of gold and 1.28 ounces of silver to the ton in 1936.

[25] No overburden overlaid on the carbonate ore body, but as depth was attained stripping was required at the edges of the pit. The sulphide ore dips under waste. In 1935 ratio of stripping to ore was 1:1; the ultimate ratio will be 1 of ore to 0.83 of waste.

[26] Ratio of stripping to ore mined, 1.76:1.

[27] Ratio of stripping to ore mined, 2.41:1.

[28] Average.

[29] Sulphide. Waste, 2.3 tons per cubic yard.

[30] West side 1934; east side 9.

[31] Above 160-foot level, from 50 to 270 feet. Below, 160-foot level, 25 to 35 feet.

[32] Variable.

CHINO

The Chino mine of the Nevada Consolidated Copper Co. is at Santa Rita, N. Mex. It lies in a small basin and originally comprised two open-cuts which have been joined. (See fig. 25.) The ore body consists of disseminated copper minerals in granodiorite, quartz diorite, and metamorphosed sediments. The ore alternates with waste that has to be removed as the ore is mined. Mining is done in 50-foot

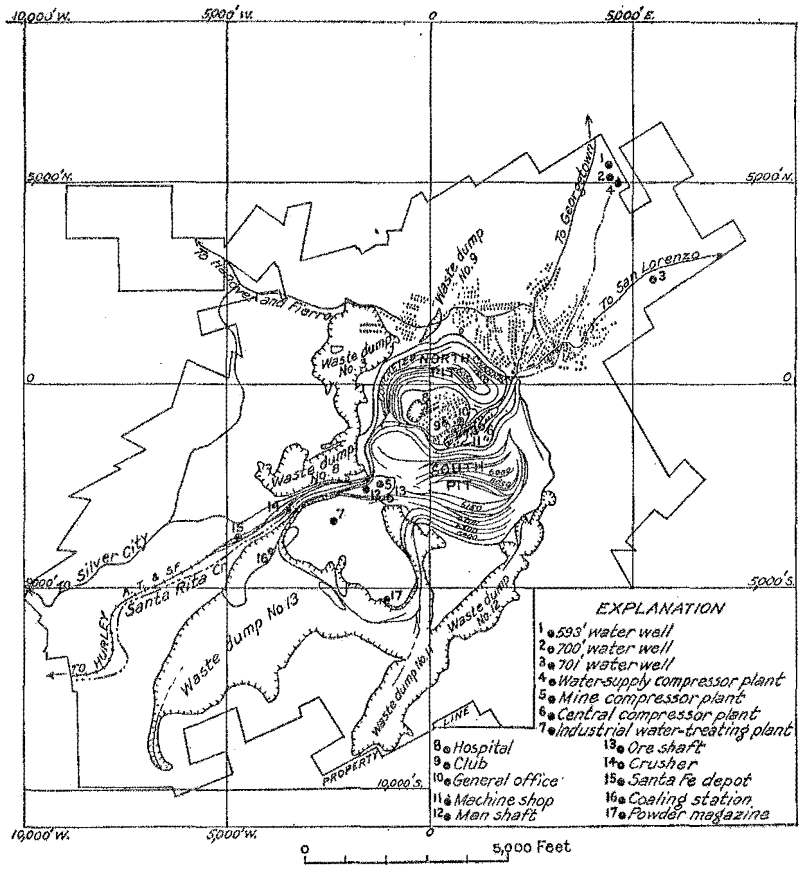


FIGURE 25.—Property map of Chino mines, showing pit outline, Santa Rita, N. Mex.

benches by four steam-, nine electric-, and one Diesel-powered shovels. A shaft has been sunk and some lateral development work done preparatory to mining part of the ore body at the edge of the pit by an underground method. The concentrator, which has a normal daily capacity of 13,500 tons, is at Hurley, 10 miles from the mine; the concentrates are shipped to El Paso, Tex., for smelting.

COPPER FLAT

The Copper Flat mine of the Nevada Consolidated Copper Co. is in the Ely district at Ruth, Nev.; about 1 mile away is the Ruth ore body of the same company, mined by a caving method. At both mines the ore consists of disseminated copper sulphides, mainly in monzonite porphyry. At Copper Flat the ore body occurred in and below a flat-topped hill which has been largely removed. The ore is mined in 50-foot benches by three electric shovels with 4½-cubic yard dippers and one with a 4-yard dipper. The ore from both mines is milled and the concentrate smelted at McGill, about 26 miles away.

SACRAMENTO HILL

The Sacramento Hill mine of the Copper Queen Branch, Phelps Dodge Corporation, is at Bisbee, Ariz. The hill comprises an intrusive mass of quartz-monzonite porphyry, part of which contains disseminated copper sulphides overlain by a capping which averages 280 feet in thickness. The pit had the shape of an oval bowl and at the 5,360 level was about 1,200 feet wide and 1,400 feet long.

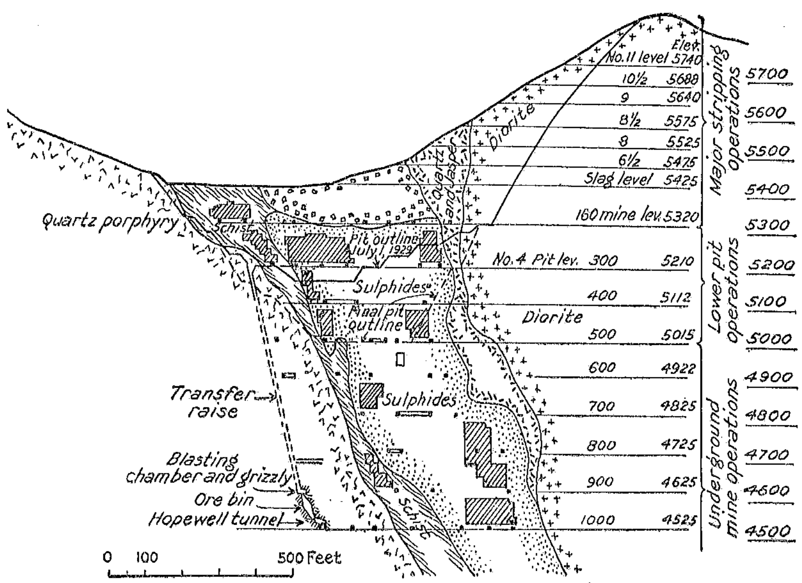


FIGURE 26.—Typical section of United Verde pit, Jerome, Ariz.

The ore was mined in 30-foot benches by seven steam shovels with 3½-cubic yard dippers. In 1929 the pit had reached a considerable depth with high banks on two sides. The hazard from slides increased with depth, and haulage of the ore up the spiral tracks from the lower levels was expensive. Therefore, open-cut mining was discontinued, and the ore remaining in the bottom of the pit was mined by a glory-hole method. The ore was concentrated at a mill about 2 miles from the mine, and concentrates were shipped to Douglas for smelting.

UNITED VERDE OPEN PIT

The open-pit mine of the United Verde Branch, Phelps Dodge Corporation, at Jerome, Ariz., is at the top of an ore body mined principally by a cut-and-fill method. The ore bodies of this company occur mainly in an immense body of massive pyrite, with a schist and porphyry foot wall which is partly mineralized; the hanging wall is barren diorite and greenstone. A fire area that extends from the 800-foot level to the surface includes a large tonnage of ore which could not be economically mined by underground methods.

The open-cut work was divided into two operations. In the "upper pit" it comprised chiefly stripping overburden; in the "lower pit" it was confined mainly to the ore. A typical section of the mine is shown in figure 26. The dotted lines show the final outlines of the pit as originally conceived; since then the plans have been amended. At the end of 1935 about 3,000,000 tons of ore remained to be mined by the open-pit method.

FLIN FLON

The Flin Flon mine of the Hudson Bay Mining & Smelting Co., Ltd., is situated at Flin Flon near The Pas, Manitoba. The deposit was discovered in 1915 in an uninhabited district. In addition to the mining plant, concentrator, and smelter, a 60-mile railroad and a town had to be built prior to mining. Moreover, a process had to be worked out first to treat a very complex ore. Production started in 1930.

The ore body consists of chalcopyrite disseminated through schist and massive iron, copper, and zinc sulphides replacing quartz porphyry. The sulphides are very hard, but the schist ore is soft. The ore body, where it is worked by the open pit, is about 1,500 feet long and has a maximum width of 300 feet. The open pit will probably extend to a depth of 280 feet. The overburden ranged up to 100 feet in thickness; its removal was completed in 1932. A total of 700,000 cubic yards of waste has been moved. During 1931 and 1932 an average of 658 cubic yards daily was handled, although work was actually done only in the summer months.

Open-cut mining is done with two electric shovels on caterpillar tractors, equipped with 4-cubic yard dippers.

OPEN-CUT COPPER-MINING PRACTICES

Open-pit work, unlike underground mining, varies little in general method. The material is mined in benches by power shovels and conveyed by cars on standard-gage railroad tracks to a concentrator or disposal dump. An exception to this method of transportation is practiced at the United Verde open-pit mine, where trucks are used to convey the ore to ore passes.

Overlying waste is first stripped; moreover, overburden beyond the limits of the ore at the edges of the pit must be removed to permit mining the boundary ore. As depth is attained this slope must be carried continually farther back; in time the ratio of overburden that must be removed to ore recovered may make open-pit mining uneconomical. All of the open-pit mines listed plan eventually to recover the boundary ore at the bottom of the deposits by underground methods.

Although the general method of excavation is the same at all the open-pit copper mines, numerous variations may exist in mining practices. All phases of open-pit mining are related, and all are influenced by the physical characteristics of the rock. The digging unit is the "heart" of an open-pit operation; other phases are carried on to keep the shovels busy.

The shape of the ore body and the topography of the surrounding country govern the manner in which the mine is laid out. Where the ore extends below the approaches to the mine, spiral tracks are usually required, as at the Sacramento Hill and the Copper Flat mines. On the other hand, where the ore occurs in a mountain with steep sides, as at the Utah Copper mine, a relatively long trackage with switchbacks is necessary. Cross sections of the ore bodies and pits at the Utah Copper, New Cornelia, and United Verde open-pit mines are shown in figures 23, 24, and 26. Figure 25 shows a plan of a lay-out of tracks at the Chino mine. General data on mining practices at open-cut copper mines are shown in table 20.

Benches.—The number of benches necessary to mine a given tonnage depends upon the topography. The number at any mine will vary somewhat as old ones are completed and new ones started. Table 19 shows the number at each mine. At the Utah Copper mine, where the surface slopes steeply, 25 are needed; at New Cornelia with its relatively flat topography the required tonnage can be mined from three or four working terraces. At the Utah Copper relatively narrow benches are used to keep the amount of stripping at the top of the slope to a minimum. In deposits such as that at Ajo wide benches are practicable. The minimum width of bench on which a shovel can work and a track be maintained is about 30 feet. Benches at the Chino mine are shown in figure 29 (p. 136).

With high benches more material can be moved with each set-up of a shovel, and less track work is required per ton of material mined; the present tendency, however, is to use lower terraces to permit cleaner separation of ore and waste and to promote safety of operation. The face of low benches can be trimmed with the shovel dipper. A relatively low bench is used at the New Cornelia on account of the tendency of the ore to break coarsely and the difficulty of drilling long air-drill holes which are used for primary blasts at this mine. High benches are sometimes necessary because of difficulty of approach or other topographical reasons.

Three benches have been mined and two remain at the Flin Flon open-cut mine. The ore benches were 50 feet high and 300 feet wide. The lower benches have been reduced to 30 feet high and will be less than 100 feet wide.

Angle of slope.—The maximum angle at which material will stand in unbroken banks and the safe maximum over-all slope of the sides of the pit are important factors in open-pit mining, especially where a mine is on a mountainside or the workings are surrounded by high hills; a variation of only a few degrees may make considerable difference in the amount of stripping required at the upper edges of the pit. Moreover, the safe over-all slope must be determined before the ore reserves of the mine can be established, as obviously the ratio of stripping to ore in a pit depends upon this angle.

Although the face of a single bench may have a relatively steep slope it does not follow that the face of the whole pit will stand at the same slope. The slope of each bench can be varied to fit local conditions, but the over-all average of the sides of the pits must be carefully worked out to prevent dangerous or expensive slides. Berms are usually left in the face of a pit as each bench is finished. They reduce the over-all slope of the sides and hold spalls resulting from weathering.

TABLE 20.—*Mining details, open-cut copper mines*

	Utah Copper [1]	New Cornelia [2]	Chino [3]	Copper Flat [4]	Sacramento Hill [4]	United Verde [5] open pit (lower pit)	Flin Flon [6]
Shovels:							
Number and type	23 electric	10 steam, 2 electric	9 electric, 4 steam, 1 Diesel.	4 electric [7]	7 steam	7 electric [7]	2 electric.
Size of dippers (cubic yards).	4½	Steam, 4; electric, 4½.	12 4-cubic yard, 1 8-cubic yard, 1 1¼-cubic yard.	1 4½-cubic yard, 3 4-cubic yard.	3½	2 4-cubic yard, 1 3-cubic yard, 4 1¾-cubic yard.	4.
Kind of traction	Caterpillar	Caterpillar	Caterpillar	Caterpillar	6 railroad, 1 caterpillar.	Caterpillar	Caterpillar.
Capacity (tons per 8 hours).	5,200 [8]	Steam, 2,500; electric, 4,000.	2,810 [9]	5,800	2,100	2,230 4-cubic yard, 1,200 3-cubic yard, 880 1¾-cubic yard.[10]	1,000.
Locomotives:							
Number and type	41 electric	16 steam	23 steam	10 aparejo tank, 4 saddleback.	15 saddleback	5 steam	3 electric.
Size (tons)	75	67	90	80	55	3, 85; 2, 55	2, 85; 1, 20.
Cars:							
Capacity (tons)	Ore, 80; waste, 60	40	40	Ore, 60 and 80, average 68; waste, 43 and 54.	12	Ore [11]; waste, 35	60.
Number to train	Ore, 11; waste, 5	6	8	Ore, 9	4	5 or 6	2.
Drills	3 ¼-inch piston	Churn and air hammers.	Churn, air hammer, air jackhammer.	Churn	Churn, air hammer, air jackhammer.	Churn, Leyner, jackhammer.	Electric churn.
Drill holes:							
Kind	Toe	Bench	Bench, toe	Bench	Bench, toe	Bench, toe	Bench.
Distance apart of rows (feet).	At bottom of slope	15	Bench, 10 feet from crest bank; toe, at bottom of slope.	55 to 60	Bench, 21; toe, at bottom of slope.	Churn at edge of bank	12.
Distance apart of holes in rows (feet).	15	15	Bench, 25; toe, 15	18 to 21	Bench, 21, toe, 15	10 to 12	15.
Depth (feet)	20 to 25	28	Bench, 30 to 55; toe, 22.	57	Bench, 35; toe, 21	Churn, 30 to 38; air, 20.	58 and 34.
Distance drilled per 8 hours (feet).	45	90	Churn, 43; air, 46	84 [7]	Churn, 38.5; air, 48	Churn, 6 to 40, average 17; air, 8 to 80.	12.7.
Ore broken per foot drilled (tons).	45			55 to 60 [7]	Churn, 30; air, 36		17.5.
Cost of drilling per foot			Churn, $0.62	Ore,[12] $0.56; waste, .57.	Churn, $0.48; air, .25		$1.45.
Cost of drilling per ton				Ore,[12] $0.007; waste, .004.	$0.021	$0.174 per yard broken [7]	$0.083.

Explosives:							
Kind for chambering	60-percent ammonia dynamite.	60-percent gelatin dynamite.	40-percent ammonia dynamite.	40-percent ammonia dynamite.	40-percent gelatin dynamite.	35- to 60-percent gelatin dynamite.	50-percent gelatin dynamite.
Kind for main blasts	do	Free-running quarry.	Churn, 35-percent granulated ammonia dynamite; air, 40-percent ammonia dynamite.	70-percent hercomite.[7]	Granulated ammonia dynamite.	35- to 60-percent gelatin and equivalent in granular ammonia dynamite.	Do.
Ore broken per pound of explosive (tons):	8	5.9	6	Ore,[12] 4.91; waste, 4.94.	2.5	2.84 [7]	2.3.
Cost of explosive per ton of ore.			$0.055	Ore,[12] $0.029; waste, .027.	$0.025	$0.047 per cubic yard broken.[7]	$0.081.[13]
Cost of drilling and explosive per ton of ore.	$0.026	$0.070	$0.096	Ore,[12] $0.038; waste, 0.036; average 0.037.	$0.101	Ore,[14] $0.145; waste, 0.109 per cubic yard broken.	$0.164.[13]

[1] Soderberg, A., Mining Methods and Costs at the Utah Copper Co., Bingham Canyon, Utah: Inf. Circ. 6234, Bureau of Mines, February 1930. Communication, Moffatt, D. D., general manager, Utah Copper Co., November 1935.

[2] Ingham, G. R., and Barr, A. T., Mining Methods and Costs at New Cornelia Branch, Phelps Dodge Corporation: Inf. Circ. 6666, Bureau of Mines, October 1932. Communication Curley, M., February 1936, Manager New Cornelia Branch, Phelps Dodge Corporation.

[3] Thorne, H. A., Mining Practices at the Chino Mine, Nevada Consolidated Copper Co., Santa Rita, N. Mex.: Inf. Circ. 6412, Bureau of Mines, March 1931. Communication Tempest, R. B., general manager, Nevada Consolidated Copper Corporation, Chino Mines Division, January 1936.

[4] Gardner, E. D., Drilling and Blasting in Opencut Copper Mines: Bull. 273, Bureau of Mines, 1927.

[5] Alenius, E. M. J., Methods and Costs of Stripping and Mining at United Verde Open Pit, Jerome, Ariz.: Inf. Circ. 6218, Bureau of Mines, February 1930. Communication Saben, W. G., manager, United Verde Branch, Phelps Dodge Corporation, January 1936.

[6] Roche, M. A., Opencut Blasting at Flin Flon: Canadian Min. Jour. vol. 54, June 1933, p. 219; Canadian Inst. Min. and Met. Eng., vol. 36, 1936, pp. 371–377; communication, Phelan, R. E., general manager, Hudson Bay Mining & Smelting Co., Ltd., August 1936.

[7] 1935.

[8] 1934.

[9] 1931.

[10] Waste only; ore hauled in 10- and 20-ton trucks.

[11] Trucking equipment consists of 10-ton gravity dump trucks and 20-ton hydraulic dump trucks.

[12] 1925.

[13] 1931–35, inclusive.

[14] 1928.

The character of the rock or ore is the most important factor in determining safe angles of slope, but the height of benches and depth of pit also have a bearing on the subject. Hard rock usually stands safely at steeper angles than decomposed or soft material. The over-all slope, however, depends mainly upon the structural features of the rock mass. Banks are likely to break to prevailing planes of weakness in the rock if the dip of the fractures is near the angle of slope. Disastrous slides have occurred along relatively flat fault zones in pits. Where practical the mine is laid out so that prevailing planes of weakness or fault planes dip into or run at right angles to the sides of the pit. As depth is attained slides are more likely to take place; an angle of slope that is safe for a shallow pit may be dangerous for a deep one in the same material. The natural angle of repose of broken rock is about 35°. Slides may take place, however, at flatter angles on the sides of deep pits in structurally weak rock.

Time is an important factor of ground movement in some high banks. Rain, snow, freezing, and thawing hasten ground movement; as time passes the movement gradually accelerates until a slide occurs. Minor dislocations of rock usually serve as a warning of a major slide. When trouble is anticipated on steep, fractured banks, bench marks are usually established at critical points on the bank. The points are closely watched and surveyed at intervals; thus major slides often can be anticipated. Banks may be carried at a relatively steep angle when mining can be completed in a short time; the longer banks stand the more dangerous they become.

The general over-all angle of slope of pits in open-cut copper mines ranges from 26° to 70° (table 19). At the Utah Copper mine the average bank slope is 50°; in 1935 the over-all angle of pit slope was 26°. At Ajo, where relatively wide benches are maintained, the ultimate over-all angle of slope has not yet been a factor but has been tentatively estimated at 45° for one section of the pit. The average slope of the bank faces is about 50°. At Chino it has been found that an over-all slope of 45° is safe, except in two comparatively small areas. No serious trouble from slides arises provided one shovel cut of 25 feet is left as a berm as each bench is finished. The safe over-all angle in part of the Copper Flat pit is 40°; in one section it is 45°. Individual benches stand at 45° to 60°.

At the open-pit mine of the United Verde Copper Co. an over-all angle of 45° was used in the softer rock and one of 60° in a high bank of hard diorite capping. The latter slope proved too steep, and in 1931 a slide of about 500,000 cubic yards occurred; however, subsidence, in 1929 in the underground workings caused cracks to form in the overlying diorite at the edge of the open pit and probably was directly responsible for starting the slide. The slide was cleaned up and the broken pit wall stripped back to an over-all slope of 45°. A total of about 5 million cubic yards of stripping was moved in the upper pit; the work was completed in 1935.

At the Sacramento Hill mine the ultimate over-all slope was 45°. The slope of the individual benches while being operated averaged about 75°. Berms 15 feet wide and 60 feet apart were left in the face in the lower part of the pit; the over-all slope there was the same as in the upper face where no berms were left.

The over-all slope of the Flin Flon pit is 70° on the foot-wall side and 60° on the hanging-wall side where the tracks are located.

DRILLING AND BLASTING

Drilling.—The width and length of the benches affect the selection of the drilling and blasting system. On narrow benches, where room is at a premium, there is obviously an advantage in following a procedure by which holes for blasting can be drilled at the toe or into the face of the bank and in blasting relatively short sections at a time. Long and wide benches permit a system by which many bench holes can be drilled and shot together. The most efficient operation of power shovels demands an ample supply of broken ore and a minimum of interruptions.

If the ore is uniform and breaks well there is little trouble with boulders, and secondary blasting is held to a minimum. Under these conditions drill holes can be spaced farther apart, thus lowering drilling costs. If the rock breaks in large fragments a drilling system should be chosen in which a large part of the direct force of the explosion is used to shatter the rock.

There is an economic balance at each mine between the number of holes drilled and the amount of explosives used. Sometimes less explosive would be required to break the ground if more holes were drilled, but the total cost of breaking the ground would be greater.

Methods of arranging the drill holes at open-cut mines are shown in figure 27; the types of drills used are given in table 20.

The cost per foot of drilling with air drills is considerably less than that with churn drills; however, a new type of electric churn drill recently put on the market has a much greater drilling speed than the old steam or gasoline rigs and a correspondingly lower cost per foot. This new machine may result in greater use of churn drilling.

A comparison of the cost of breaking ground with air drills and churn drills at the United Verde mine showed that, although the cost of powder consumed in blasting churn-drill holes was 50 percent more, considerable was saved in labor, power, and repairs. The total cost per ton broken with the churn drills was 16 percent less than with the air drills. In general, churn drills are used where the holes are relatively far apart and shot with heavy charges of explosive.

At the Copper Flat mine churn drills have proved very successful. The ore breaks well, and relatively little secondary blasting is necessary. Moreover, very little hard bottom requiring drilling and blasting is left in this ground. At the Chino, Sacramento Hill, and United Verde mines, however, churn-drill blasts have to be augmented by toe holes to break the bottom ground. Moreover, at the Chino mine where the rock breaks in large fragments the footage drilled in breaking boulders is about double that drilled for primary blasts. At the Utah Copper mine the ore is broken principally by toe holes, supplemented occasionally by holes in the face of the bank.

At the United Verde open pit, where a great variety of ground conditions exist, both churn and air drills are used. In stripping in dense diorite a combination of churn-drill and toe holes (fig. 28) was adopted after either method alone had proved inadequate to break the ground satisfactorily. Churn-drill holes are not drilled within 15 or 20 feet of the final pit limits. The final drilling is done with jackhammers; the holes are spaced close together and loaded lightly. This practice, although more expensive, leaves a more solid wall, as most of the fracturing caused by the heavy shots is eliminated.

Churn-drill holes are preferable in the hot, gassy ground of the lower pit as they can be loaded more safely and quickly than air-drill

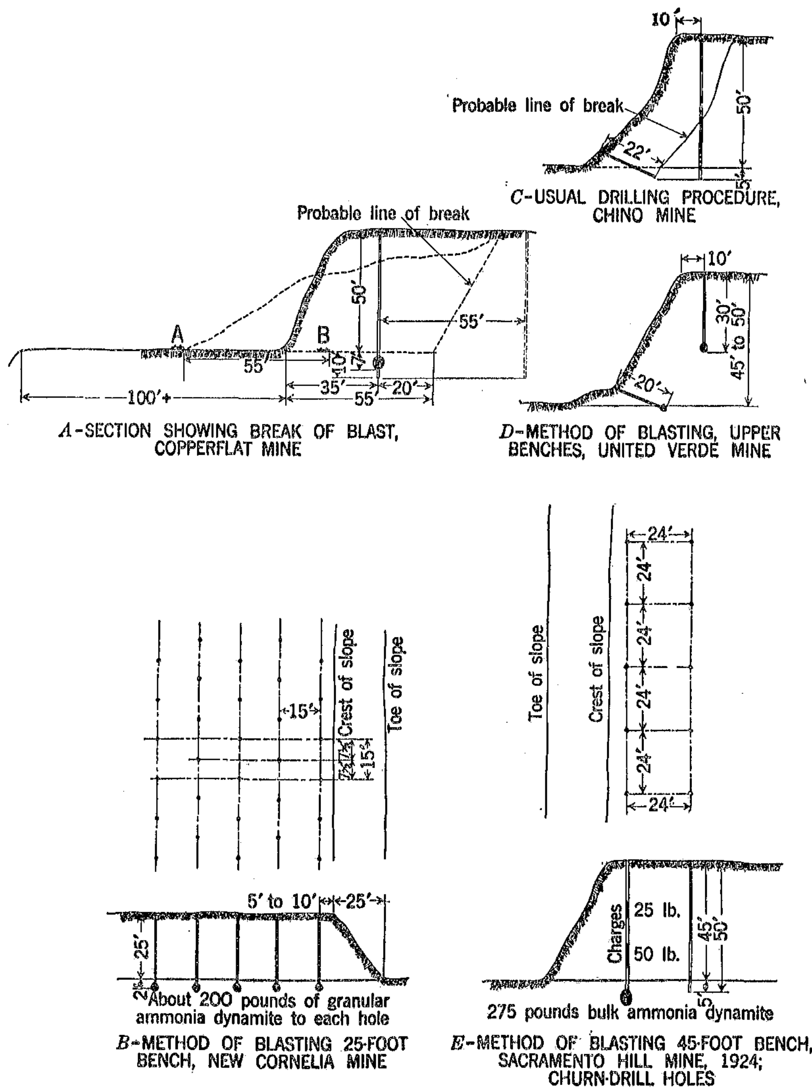


FIGURE 27.—Arrangement of drill holes for blasting in open-cut copper mines.

holes because of their larger bore. In hard sulphide, toe holes are also needed to break the ground.

At the New Cornelia mine, where close spacing of holes (on 15-foot centers) is required to break the ground, air drills are used. The row next to the bank may be drilled at an outward angle to reduce the normally heavier burden on these holes. Two or three rows are blasted at a time. The broken ground of the outer row of holes acts as a cushion for the back rows, which may be loaded more heavily to cause better fragmentation of the ore without scattering the blast and covering the tracks. A change to churn drills was being considered in the autumn of 1935; one churn drill was being run for comparison. The holes were shot without chambering.

The massive sulphide ore at the Flin Flon mine is more difficult to drill and blast than the siliceous ores at the other open-cut mines. Close spacing of holes and relatively heavy charges are required to break the ground for the shovels. The practice at this mine has been to shoot a large number of 6-inch churn-drill holes as a single blast; this method causes relatively little interference with the loading and transportation of the ore. In the upper part of the pit 58-foot holes were drilled 20 feet apart in rows; in the lower part, 34-foot holes were drilled 15 feet apart. Rows are 12 feet apart.

A half million tons were broken in one of the blasts (no. 4) in August 1934.[18] The area blasted was 150 feet wide and 700 feet long. The main blast comprised 232 holes spaced on 17.5-foot centers in staggered rows (fig. 28) and averaged 58 feet deep. An additional 175

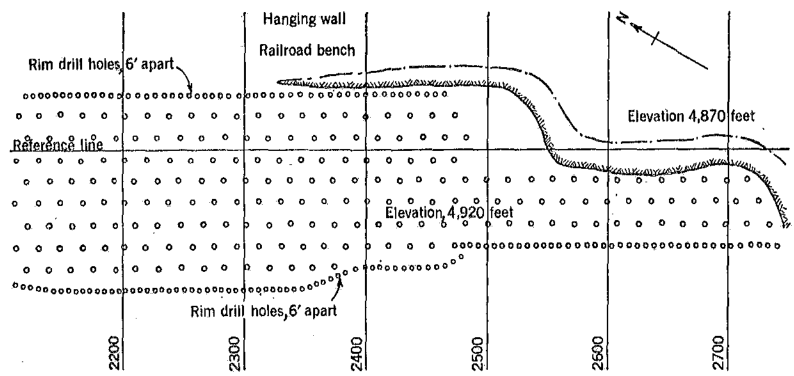


FIGURE 28.—Plan of blast no. 4, 4,920 bench, Flin Flon mine.

"rim holes", 6 feet apart, were drilled on the foot-wall and hanging-wall outlines of the shot. Another 105 holes were drilled 30 feet deep in soft schist along the foot wall of the pit outside of the blasted area to create a line of weakness and insure a clean break on the foot wall. A total of 1,297 drill shifts was required to drill the holes which had a total footage of more than 5 miles. Seventy more shifts and 15,850 pounds of 50-percent strength Polar Forcite gelatin were used to spring the holes. After chambering, 200 machine shifts with the drilling machines were required to bail the broken ore out of the holes. A drill crew consisted of two men. A total of 300 man-shifts was worked to load the holes; the total charge comprised 123 tons of 50-percent strength Forcite gelatin in 5- by 16-inch cartridges.

The shot was fired with 550-volt alternating current split into 5 parallel circuits, 27 series in parallel to the circuit, and 15 No. 8 instantaneous electric caps to the series. The wiring was done in 1 day. The following items were used for the wiring and detonating: 2,050 No. 8 detonators, 6,000 feet of No. 6 lead wire, 2,000 feet of

[18] Roche, M. A., Breaking Half a Million Tons in One Blast: Min. and Met., August 1934, pp. 340–341.

No. 4 feeder wire, 300 pounds of No. 20 connecting wire, and 290 pounds of friction tape.

Ore broken totaled 472,000 tons, or enough for 6 months' operation. In addition, 33,000 tons of waste were broken. The pit was 150 feet deep and 3,000 feet long after this blast.

Drills.—The drilling equipment at the Utah Copper mine consists of 3¼-inch reciprocating drills used on tripods without leg weights. Modern hammer drills have been tried repeatedly, but under conditions at this mine the piston type serves best for drilling the desired 22-foot flat holes. Moreover, water under pressure is required for hammer drills; to supply this, water lines would have to be maintained on each level, which would be nearly impossible during the winter months at Bingham. At Chino and at the Arizona mines freezing of pipe lines is not a serious problem, and heavy hammer drills have replaced piston machines.

Air-hammer machines of the heavy drifter type are used for drilling the holes for the main blasts at the New Cornelia mine and in part of the United Verde open-cut mine. The drills were formerly suspended by a rope block from a tripod made of three pieces of 1½-inch pipe. At both mines the tripods have been discarded for wagon drills. With the wagon drills speed of drilling at the New Cornelia increased about 10 percent and at the United Verde 50 percent over the former systems. At the United Verde a wagon drill is used for vertical holes and heavy drifters mounted on tripods for toe holes in hard ground. Toe holes in soft ground are drilled by hand-held jackhammers.

The substitution of electric drills for steam drills at United Verde saved the difference between $8 per drill-shift for fuel for the steam drills and $0.60 for power for the new electric drills.

Four new-type electric drills at the Copper Flat mine replaced nine old-style steam drills. The drilling speed of the new drills is about 2½ times that of steam drills and in 1935 averaged 84 feet per 8-hour shift. Increased drilling speed is attained by a heavier stem and a faster stroke. The drills are mounted on caterpillar treads and are moved quickly from hole to hole or bench to bench under their own power. A bit will drill 30 to 150 feet before it needs to be resharpened; the average is 60 feet.

Primary blasting.—The amounts and kinds of explosives used in the seven mines are shown in table 20. Enough explosive is used in the main blasts not only to break the ground but also to shatter the ore as much as possible without throwing it out on the tracks or shovels. Boulders and high bottom cause delays at the shovels which may cost more than the saving in explosive if a charge just large enough to break the ground is used.

As it is desirable to have the broken ore in the same relative position after the blast as before, most of the explosive used in the main blasts generally is concentrated in chambers at the bottoms of the holes. Explosives are needed occasionally in the upper parts of deep churn-drill holes to shatter the ground. (See fig. 27, *E*.)

At most of the mines granulated ammonia gelatin is used in dry churn-drill holes and the same explosive in cartridge form in dry air-drill holes. Gelatin dynamite is used in wet work at all the mines except the Copper Flat, where ammonia dynamite may be used.

Electric detonators are used for setting off the main blasts at the New Cornelia, Chino, United Verde open pit, Sacramento Hill, and

Flin Flon mines; at the Utah Copper mine cap and fuse are still used. Cordeau Bickford is used at the Copper Flat and Chino mines for churn-drill holes. Electric caps are used for detonating springing charges at all of the mines except Utah Copper. Batteries are used at the Chino and Copper Flat mines and current from power lines at the New Cornelia, United Verde, and Sacramento Hill mines for shooting the springing charges.

Secondary blasting.—At most of the open-pit mines any fragment of ore that will pass through the dipper of the power shovel can be handled now at the concentrators. Improvements in dumping arrangements and larger primary crushers have reduced secondary blasting costs at some of the mines.

Boulders in sight are block-holed and blasted before starting to load out a blast at most mines. Those uncovered during loading that are too large to move are block-holed as found; however, at some mines boulders are broken by mud-capping so that there will be a minimum delay in the operation of the shovel. The ore at the New Cornelia is the most difficult to break of that in any of the open-cut mines except the Flin Flon. Secondary blasting is also a serious problem at Chino. About 20 percent of the labor cost of drilling and blasting at New Cornelia is for breaking boulders; it is less at the other western mines.

At the Copper Flat mine boulders that will not go through the shovel are piled back of the shovel along the face of the bank. As the shovel moves on, a portable air compressor is pulled up by a tractor. The boulders are then drilled and blasted without interfering with shovel operations.

Coyote blasting.—Coyote blasts were used at most of the open-pit mines early in their development for breaking irregular bluffs or ridges where benching would be impractical. This method, however, has not been used at any of the mines in later years. The method is not recommended for breaking blocks of ground where the same level is to be excavated later by benching. The huge blasts start incipient movement along planes of weakness in the unbroken ground which may cause slides in later banks.

LOADING PRACTICES

Power shovels instead of power draglines are used at all of the open-cut copper mines, because the shovels are better adapted to the hard digging necessary to load the ore. Improvements in shovels and shovel practice during recent years have had an important part in reducing mining costs in the open-cut mines.

The capacity per shift of a given power shovel and, incidentally, the cost per ton of loading depend partly upon the supply of broken ore and partly upon the percentage of coarse material in the broken rock. Material that breaks well can be loaded faster than coarse ore; boulders too large to be loaded cause delays while they are blasted. Delays due to mechanical difficulties or break-downs have been largely eliminated in the equipment now being used in open-cut copper mines. Regular inspections and overhauling keep the equipment in running order; spare units usually are kept in reserve to replace those out of service for general overhauling or major break-downs.

All operations in an open-cut mine center around the digging units; all other activities are timed to keep the shovels loading ore during the maximum length of time each shift. Good drilling and blasting practices keep the shovels supplied with broken ore; the most efficient transportation system immediately replaces a loaded train with a string of empty cars.

The development of open-cut copper-mining practices is perhaps best exemplified at Utah Copper, the largest individual copper mine in North America and the first to use the open-cut method. The first power excavators placed in operation at Bingham were 60-ton steam shovels. Soon, however, the 90- to 100-ton railroad-type shovel on standard-gage track became the accepted digging unit.

In 1923 shovel practices were improved greatly by equipping the excavators with caterpillar traction. This obviated laying short sections of tracks as a shovel advanced and consequently saved the labor of four or five men in ground crews at each shovel. More important than the saving in labor, however, was the facility with which a caterpillar-equipped shovel could move forward and backward under its own power.

In 1923 the first electric shovel was introduced at the Utah Copper mine. All shovels were electrified by 1934. The average daily capacity of the old railroad-type steam shovel with 3½ cubic yard dippers was 2,350 tons per 8-hour shovel shift in 1923, compared with 5,200 tons for the modern shovels with 4½-yard dippers in 1934. At the Chino mine during 1 year of normal operations when an equal number of shifts were worked by steam and electric shovels, it was found that a saving of more than $70 per 8-hour shift was made with each electrically operated shovel.

At the Copper Flat mine two 4-yard, full-revolving electric shovels were installed during 1931 to replace two steam shovels. Owing partly to the greater efficiency of the electric shovels and partly to improved track conditions the tons mined per shovel-shift for the year increased 10 percent over 1930. In 1935 all (nine) of the old-style steam shovels had been replaced by four modern, full-revolving electric shovels with caterpillar treads. The four electric shovels, three with 4-cubic yard and one with 4½-cubic yard dippers, handled the same yardage on one shift that was loaded formerly on two shifts with the nine steam shovels. In October 1935, an average of 5,800 tons was handled daily by each shovel shift.

The flexibility and economy of the electric shovel have been improved by introduction of the full-revolving feature and Ward-Leonard control. The full-revolving shovel can travel and dig equally well in either direction and can place the large boulders behind it out of the way of immediate loading operations. The Ward-Leonard installation gives the operator the advantage of the smooth and positive control of direct current.

At the Utah Copper mine each shovel takes a cut 20 feet wide, beginning at one end of the bench and proceeding across it to the other end; no waste is sorted from ore on the benches. When the shovel reaches the end of its travel the direction is reversed and the operation repeated. The progress of each shovel is arranged so that one will not be directly above another. The material is drilled and blasted ahead of each shovel.

At the Copper Flat mine two cuts were taken with a shovel after each blast in 1935. After the first cut, the track was moved in 55 to 60 feet. The second cut cleaned up the blast back to the solid. The shovels are capable of taking a 68-foot cut.

FIGURE 29.—Benches at Chino mine.

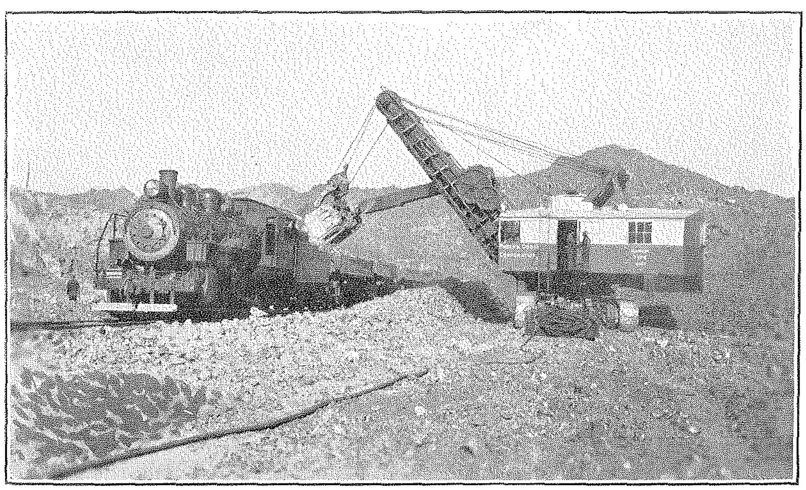


FIGURE 30.—Electric shovel, New Cornelia mine.

FIGURE 31.—Loading waste at Utah Copper mine; electric shovel, locomotive, and dump cars.

The electric shovels at the New Cornelia (fig. 30) can take a 63-foot cut as against a 38-foot cut for the steam shovels. This extra capacity permits keeping the ore tracks farther from the bank and enables the electric shovel to clean up the ground broken by heavier blasts in one cut.

At the Utah Copper mine ten to twelve 80-ton ore cars usually are brought in and spotted for loading; the locomotive remains coupled. When the train is loaded and on the way to the yard, another locomotive brings in the next train, and the process of loading continues. The average time of loading an ore train with a capacity of 800 to 1,000 tons is 1 to 1¼ hours. Waste trains comprise three to seven 30-yard cars (fig. 31) depending on the proximity of the waste dumps; the loading procedure is the same for waste as for ore. A similar routine of loading is followed at the New Cornelia, Chino, Copper Flat, and Sacramento Hill mines.

Ribs of waste occur in the ore at the Chino mine and to a small extent at the New Cornelia; this waste is taken to the dumps. Concentrating ore, leaching ore, and waste were separated at the Sacramento Hill mine. A grab sample was taken from each car as loaded to a field assay office and immediately run for copper by a color method. By the time the trains reached the assembly yards the assays were out. The material was then taken in trains by standard locomotive to the concentrator, a leaching dump, or a waste dump, according to the copper content as shown by the field assay.

Because of the need for close sorting of the various materials mined at United Verde, small shovels (1¾-cubic yard dippers), relatively low benches (25 to 33 feet), and autotrucks are employed in the lower pit. The small shovels can be moved readily in order to drill and blast to the best advantage. Soft ore is generally removed first, leaving the harder material clean. The work is planned and the levels are developed to assure a constant production of any desired material. The shovels can work on any exposed bank, and the trucks used for haulage can be routed to any of the raises through which the ore is dropped to a haulage adit below.

TRANSPORTATION

The transportation of ore to a concentrator or waste to a dump is a railroading problem. Standard switches, frogs, and other railroad equipment are used for tracks. All permanent tracks are ballasted, but the lines on the benches that are moved after each blast usually are not.

Electric locomotives have proven more efficient than steam at the Utah Copper and Flin Flon mines; other companies are giving this type of motive power consideration (1935).

Each shovel at the Utah Copper mine is served by two locomotives. Movement of trains is controlled by flagmen placed at advantageous points. Loaded ore cars are taken to an assembly yard and turned over to the railroad department for transportation to the concentrator. "Waste cars" are pulled to dumps and unloaded by the mining department.

Before electrification was started the standard transportation equipment at the Utah Copper mine comprised 85-ton modern steam locomotives. In February 1930 forty-one 75-ton electric locomotives had replaced the same number of steam units then in service. Tests showed that the trolley-type electric locomotive was the most effective, but seven locomotives of the combination trolley and storage-battery type are used for special service and emergencies. All locomotives are constructed to permit ballasting up to 90 tons. They are of the articulated type and are capable of hauling 12 empty cars up the 4-percent grades on the switch-backs at 12 miles an hour. The locomotives are equipped with a conventional main pantograph, two sidearm collectors, and a cable reel containing 2,000 feet of cable.

In 1934 approximately 66 miles of trolley had been erected; standard catenary construction (fig. 32) is used for trolley lines over permanent tracks, and direct suspension is employed elsewhere. Owing to the necessity of operating spreaders to clear snow from the tracks, all towers are placed on the bank side of the track.

Flat-bottom ore cars which are unloaded by a rotary dump at the mill are used; 620 80-ton and 100 100-ton cars were in use in 1935; all new cars are of the larger size. Waste cars are dumped by compressed air, and the size used has increased gradually until at present 36-cubic-yard cars are standard equipment.

Ninety-pound rails are used in new construction and in replacing the 65-pound rails first used for the mine tracks. With two exceptions, all switch-backs have single tracks.

Track on the shovel levels is shifted by mechanical track shifters; nine are in use. The crew for one of these machines is six men. Three to five track gangs of 20 men each are also used on the tracks. Track on the dumps is shifted by track gangs using lining bars.

At the New Cornelia mine 16 oil-fired, steam, railroad-type locomotives are used in transporting the ore from the pit to the concentrator; 5 of the engines are used as spares. The maximum track grade is 2 percent and the maximum curvature 20°. Seventy-pound rails are used throughout the mine. The ore is loaded into 42-ton air-dump cars and hauled in trains of six cars each. When a train is loaded it is hauled directly to the mill with the same engine.

Motive power at the Chino mine comprises eleven 90-ton, eight 85-ton, and four 47½-ton coal-burning locomotives. The smallest ones are used for switching and delivery of coal, powder, and other supplies. The standard equipment for handling both ore and waste is a 20-cubic-yard dump car. Ten cars make a train.

At the Copper Flat mine both spirals and switch-backs are used in bringing the tracks from the mine. The maximum upgrade is 2½ percent; pitches of 4 percent are used occasionally on the downgrade. Steam locomotives of the aparejo and saddle-tank types are used in the pit. A standard railroad engine is used for switching. The ore is loaded into 60- and 80-ton gable-bottom steel cars and the waste into 20- and 25-cubic yard dump cars. Ore trains comprise nine 60- and 80-ton cars; the average load per car is 68 tons. When a train is loaded a pusher engine is coupled on the rear to take it out of the pit. Fourteen locomotives serve four shovels. Standard railroad equipment is used on the railroad from the yards to the mill, 26 miles distant.

At the Copper Flat mine the ground is graded for the tracks and the tracks are shifted by a bulldozer (a tractor with a road blade on the front), of which two are used in the pit. The bulldozers also clean up the tracks after blasts, level off the ground ahead of the shovels, make ramps between levels to facilitate easy transfer of equipment, and make roads throughout the pit. They are employed as tractors to move a portable air compressor. Trucks handle supplies throughout the pit.

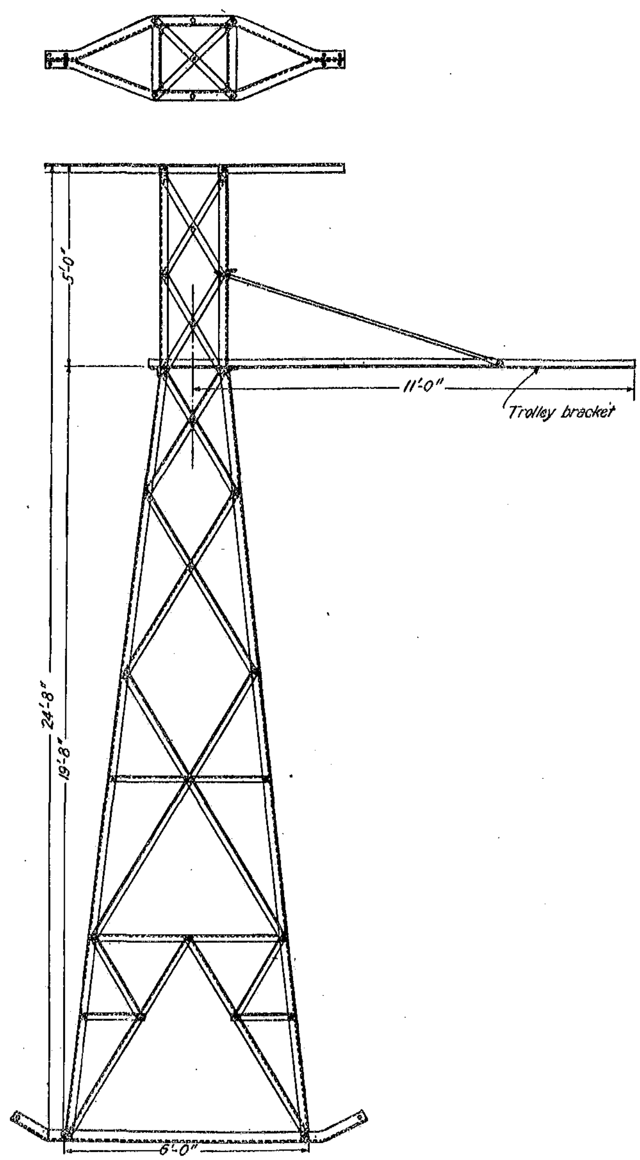


FIGURE 32.—Portable transmission and trolley tower, Utah Copper mine.

Due to the shape of the Sacramento Hill mine sharp curves (up to 40°) were necessary. This feature prevented the use of six-wheel standard locomotives. The locomotives selected were of the saddle-back type and were the heaviest available that could be carried safely on four wheels. Trains of four 25-yard cars were the heaviest that could be hauled up the maximum grades of 2½ percent.

Two electric locomotives are employed at the Flin Flon mine. A train consists of two 50-ton cars. A maximum grade of 6 percent is used out of the pit.

In the upper pit where trackage is available motive power at the United Verde comprises 3 six-wheel and 2 four-wheel steam locomotives. Waste is hauled in 20-cubic-yard dump cars, five or six making a train. A new type of car with drop doors has speeded stripping; the lift doors of the old style of car limited the size of fragments that could be dumped.

In November 1935 about 1,000 tons of ore daily were transported directly from the pit over the company railroad to the smelter, about 11 miles distant by rail; 2,000 tons were dropped through ore passes to the Hopewell tunnel and thence hauled 6 miles by rail to the smelter.

A locally built machine comprising a bulldozer blade actuated by an air cylinder saves considerable labor by leveling off dumps when moving tracks. The blade pushes the material alongside of the track at right angles over the edge of the dump. Compressed air is supplied by a compressor mounted on the machine.

A bulldozer at the United Verde open pit has proved handy. It is used for making truck roads, cleaning up after blasts, pushing ore in nearby raises; with a trailer it serves as a tractor for hauls up to 300 feet and levels the ground on benches in waste for laying tracks.

Trucks are used in the lower pit for carrying the ore and waste to ore passes. The haulage equipment in 1935 comprised:

6 trucks, 6-wheel, 4-wheel rear drive, 4-cylinder, 10-ton capacity.
5 trucks, 6-wheel, 4-wheel rear drive, 6-cylinder, 10-ton capacity.
6 tractor trucks, 2 front wheels, caterpillars on rear, 6-cylinder, 10-ton capacity.

The trucks were equipped with a standard gravity side-dump body, designed and built by the United Verde Copper Co. By releasing the latches the load could be dumped very quickly when the truck was slightly tilted. Most of this equipment was replaced (1936) by four 20-ton trucks equipped with hydraulic side-dump bodies and 10 pneumatic tires.

The tractor trucks are useful only for keeping operations going under bad conditions, such as in deep mud and up steep grades. They are especially valuable during bad weather but cannot compare with the six-wheel trucks under fair conditions.

In 1928, 167 cubic yards or about 390 tons of material were trammed per truck-shift. The average length of haul was 500 feet one way. Gasoline consumption amounted to 1 gallon for 11.60 cubic yards trammed. For each shovel-shift 2.70 truck-shifts were required to keep the shovels operating at full speed.

UNDERCUT BLOCK-CAVING METHOD

The undercut block-caving method is used for mining large, disseminated deposits of considerable horizontal extent and usually over 100 feet in thickness. The ore bodies at the Inspiration mine (fig. 33) are representative of those mined by this method. It is used where open-cut mining is not applicable and where loss of ore and dilution by waste will be more than compensated by the relatively low mining costs.

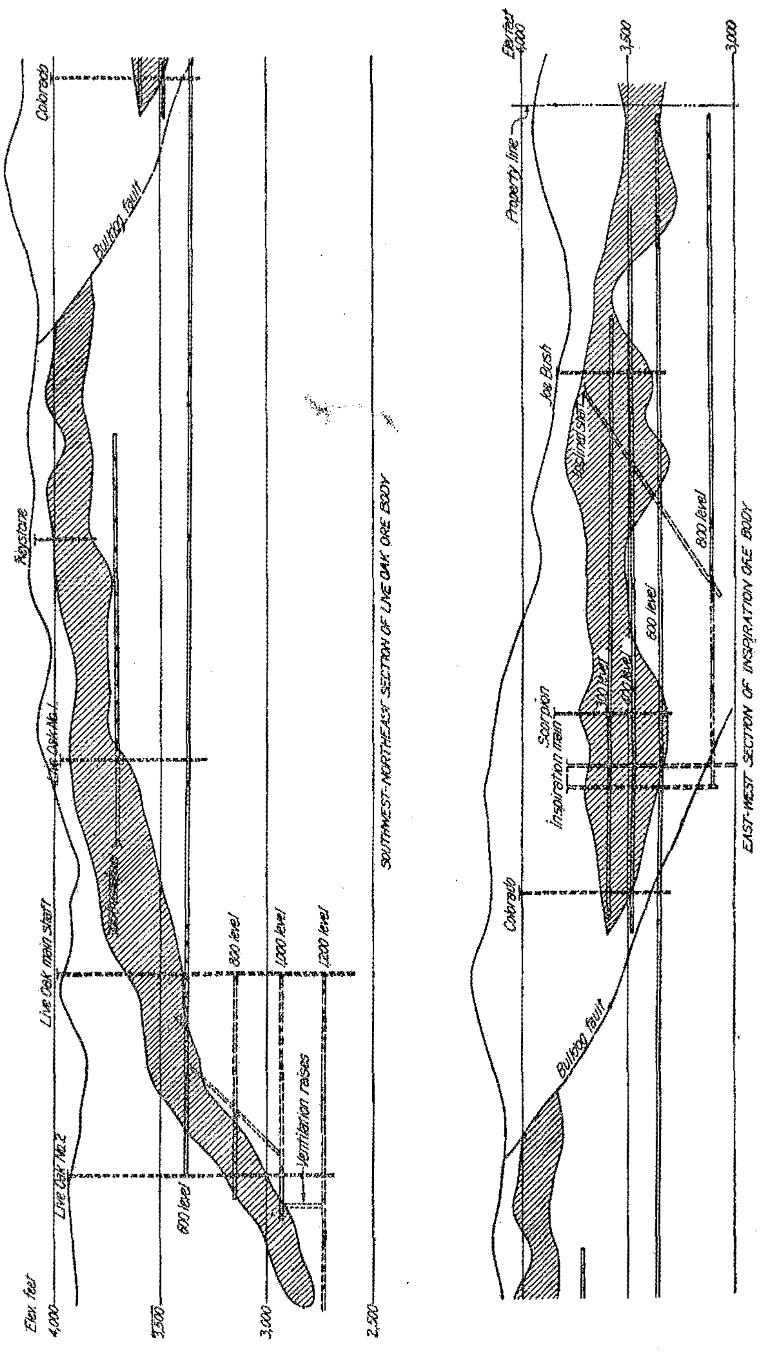


FIGURE 33.—Sections of Live Oak and Inspiration ore bodies, Inspiration, Ariz.; vertical and horizontal scales the same.

From 10 to 20 percent of the ore may be lost owing to the limitations of the method. Several companies using this system of mining, however, plan to leach the caved material remaining after underground operations have been completed.

As the ore bodies mined by this method are of relatively low grade, the work to be profitable must be done on a large scale. Operations are conducted on three levels—the level on which the ore is undercut; the grizzly level, on which the ore is drawn from the stopes; and the haulage level, on which the ore is transported to the shaft or surface. After the ore is undercut it is drawn through regularly spaced draw raises on the grizzly level. The ore passes through the grizzlies, then drops through gathering raises to the haulage level below, where it is loaded into cars.

The deposits being mined by the undercut block-caving method were drilled and their depth, thickness, and horizontal extent determined before plans were drawn for mining. Although major changes cannot be made in a method of mining a block of ore once it is established, improvements and refinements have been made in practice. Moreover, as experience has been gained important improvements have been made in applying the general methods to new ore bodies. Much preliminary development and stope preparation are required before extraction can begin. This work requires considerable time and capital.

Table 21 shows the normal daily tonnage and yearly production of copper from each of the mines using the undercut block-caving method. The spacing of development work preparatory to caving at seven of the mines is given in table 22. A general description of the principal mines using the method follows, also an account of the different phases of the operation.

TABLE 21.—*Production data at North American copper mines using undercut block-caving methods*

	Inspiration	Miami	Ray	Morenci	Ruth	Consolidated Coppermines	Porphyry	Old Dominion
Location	Inspiration, Ariz.	Miami, Ariz.	Ray, Ariz.	Morenci, Ariz.	Ruth, Nev.	Kimberly, Nev.	Bisbee, Ariz.	Globe, Ariz.
Normal daily production of ore...tons	[1] 18,000	18,000	12,000	4,500	3,500	3,500	1,800	[2] 900
Grade of ore...percent of copper	[3] 1.155 [4] 1.224 [7] 1.335 [8] 1.204	[3] 0.830 [4] .716 [7] .696 [8] .774	[5] 1.246	2.0	[6] 2.0	1.1	2.3	2.5
Total yearly production of ore:								
1929...tons	5,800,000	5,018,000	3,610,000	1,704,000	1,114,000	1,071,000	474,000	[6] 220,000
1930...do	3,042,000	6,125,000	2,327,000	1,346,000	644,000	1,115,000	579,000	[6] 243,000
1931...do	2,625,000	4,439,000	1,305,000	1,304,000	408,000	617,000	[6] 175,000	
1932...do	464,000	1,418,000	592,000	752,000	243,000	74,000	None	None
1933...do	None	None	None	None		None	None	None
1934...do	None	356,000	None	None		None	None	None
Total yearly production of copper:								
1929...pounds	107,516,000	58,841,000	[6] 66,000,000	55,074,000	[9] 45,000,000	22,732,000	18,478,000	18,943,000
1930...do	65,264,000	67,125,000	[6] 36,000,000	42,618,000	[9] 25,000,000	32,616,000	24,207,000	17,597,000
1931...do	61,368,000	50,573,000	[6] 24,438,000	38,344,000	[6] 16,000,000	15,076,000		11,348,000
1932...do	17,024,000	[6] 15,814,000	[6] 14,404,000	23,861,000	[6] 10,000,000	1,740,000	None	None
1933...do	None	None	None	None		None	None	None
1934...do	None	[6] 14,322,000	None	None		None	None	None
Total production of ore to end of 1930...tons	69,387,000	49,000,000	49,000,000	[10] 11,000,000	([11])	4,200,000		
Total production of ore to end of 1934...do	72,476,000	55,213,000	50,897,000	[10] 13,056,000				
Total production of copper to end of 1930...pounds	1,256,000,000	1,002,000,000	1,115,950,000	[10] 410,700,000	([11])	106,000,000		[12] 600,000,000
Total production of copper to end of 1934...do	1,334,392,000	1,082,709,000	1,154,792,000	[10] 472,905,000				611,000,000
Size of ore body:								
Length...feet	8,000	3,500	7,000	2,000				350
Width...do	800	2,700	1,500	600				200
Thickness...do	200	325	250		120	175		100

[1] 1935, 10,000 tons.
[2] Caving only; total mine, 1,500 tons.
[3] 1929.
[4] 1930.
[5] 1928.
[6] Estimated.
[7] 1931.
[8] 1932.
[9] Estimated; net production for Nevada Consolidated Copper Co. Nevada, Arizona, and New Mexico mines for 1929, 266,275,000 pounds; for 1930, 141,980,000 pounds.
[10] By block caving since 1922.
[11] Included with Copper Flat.
[12] Total for mine, 1904 to 1930.

TABLE 22.—*Spacing of development work preparatory to caving at 7 copper mines*

	Inspiration [1]	Miami [1]		Ray	Morenci		Ruth	Consolidated Copermines	Porphyry (Copper Queen)
		Sulphide	Oxide		Upper lifts	Lower lifts			
Normal daily output...tons..	[2] 18,000	18,000	3,000	12,000	4,500	[3] 4,500	3,500	3,500	1,800
Distance apart of haulage drifts...feet..	50–100	150	100	50	56	----	87.5	60, 120	50–150
Distance apart of gathering raises...do....	25	50	50	25	28	72	25	30, 60	40
Distance from haulage level to grizzly level...do....	30–100	100	66	40	50	[4] 240	50	40–110	50–100
Inclination of gathering raises...degrees..	55	55	54	55+	75	68	50	65	60
Distance apart of grizzly drifts or raises...feet..	25	50	50	25	28	36	12.5	30	40
Distance apart of drawpoints along grizzly drifts...do....	16±	25	25	25	21, 24	18	12.5	15	20
Plan area served by each drawpoint...do....	12.5 by 16±	12.5 by 12.5	12.5 by 12.5	----	14 by 18.67	18 by 18	12.5 by 12.5	15 by 15	20 by 20
Area served by each drawpoint...square feet..	200	156.25	156.25	312.5	261.3	324	156.25	225	400
Distance of undercut level above grizzly level...do....	16–20	30	30	----	20	20	10	12	16
Distance apart of undercut drifts or raises...do....	11, 14, 16	25	25	12.5	18.6, 14	18	12.5	15	20
Width of panels...do....	100–180	150	150	200	112	162	87.5	120	100
Length of panels...do....	Up to 800	Up to 1,200	Up to 300	Up to 600	Up to 600	----	Up to 500	----	Up to 200
Width of blocks...do....	100–180	150	150	200	----	162	----	120	100
Length of blocks...do....	100–125	200	200	150–200	----	200	----	----	[5] 125
Height of blocks...do....	125–225	260–320	262	100–300	150–270, [5] 164	270	40–200, [5] 135	[5] 175	60–250

[1] 1935.
[2] 1935, 10,000 tons.
[3] Both lifts not worked simultaneously.
[4] Established by existing level; 80 to 100 feet preferred.
[5] Average.

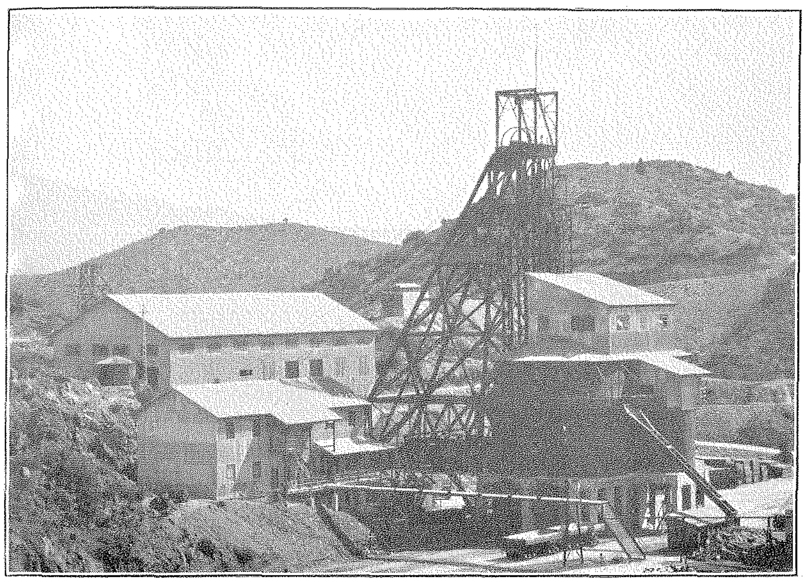


FIGURE 34.—Live Oak main shaft; surface.

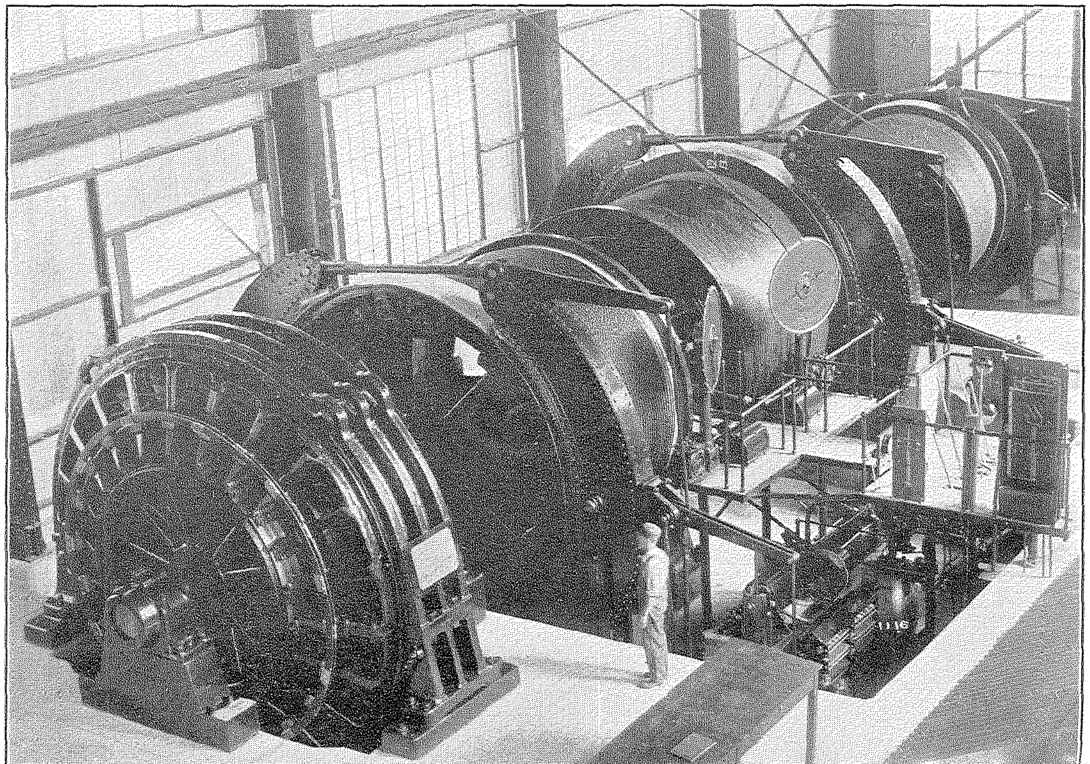


FIGURE 35.—Live Oak ore hoist and man-cage hoist.

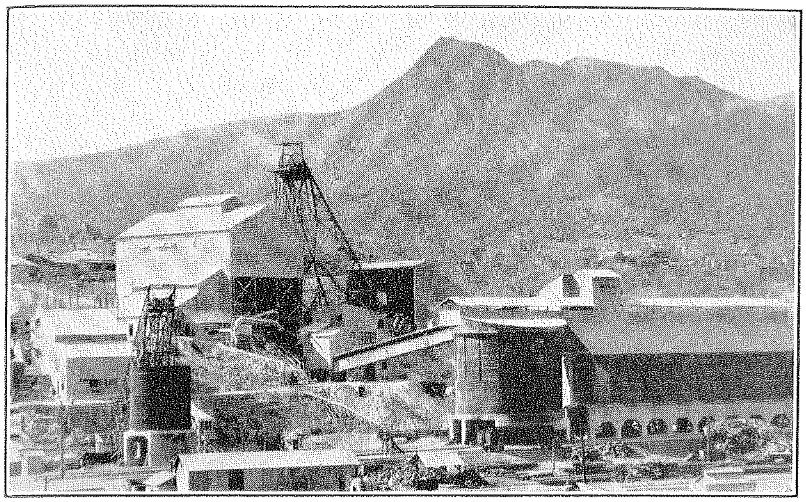


FIGURE 36.—Surface plant at No. 2 shaft, Ray mine, Nevada Consolidated Copper Co.

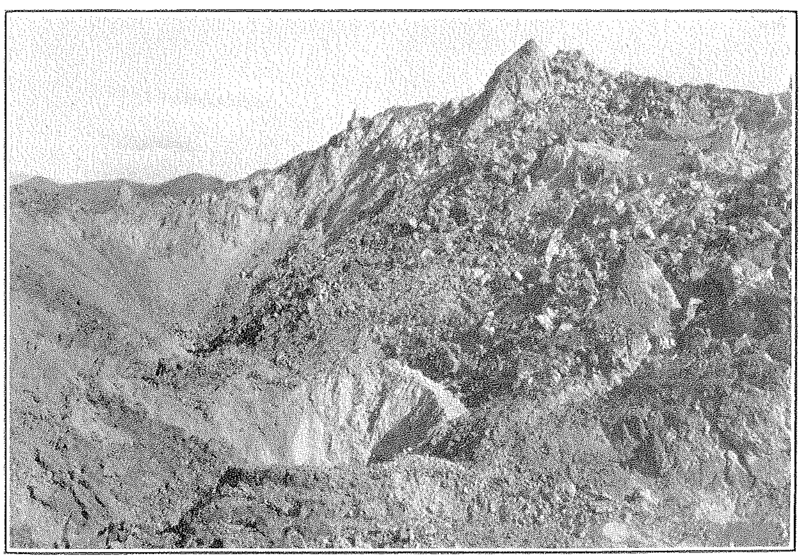


FIGURE 37.—Caved capping, Morenci mine.

INSPIRATION

The Inspiration mine is in the Globe-Miami district at Miami, Ariz. Sulphide ore is treated in a concentrator about 2 miles from the mine; oxide and mixed ore are leached at a plant near the main shaft. Concentrates are smelted at the International smelter about a mile from the mill.

The total length of the Inspiration ore body (fig. 33) is about 8,000 feet, the average width 800 feet, and the average thickness 200 feet.[19] The mine is worked in two divisions—the Inspiration and the Live Oak. Figure 34 is a photograph of the surface at the Live Oak shaft. Figure 35 shows the hoist. The Inspiration end of the property was developed first and is worked through twin shafts 700 feet deep. This division has three haulage levels—the 400, 600, and 850. The Live Oak division is developed by a 1,400-foot shaft. The shaft is connected to the Inspiration shaft on the 600-foot level, and the ore is dropped to this level for transportation to the Inspiration shaft. The general method of mining is shown in figure 38. The manner of undercutting shown, however, is not standard at this mine.

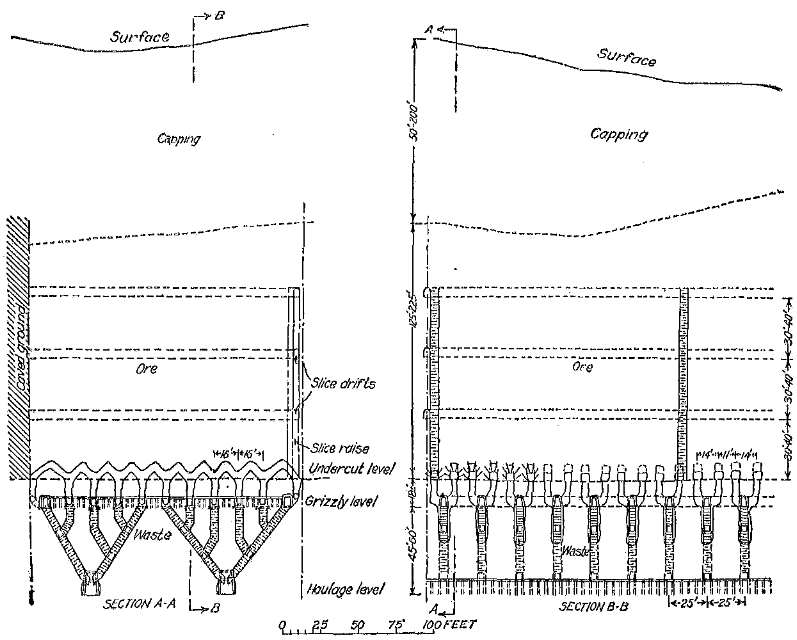


FIGURE 38.—Sections of caving block, Inspiration mine.

MIAMI

The mine of the Miami Copper Co. is at Miami, Ariz. The ore is treated at the mine in a concentrator and a leaching plant. Concentrates are smelted at the plant of the International Smelting Co. nearby. The present main ore body is 3,500 feet long, 2,700 feet wide, and an average of 325 feet thick; its area is about 100 acres.[20]

[19] Stoddard, Alfred C., Mining Practice and Methods at Inspiration Consolidated Copper Co., Inspiration, Ariz.: Inf. Circ. 6169, Bureau of Mines, 1929, 23 pp.

[20] Maclennan, F. W., Miami Copper Co: Method of Mining Low-grade Ore Body: Trans. Am. Inst Min. and Met. Eng., Year Book for 1930, pp. 39–86.

The mine is worked through a 1,120-foot shaft sunk for mining the original ore body. Supplies are trammed to the stopes on the 570-foot level, and the ore is hauled to the shaft on the 720-foot level. Three other levels—the 850, 1,000, and 1,120—will be used for mining lower parts of the ore body.

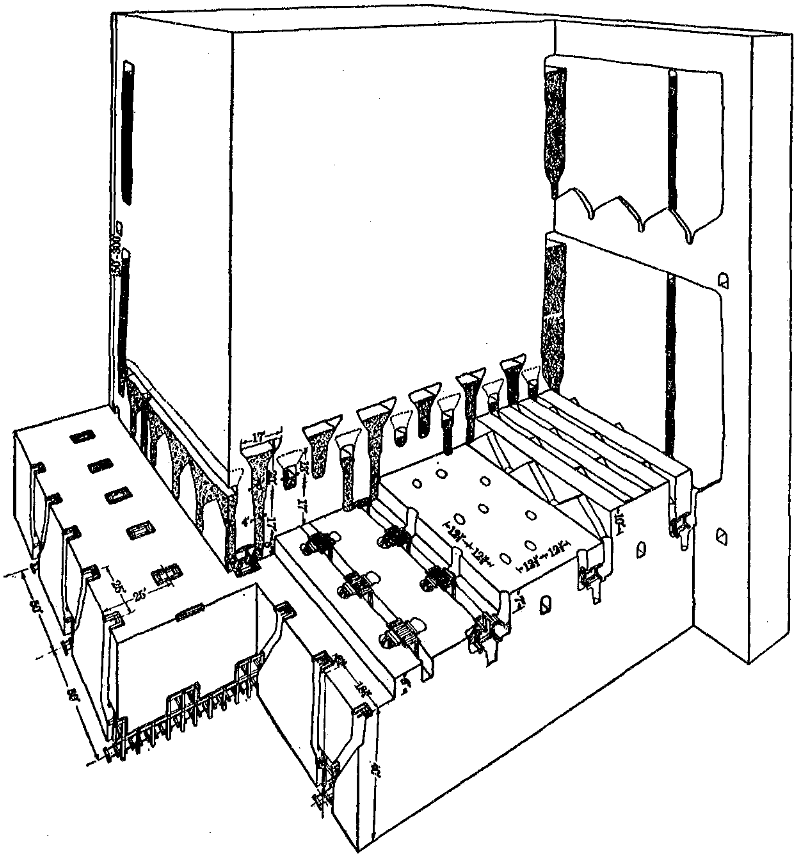


FIGURE 39.—Undercut caving method, Ray mine.

RAY

The Ray mine of the Nevada Consolidated Copper Corporation is at Ray, Ariz. The ore is concentrated at the company mill at Hayden, about 20 miles away. The concentrates are smelted at the Hayden plant of the American Smelting & Refining Co. The deposit is irregular both in plan and section. It occurs in two connected bodies. The total length of the deposit is about 7,000 feet; it has an average width of 1,500 feet and thickness of 250 feet, but the thickness ranges from 40 to 600 feet.[21] The mine is developed as a whole by two vertical shafts for hoisting ore, two inclined shafts for handilng mine waste and supplies, and one vertical shaft for handling men.

[21] Thomas, R. W., Mining Practice at Ray Mines, Nevada Consolidated Copper Co., Ray, Ariz.: Inf. Circ. 6167, Bureau of Mines, 1929, 27 pp.

The main haulage levels are at 150-foot intervals, and to date four have been run. Figure 36 shows the surface plant at no. 2 shaft and figure 39 the general method.

MORENCI

The Morenci mine of the Phelps-Dodge Corporation is at Morenci, Ariz. Prior to 1922 top slicing was the principal mining method used in the district, although nearly every known method had been used to a greater or less extent. From 1923 to 1926 the so-called Morenci slide method was perfected and used. The caving method, illustrated in figure 40, was devised from the Morenci slide. The Humboldt ore body, the last one mined, had a maximum length of 2,000 feet, a

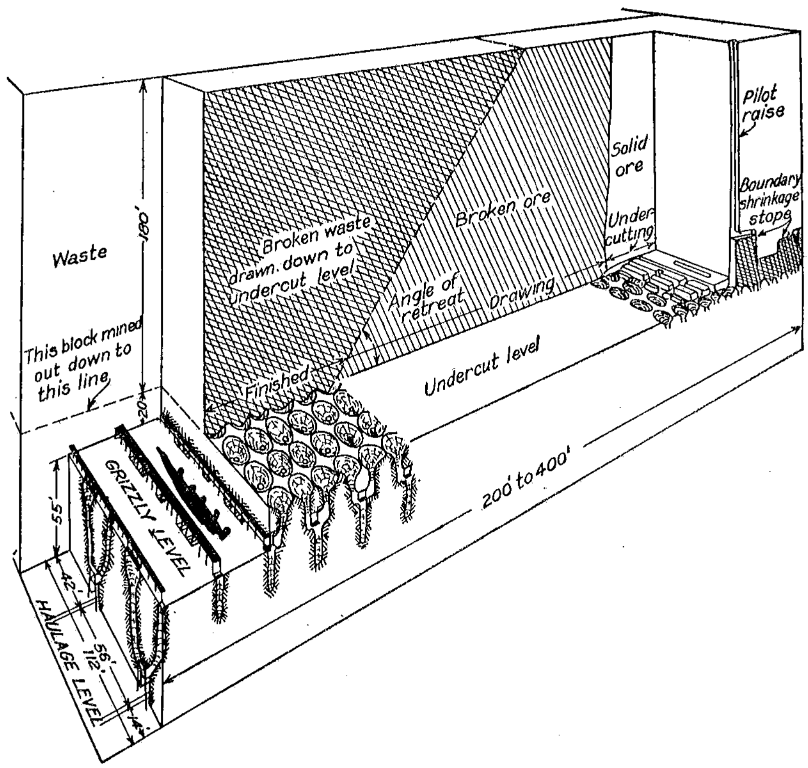


FIGURE 40.—Perspective of Morenci block-caving system, Morenci, Ariz.

maximum width of 600 feet, and a vertical range of about 1,000 feet. The ore was hoisted in a 973-foot shaft at the concentrator, about 4,000 feet from the stopes. Levels 6, 9, and 14 were haulage levels. The concentrate was shipped to the company smelter at Clifton, about 6 miles away. Before 1930 all ore produced was taken from above level 6. Men and supplies were lowered through a separate shaft near the stopes. Figure 37 is a photograph of caved capping at the Morenci mine. The mine was closed in 1932; future production will probably be from the Clay ore body, which will be mined by open-cut methods.

RUTH

The Ruth mine of the Nevada Consolidated Copper Co. is in the Ely district at Ruth, Nev. The original Ruth ore body as developed was oval, 1,600 feet long and 1,200 feet wide, and averaged 120 feet thick.[22] As mining and milling methods were improved lower-grade ore could be mined profitably. The inclusion of lower-grade material has considerably increased the size of the ore body. The mine is worked from a vertical shaft, from which nine levels have been run.

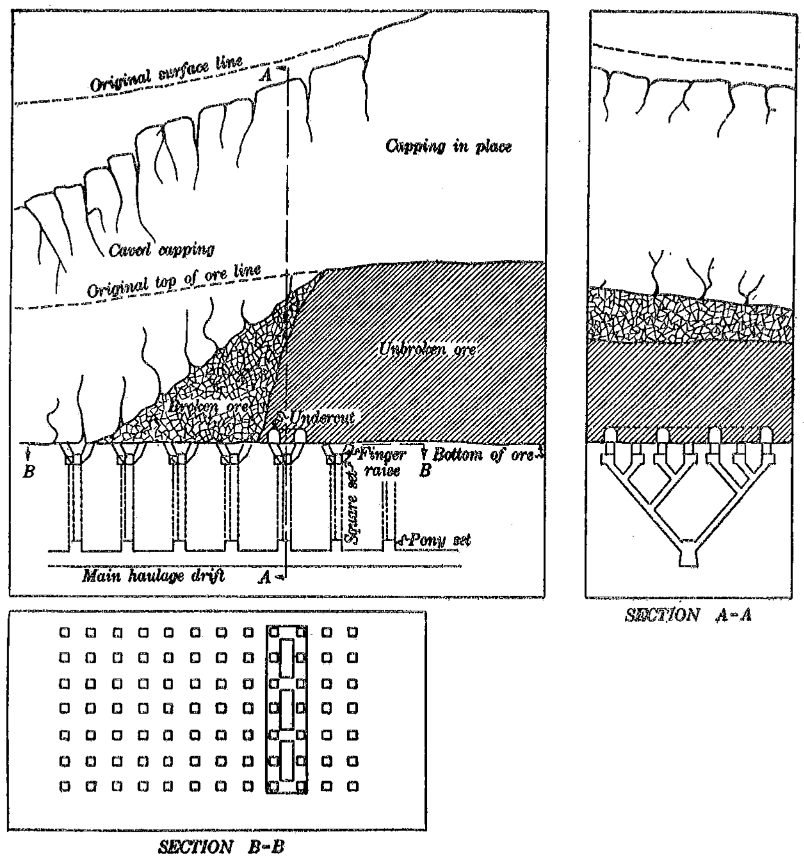


FIGURE 41.—Caving system, Ruth mine.

The ore is soft and caves readily. Branch raises are used to give access to the drawpoints and to gather the broken ore. The ore is concentrated at McGill, in the same plant that treats the ore from the open-cut pit. The system of mining and plan of undercutting are shown in figure 41. In October 1935 about 1,300 tons daily (instead of 3,500 tons as before) was being mined at Ruth; it was expected that the 1935 figure would be the normal annual tonnage of the mine during the remainder of its life.

[22] Parsons, A. B., Nevada Consolidated Copper Co.: Min. and Sci. Press, Mar. 5, Apr. 16, May 21, and Sept. 3, 1921.

CONSOLIDATED COPPERMINES CO.

The property of the Consolidated Coppermines Co. is at Kimberly, in the Ely district of Nevada. The porphyry ore bodies, which are mined by the undercut block-caving method, adjoin the Copper Flat open-cut mine of the Nevada Consolidated Copper Co. The ore is mined through a new five-compartment shaft at which a modern surface plant has been constructed. Figure 42 shows the method of mining the main ore body. A relatively small tonnage of high-grade ore from a different ore body is mined by square-setting. The ore is treated under contract at the Nevada Consolidated concentrator at McGill.

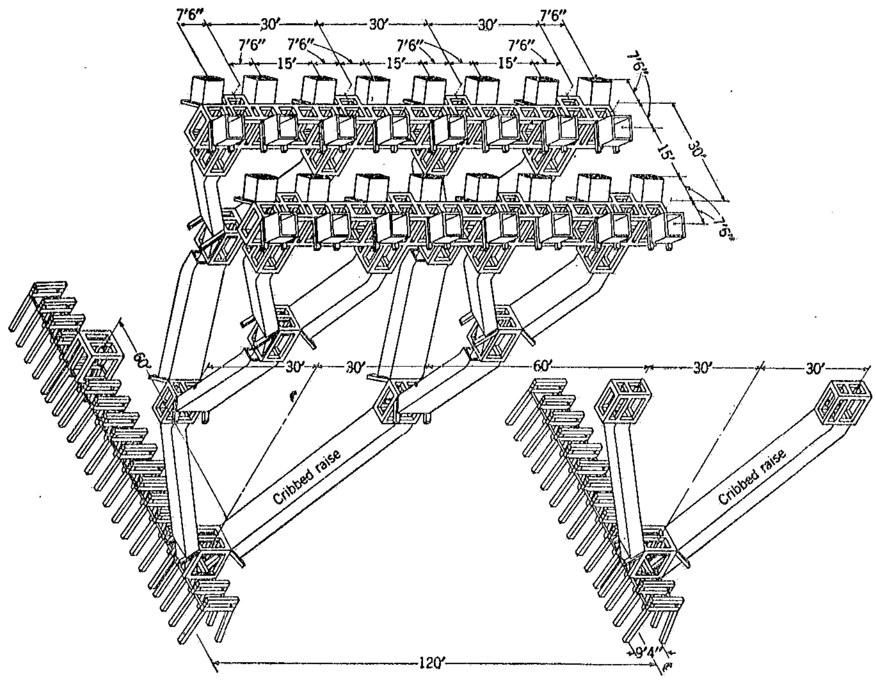


FIGURE 42.—Method of mining, Consolidated Coppermines.

PORPHYRY

The Porphyry mine of the Copper Queen Branch, Phelps Dodge Corporation, is at Bisbee, Ariz. The main porphyry ore body was mined by the open-cut method and later by glory holes. In 1925 the mining of an associated ore body which extended to a depth of 2,000 feet was begun by an undercut block-caving method through the Sacramento shaft.[23] Levels were run at 100-foot intervals. The ore was milled at the concentrator, which is about 2 miles from the Sacramento shaft; the mill was built for handling the ores mined by the open-cut method. Concentrates were smelted at the company smelt-

[23] Gardner, E. D., Undercut Block-caving Method of Mining in Western Copper Mines: Inf. Circ. 6350, Bureau of Mines, 1930, 43 pp.

er at Douglas. Figure 43 is a perspective of the general method of mining. The Porphyry mine was shut down in 1932; later, half of the concentrator was dismantled.

OLD DOMINION

The Old Dominion mine is at Globe, Ariz. The principal ore bodies occur as shoots in a vein that ranges from 2 to 60 feet in width and dips

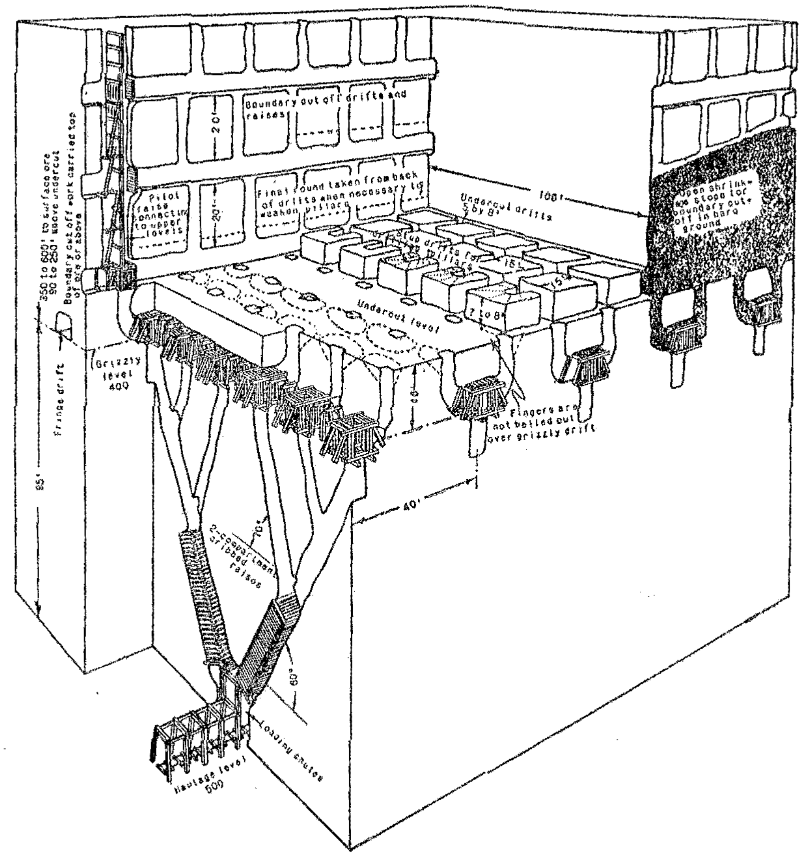


FIGURE 43.—Perspective of undercut block-caving method, Copper Queen mine.

60°. An independent ore body on the 2,300-foot level, which has a maximum cross section of 200 by 350 feet and a thickness of 100 feet, was worked by a caving method that was adapted from the Morenci slide method previously mentioned. The Morenci slide was also used in wide parts of the vein.[24]

The percentage of ore obtained by the various mining methods used at the Old Dominion in 1929 was as follows:

Caving	57.0
Square-setting	21.3
Shrinkage	14.1
Top slicing	3.8
Open stopes	3.8

[24] Shoemaker, A. H., Mining Methods at the Old Dominion Mine, Globe, Ariz.: Inf. Circ. 6237, Bureau of Mines, 1930, 21 pp.

The Morenci slide has an advantage in relatively thin ore bodies over the caving methods used in the porphyry copper mines in that less development work is required for an equal area of stope. The general lay-out used in the ore body on the 2,300-foot level at the Old Dominion mine is shown diagrammatically in figure 44. The ore was cut off on all sides by shrinkage stopes. Transverse stopes, usually about 40 feet high, were run to prevent the weight of an overthrust from settling on advanced development work and to prevent

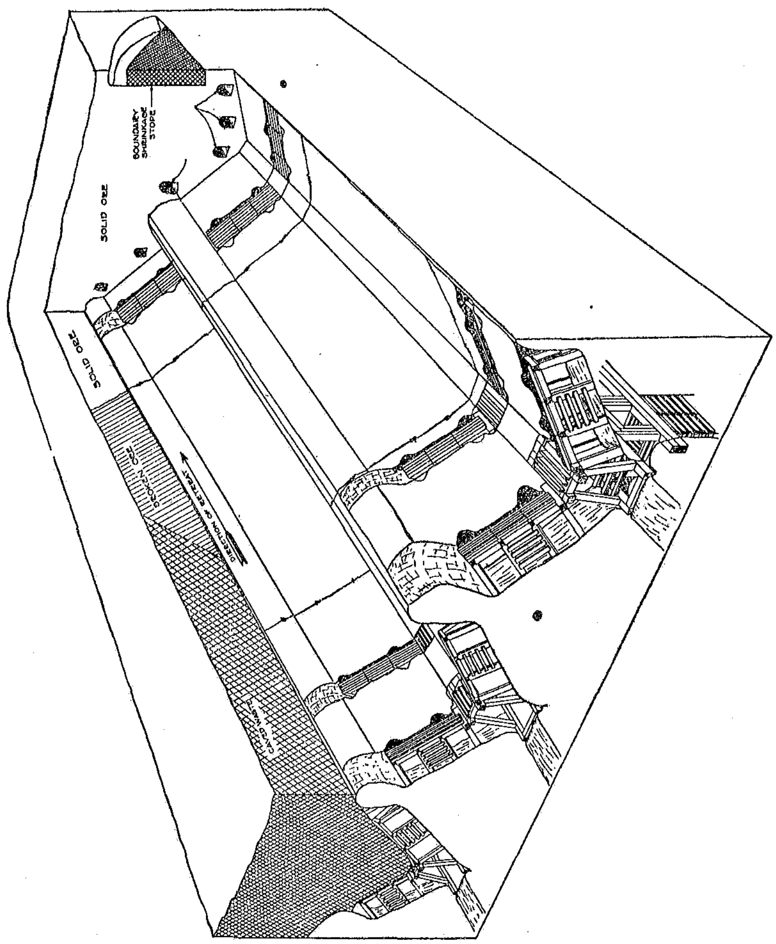


FIGURE 44.—Perspective drawing showing general lay-out for mining 2,300-foot level ore body, Old Dominion mine.

arching of the back over a large area when the section was drawn. Drifts were run through the ore body on 40-foot centers. Timbered slides 10 feet apart, at an inclination of 36°, and about 18 feet long were then put up from the drifts.

Transfer raises from the 2,400-foot level served four slides that furnished the drawpoints for a 40- by 40-foot section. The boundary between broken ore and waste was a plane dipping about 50° in the opposite direction from that in which the stope was advanced.

UNDERCUT BLOCK-CAVING PRACTICES

The principle of mining by the undercut block-caving method has been stated as follows:[25]

The main features in mining by an undercut block-caving method are (1) breaking of the ore by the force of gravity, thereby eliminating drilling and blasting in stopes, and (2) drawing the ore under control through raises into haulage cars, thereby doing away with the necessity for shoveling.

The method is characterized by an intensive production of ore from a relatively small area, which permits repetition of a number of simple operations. A thorough study of this work makes possible standardization of nearly all operations and efficient working of the mine.

Caving is induced by undercutting and cutting off the blocks from surrounding ground. The ore is drawn through openings uniformly spaced below the block.

By undercutting is meant the blasting and partial or complete removal of a layer of ore across the base of the column to be mined so as to permit the ore above to cave and break by gravity. The undercutting practice is not uniform at all mines, and the methods in use have been developed mainly to fit the particular conditions existing at each place.

Undercutting is supplemented by either cutting off the block entirely from the surrounding ground or weakening it along its boundaries so as to assist the caving action and confine it to the block mined.

For perfect drawing the drawpoints should be closely and regularly spaced, but a compromise must be made between close spacing, which increases ore recovery and reduces dilution, and wider spacing, with resulting economy in preparatory costs and operating repairs. The character of the ground is the controlling factor in drawing practice.

In the original Inspiration ore body and at the Miami and Ruth mines the ore breaks finely and tends to pack and when drawn pipes vertically, with little spread beyond the drawpoints. This condition makes it necessary to space the drawpoints as close together as economy and the necessity for maintaining supporting pillars will permit. At Morenci, Ray, and at the Copper Queen, where the ore is coarser and harder, the drawpoints are larger and spaced farther apart.

Dilution is caused by the irregular movement of ore and capping toward the drawing points forming pipes of waste, which may reach a drawpoint in advance of the top boundary of the ore, or by waste being drawn in from the sides or ends of a block. Dilution may also be caused by the general infiltration of fine capping down through coarsely broken ore. Moreover, in all caving there is a gradual mixing of the ore and capping as the ore is drawn down. If properly drawn a large part of the ore comes to the chutes clean. After dilution starts the proportion of waste increases and that of the ore decreases.

In mining a block of ore by a caving method three things must be constantly watched and controlled:

1. The mining must be done in such a manner that excessive weight does not develop on the grizzly level. This is generally controlled by the rate at which the ore is pulled.

2. The ore should be broken sufficiently in caving to be subsequently handled without further breaking. As a rule, the slower the drawing rate the more the ore is broken; also the slower the drawing rate the more weight is likely to develop on the grizzly level. Generally a balance is maintained between the two to get a maximum efficiency.

3. The ore must be drawn in such a manner as to get a maximum recovery and minimum dilution with waste.

The point at which drawing of a block ceases depends upon the minimum grade of ore that can be handled and upon the value of ore that yields the best financial returns. If the plant capacity is large, drawing may continue until near the point of no profit. If not, to obtain adequate profits drawing must be stopped soon after dilution appears. The method is elastic in that by the sacrifice, for example, of 15 or 20 percent of the ore, the grade can be maintained at very nearly its original figure.

[25] Gardner, E. D., Undercut Block-caving Method of Mining in Western Copper Mines: Inf. Circ. 6350, Bureau of Mines, 1930, 44 pp.

All phases of undercut block caving are interrelated. The development workings are laid out as a whole; changes in one set of workings must be met by corresponding changes in others.

HEIGHT OF LIFTS

The height of lifts at the first mines to use the undercut block-caving method was 75 feet. Experience, however, has shown that satisfactory control of the downward movement of a block of ore can be maintained with higher lifts than were formerly thought practicable. In opening up new levels the tendency is toward higher lifts. As determined at Miami,[26] the saving in development costs continues to a maximum height of 300 feet; no substantial advantages are shown with heights greater than this.

BLOCKS AND PANELS

Two general plans for stoping are followed: (1) Panels along or across the ore body and (2) individual stopes of more or less definite length. Under either plan the ore bodies usually are laid out in panels to accommodate the haulage system. The panels are then mined as continuous stopes or cut into blocks, depending upon conditions. A panel must be wide enough to permit the ore to cave by its own weight when it is undercut but not wide enough to cause the transfer of excessive weight to the extraction openings. In weak ground, for instance, the ore caves readily, but weight is more likely to develop on the openings below. The width of the panels ranges from 87.5 to 200 feet. The length may be as much as 1,200 feet but depends on the size and shape of the ore body. Figure 45 shows a typical lay-out for mining an ore body by panels.

The individual stope method permits more places to be worked simultaneously, more flexibility in drawing operations, less congestion in haulage laterals, and more selective mining when two classes of ore from the same deposit such as leaching and concentrating ore are to be separated; moreover, better control over drawing is possible with the smaller block in which the ore is drawn down slowly.

With the panel system fewer haulage laterals need to be kept open and fewer grizzlies drawn at the same time; this results in a lower maintenance cost. Moreover, as the panel is worked from one end to the other fewer end cut-off workings are needed. The panel system has an advantage in hard ground with moderate lifts, because the shearing and bending of the ore as it breaks from the solid at one end tend to break the mass into smaller fragments than if it were drawn evenly in blocks.

Angle of contact between ore and capping.—In general, the angle of contact maintained between the top of the ore and the capping depends upon local conditions, such as height of the lift, size of the block, necessity of protecting a stope from side dilution, tendency of the workings under the drawing area to take weight, and dilution of the ore by an admixture of capping.

At some mines experience has shown that side dilution is less if the ore adjoining completed stopes is drawn last. To do this the angle of

[26] Maclennan, F. W., Miami Copper Co. Method of Mining Low-Grade Ore: Trans. Am. Inst. Min. and Met. Eng., Yearbook for 1930, pp. 39–86.

contact between the ore and overlying capping must be relatively steep.

A steep angle of contact is used where the mechanical strength of the ore is low or the ore tends to be plastic due to excessive fracturing

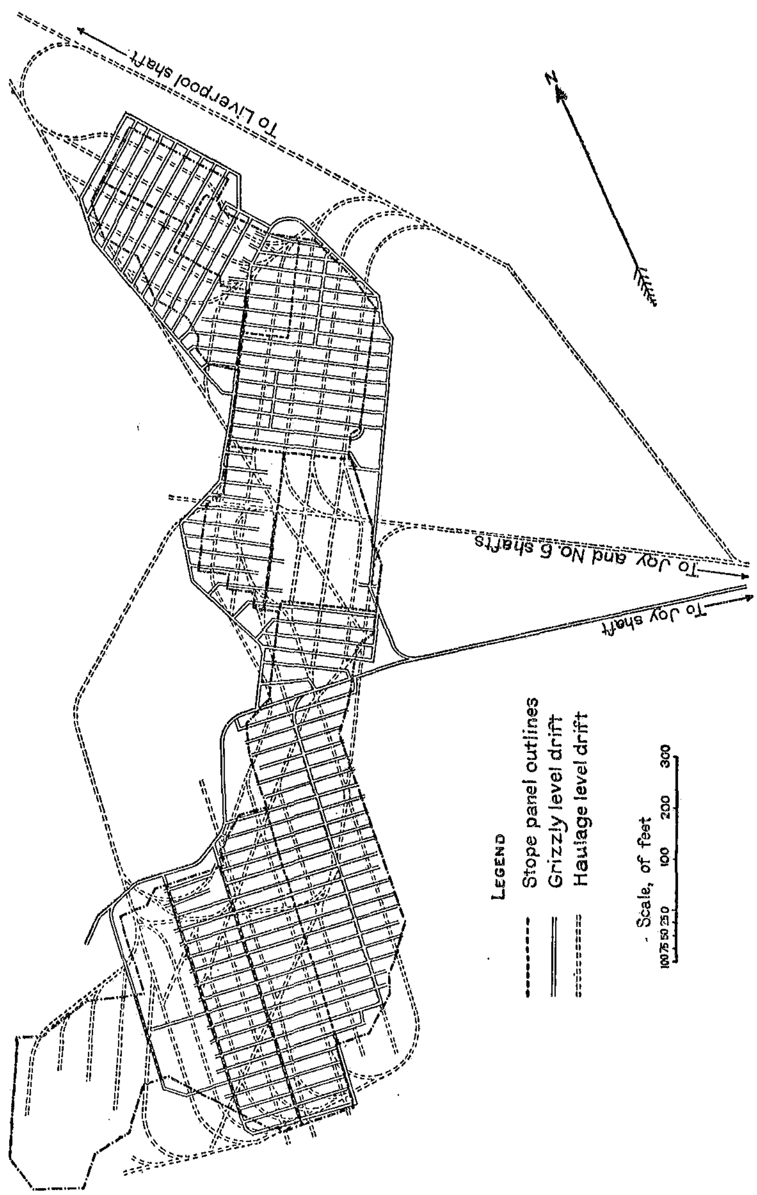


FIGURE 45.—Plan of grizzly and haulage levels, Morenci mine, Morenci, Ariz.

or a large proportion of clayey minerals. With a steep angle less weight develops on the extraction level, and the length of time necessary to keep any line of grizzlies open is less.

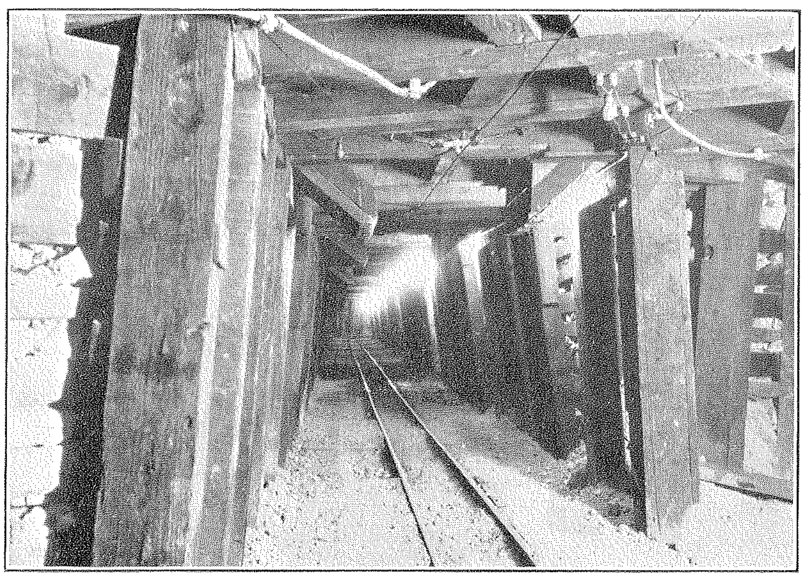


FIGURE 47.—Haulage drift, Morenci mine.

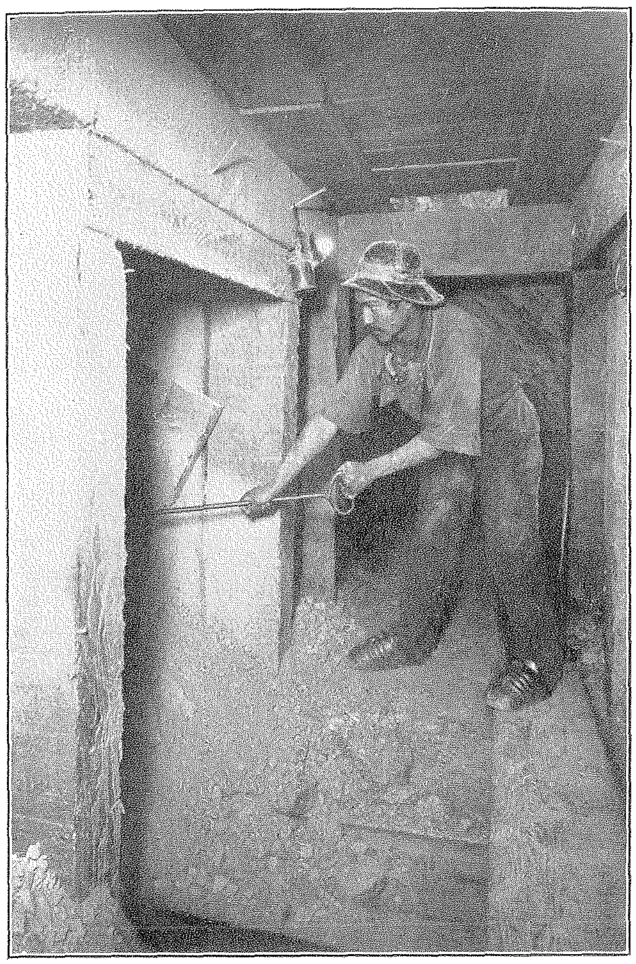


FIGURE 48.—Drawing ore from block-caving stope, Ray mine.

With extremely steep angles of contact dilution of the ore by the overlying capping is increased. Usually the angle of contact at each mine is determined by experience so as to balance loss due to dilution and repair costs due to weight.

At Ray the blocks are drawn evenly, and a horizontal contact is maintained after undercutting has been completed. At Inspiration the undercutting and drawing are begun at the side or corner of a block farthest from a completed section. The rate of drawing depends upon conditions in each particular stope, and no set angle of contact is maintained between ore and waste, although as flat a draw is desired as will fit the other conditions. At Ruth the angle of contact is kept as near 40° as possible. If the angle is steeper than this the admixture of capping with ore increases, and if it is flatter a longer time is required to mine a given part of the panel, which in turn entails additional expense in maintaining raises and drifts in heavy ground. At Morenci, where a panel system is used, the angle is 60° during the initial stage of drawing. This angle is then flattened to 50° or less in completing the stope.

DEVELOPMENT WORK

In general, development for mining by an undercut caving system comprises a haulage level, a system of transfer or gathering raises, a grizzly level, drawpoints, boundary cut-off workings, and an undercutting level. At most mines the drawpoints are spaced the maximum distance apart that will permit adequate control of the drawing; the other workings are then laid out accordingly.

Haulage drifts.—As this method takes large tonnages from relatively small areas efficient haulage is necessary. The general lay-out of the haulage system depends upon the size and shape of the ore body and its distance from the hoisting shaft or main haulage adit. The haulage system at the Morenci mine is shown in figure 45.

The ore is delivered to the haulage level through raises and loaded into cars through chutes. Haulage drifts are usually about 7½ by 9 feet in cross section. (See fig. 47.) The distance between haulage drifts or laterals is governed principally by the height of the grizzly or transfer level above the haulage level. As shown in table 22 the main haulage drifts range from 50 to 150 feet apart.

Transfer raises.—Transfer raises at the Inspiration, Morenci, Consolidated Coppermines, and Copper Queen mines are shown in figures 36, 39, 42, and 43. The transfer-raise system at Miami [27] is shown in figure 46 and that at Ruth in figure 41.

Transfer raises generally are run 4 by 4 or 4 by 6 feet in cross section. Where the ground will not stand, the raises are cribbed. As grizzly drifts usually are not run at Ruth, access to the drawpoints is through the raises, and manways are maintained in each.

As the distance between the haulage and grizzly levels increases, the distance between the haulage laterals is increased. This results in a decrease in the amount of haulage-level drifting and an increase in the footage of transfer raises needed to serve the same undercut area with the same raise interval on the grizzly level. Usually, raising costs less than drifting, which reduces the development cost of the

[27] Maclennan, F. W., Miami Copper Co. Method of Mining Copper Ore: Tech. Paper 314, Am. Inst. Min. and Met. Eng., 1930.

stope. With wider spacing of the haulage drifts and the consequent longer raises there is an increase in ore-storage capacity in the raises and less danger in blasting one end of a raise to the workmen at the

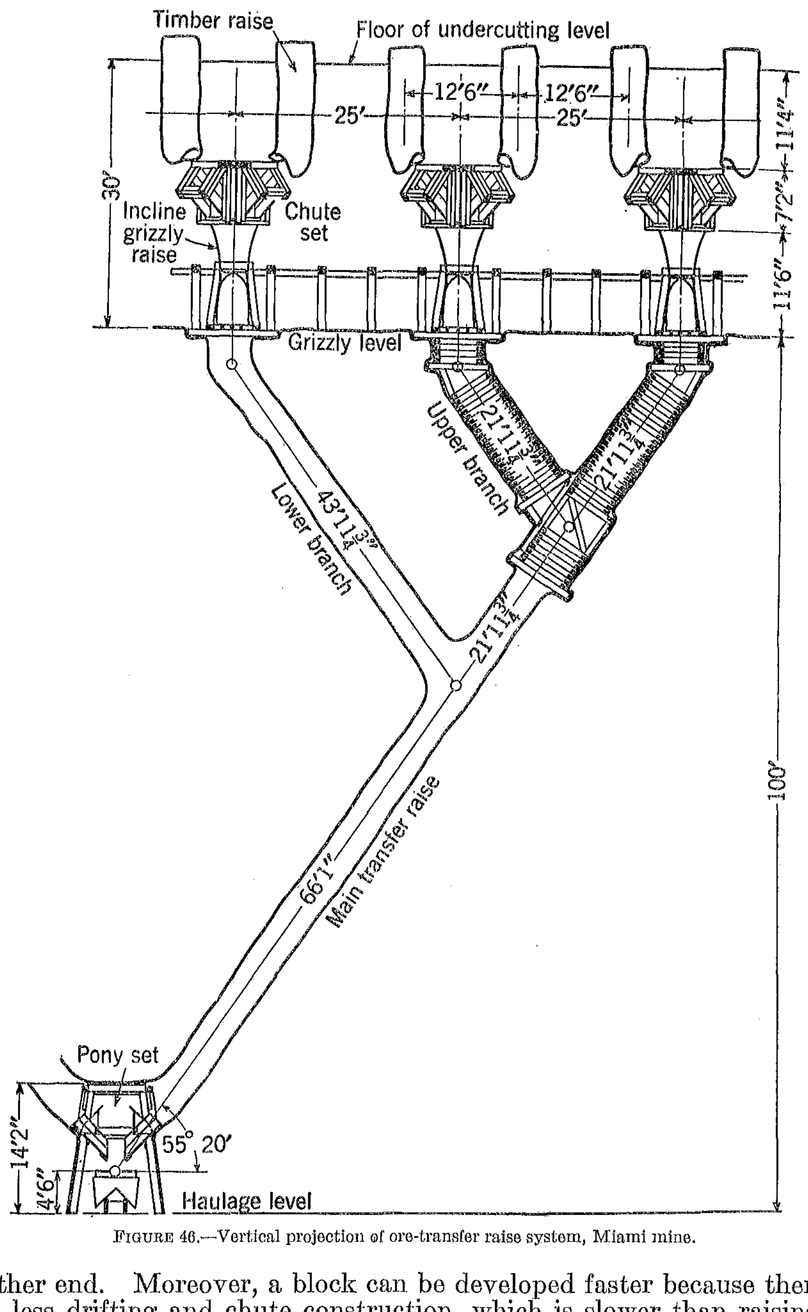


FIGURE 46.—Vertical projection of ore-transfer raise system, Miami mine.

other end. Moreover, a block can be developed faster because there is less drifting and chute construction, which is slower than raising. A smaller distance between the grizzly and haulage levels shortens and simplifies the transfer-raise system and provides a more accurate means of measuring and distributing the tonnage drawn. Increasing the number of haulage drifts reduces train congestion.

Grizzly levels.—A main level is usually run at the altitude of the grizzlies for access to the grizzlies and undercut workings. At the Ruth mine, however, the grizzly level is reached only through raises from the haulage level below. Fringe drifts usually are run along one side of a panel or entirely around the blocks; the grizzly drifts are turned off at right angles. One fringe drift may serve two panels. The grizzly level is the center of operations after production starts; crushing of these drifts and consequent repair work interrupt production and may be very costly. Openings on this level, therefore, are made as small as practicable to withstand pressure better; the level likewise is spaced far enough below the undercut level, with due regard to the increased cost of draw raises, to eliminate excessive pressure on it from undercutting and caving operations above. The ore drawn from the stopes above passes through grizzlies placed over the gathering raises. The flow is controlled by simple gates at the bottom of the drawpoint. The spacing of grizzly bars is mainly a measure of the fineness to which the ore breaks in caving at the different mines. It ranges from 18 inches at Morenci, where the ore breaks coarsely, to 8 inches at Ruth, where the ore generally breaks down into fragments the size of sand and gravel and tends to flow.

Drawpoints.—After the ore is undercut and caved it is drawn through short "finger" raises. The drawing of the ore is controlled from the lower end of these raises. Weight at the drawpoints is resisted most effectively by leaving as much unbroken ground as possible about the grizzly drifts to act as pillars. Wide spacing of the finger or draw raises, however, is limited by the requirements of good drawing, which prescribe spacing them as closely as practicable. The best spacing of drawpoints for each ore body usually has to be worked out by trial to balance properly close control of drawing, cost of development work, and maintenance of workings. The tops of drawpoints are evenly spaced at the undercutting level; as shown in table 22 this spacing ranges from 12½ by 12½ feet to 20 by 20 feet. In general, drawpoints must be close together in a soft ore that tends to pack or "chimney."

Drawpoints are belled out at the upper end; the lower ends of the raises usually come down in pairs, on either side of the grizzly drift at the top of the transfer raises. (See figs. 38–43.) This is called the grizzly-control method. Figure 48 shows ore being drawn from a drawpoint at the Ray mine.

In the original Inspiration ore body and at Miami four drawpoints on finger raises enter the sides of a square-set above the grizzly level. Figure 49 shows the spacing of drawpoints at Miami. The ore from eight points is drawn through each grizzly with the chute-set method and from only two with the other. The chute-set system has the advantage that closer spacing of drawpoints is possible, which allows closer control of the ore. Its principal disadvantage is that two men are required to draw the ore, one in the chute and one at the grizzly. Other disadvantages are that the man in the drawset works in a constricted space where he is more likely to get caught by a run of ore and that supervision of drawing is more difficult than at the grizzly level. Moreover, the drawsets are expensive to maintain.

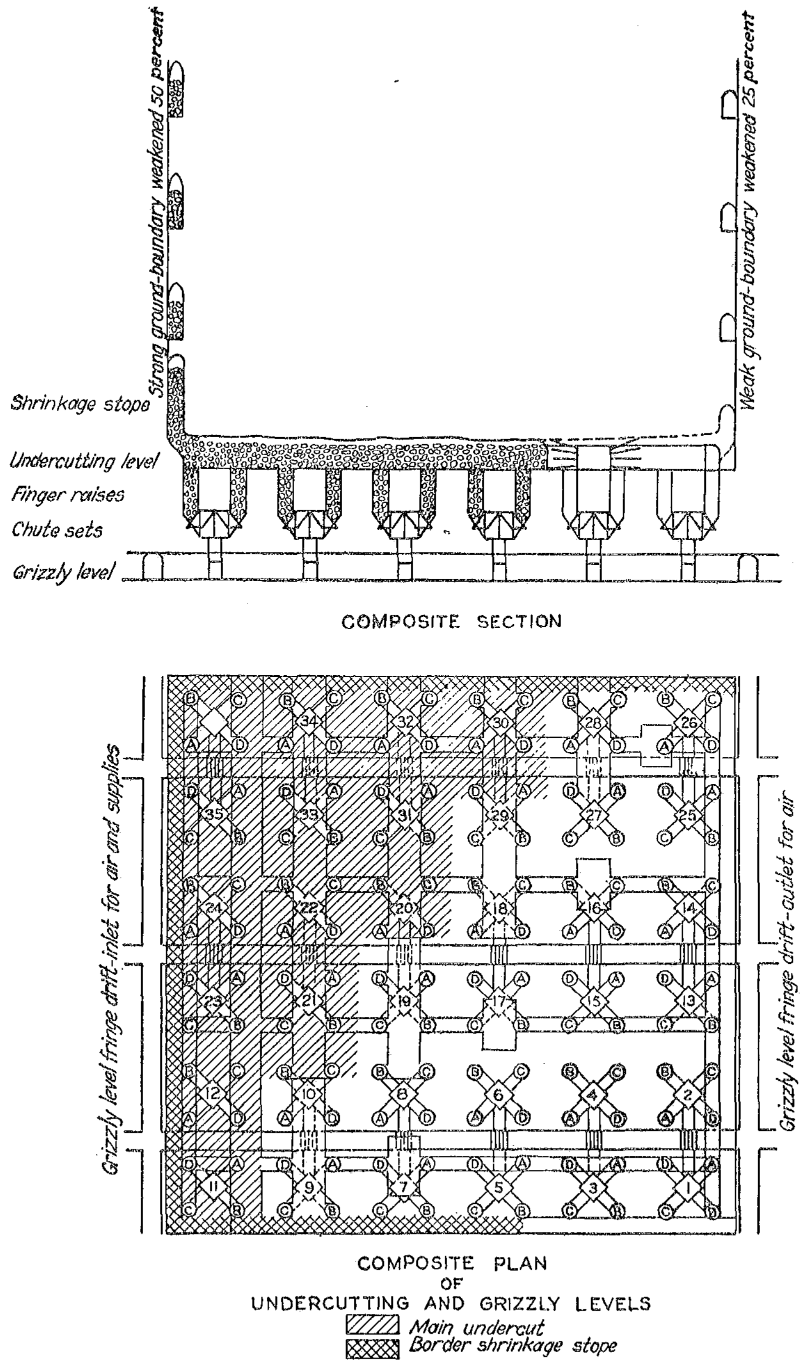


FIGURE 49.—Method of undercutting, Miami mine, Miami, Ariz.

By the summer of 1936 the Inspiration mine used only the grizzly-control method. It was reported that recoveries of ore in the blocks mined by this method were equal to the results obtained with chute-set control. Apparently, the closer supervision of drawing made possible by having the operators working on the grizzly level compensated the disadvantage of wider spacing of the drawpoints. The drawpoints in the first block of oxide ore at Miami were spaced 17½ by 17½ feet apart, and control was at the grizzly level. The experience at Miami, contrary to that at Inspiration, was that the loss of ore because of less effective control more than balanced the saving in the cost of development work; in subsequent blocks the former 12½- by 12½-foot spacing with chute control has been used.

Cut-off workings.—Before the ore is mined in blocks it is necessary to devise a method to cut it free from adjoining ground and guide the lines of shearing to the top of the ore in the desired planes. The method used to cut off the blocks or panels depends on the character of the ore and the height of the lift. Low blocks in strong ground usually must be cut off almost entirely on all sides by narrow shrinkage stopes. On the other hand, the boundaries of blocks with high lifts in fractured ground only need to be weakened. The main object of cut-off workings is to form vertical planes of weakness on the sides of the blocks to prevent the backs of the stopes from forming self-sustaining arches; however, it is necessary sometimes to avoid weakening the boundaries to such an extent that the back would drop out in huge fragments or possibly in a single large block. A complete cut-off occurs on the sides of a block adjoining finished stopes.

Cut-off workings provide a means of acquiring knowledge of the physical character of the ground which is valuable during drawing operations. The knowledge to be gained, of course, depends upon the extent of these workings. Samples taken in boundary cut-off workings show the grade of the ore in the panel or block, which also aids in controlling drawing operations.

At Ray both end and side cut-off shrinkage stopes around new blocks are carried up to the capping or to a previous mined area. In blocks with high lifts the cut-off stopes are run in two sections; the upper half is run from a sublevel. (See fig. 39.)

At Morenci, where the character of the ore approaches that at Ray, shrinkage cut-off workings in the upper lifts were first started by driving raises 140 feet apart to connect with an intermediate level 70 feet above the undercut level. (See fig. 40.) In weak ground, shrinkage stopes were then carried up 35 feet; at this elevation, sections 50 to 60 feet long, situated midway between the pilot raises, were left intact. The shrinkage stopes in sections 40 to 50 feet on each side of the pilot raises were then carried up as high as the intermediate level. In strong ground the boundary was weakened still more by driving the stopes higher. As the character of the ore varies greatly, the extent of the cut-off workings is left to the judgment of the operator. It is better to cut out too much, even at the risk of a sudden collapse, than too little so that the block hangs up indefinitely.

At Inspiration raises are run at the corners of each block. Other raises are put up on the boundary about 110 feet from the corners. Drifts 30 to 40 feet apart vertically are run around the block. These boundaries may be further weakened if necessary by blasting stope rounds in the back of the drifts. (See fig. 38.)

In the sulphide ore body at Miami, the boundaries of the block are weakened by driving boundary drifts 7½ feet high completely around the stopes at 45-foot vertical intervals. (See fig. 49.) Raises are run at all four corners of the stope to avoid lateral arching at these points. The backs of the drifts are shot down, thus weakening the boundary plane 33⅓ percent. In hard ground the boundary planes can be weakened still more by further drilling and shooting in the back of the drifts, which is inexpensive. Actual fracturing and caving down of the stope backs may be observed at the corners from the boundary caving drifts at successive elevations as they progress upward.

At the Copper Queen the blocks are cut off by a series of drifts and raises which leave unbroken pillars 20 feet long and 15 feet high along the boundary. (See fig. 43.) In hard ground the pillars can be blasted so as to make an open shrinkage stope.

Methods of undercutting.—Stopes are usually undercut by running a checkerboard system of drifts and crosscuts and blasting the pillars, by belling out the finger raises to intersect, or by a combination of both methods. In hard ore a system that will insure a complete undercut should be used; otherwise the back may not cave. Moreover, a small area of unbroken ore under a stope may transmit excessive pressure from the broken ore to the workings below. Where the ore is mined in panels, transverse shrinkage stopes may be run across them to induce caving.

At Morenci the undercut, which follows just behind the boundary shrinkage workings, is begun by running one or two drifts on the undercut level lengthwise with the panel across several grizzly lines for convenient access to the undercutting drifts. Undercutting drifts 4 by 6 feet in section, locally termed "dog holes", connect the top of the belled-out finger raises both laterally and longitudinally, thus forming a gridiron of drifts between which are rectangular pillars about 10 by 14 feet in plan. (See fig. 40.) Certain fingers are driven in advance to handle supplies for the undercut section and to serve as ladderways from the grizzly level, thus insuring a safe horizontal retreat for the men on the undercut level. The checkerboard is developed, beginning at the area already caved, by connecting one or two rows of fingers across the panel and proceeding in the direction of the stope retreat. After the drifts are completed the pillars are drilled with hand-held plugger drills. The backs of all "dog holes" are drilled to a depth of 6 feet in fan-shaped rings of stoper holes at 4-foot intervals. These holes are fired simultaneously with the pillar shots. The extra cut assists in starting caving and with the open finger raises provides room for the expansion of the broken ore. Two or three pillars are usually blasted down at a time, beginning in the heaviest ground and progressing across the stope. Where conditions permit blasting is delayed until the pillars begin to show strain. When a pillar begins to take weight before blasting immediate caving of the back is assured. Blasting is done with no. 8 instantaneous electric detonators, which have eliminated difficulties arising from cut-out holes and hazards from loose powder in the ore drawn from the stope. To open a new panel 112 feet wide, undercutting is necessary through a length of 84 to 196 feet to break up possible arches or cantilevers and to start the ore caving freely, even though the new panel may lie against the caved waste filling in a completed section.

At Inspiration undercutting drifts are run across the block at 25-foot intervals, as shown in figure 38. These drifts are widened at short intervals to allow the use of long steel in drilling rounds in the intervening pillars. Concurrently the tops of the finger raises are belled out. The pillars are then drilled and shot a section at a time. In the Live Oak division undercutting was done by connecting the finger raises with a series of inclines, as shown in figures 33 and 38. The intervening pillar between each row of raises was then drilled with stopers and shot a section at a time; this system of undercutting has been discontinued.

At Miami the undercutting level in the sulphide ore body is started 30 feet above the grizzly level from the tops of the finger raises, one or several of which are utilized as supply raises and manways; those nearest to the work are used for drawing off the broken ore. The undercutting level and its relation to the grizzly level, chute sets, and drawpoints are shown in section in figure 49. The undercutting level is opened by driving four drifts of small cross section parallel to the grizzly drifts through every third line of drawpoints, 37½ feet apart and equidistant from each side of the central grizzly drift. These drifts are connected at both ends by fringe drifts. Undercutting is started by putting up a shrinkage stope two or three rounds from one of the fringe drifts. This shrinkage stope is usually carried along the side boundaries and one end of the stope as undercutting of the main body progresses. The main body is undercut from drifts 8 feet wide, which are driven at right angles to the small opening drifts. Work is carried back diagonally, as shown cross-hatched on the plan. (See fig. 49.) The large drifts are not driven until needed for blasting to avoid premature crushing. Usually ore drawing is not started in the sulphide at Miami until a stope is completely undercut, except when it is started to avoid excessive packing of soft ore and to ease weight on the chute sets and grizzly level below.

The oxide ore body at Miami is stronger structurally than the sulphide ore body, and a modified method of undercutting it has been adopted (1936). Undercutting drifts 8 feet high and 10 feet wide are advanced lengthwise through the block on 26-foot centers by means of two mounted machines in a heading. Meanwhile, intervening 21-foot pillars are pierced lengthwise on the center lines by 5- by 6-foot drifts. Undercutting is completed by drilling rows of holes 3 feet apart in the sides and top of the 8- by 10-foot drifts. The top holes are 10 feet long to pull down the arch; side holes are 6 feet deep. Eleven rows of holes are blasted at a time in each drift; 11-delay electric detonators are used. The pillar shots break into the 5- by 6-foot drifts which are run solely for this purpose.

The first three blocks in the oxide ore body did not cave when undercut. Caving was initiated by blasting a number of powder pockets made from drifts run into the bolck from the boundary cut-off workings. The powder drifts of the first block (150 by 300 feet) were 60 feet above the undercutting level. Seventeen pockets were provided, but all were not blasted. One to three blasts were set off at a time over a period of several months. Each charge consisted of 2,500 pounds of 50-percent straight gelatin dynamite primed with 50 pounds of 60-percent straight dynamite. Caving was started in the third block (150 by 172 feet) by shooting 10 charges 33 feet and 14 charges 62 feet above the undercut level in one blast of six delays. The charges comprised 15 to 38 boxes of 60-percent-strength gelatin dynamite, each primed with a box of 40-percent straight dynamite. A total of 29,150 pounds of explosive was used. Both blasting levels were laid out and blasted as a unit; alternate pockets on each level were shot in order.

If the first blast did not induce caving, powder drifts were run higher up in the block and the blasting procedure was repeated.

At the Copper Queen a gridiron system of drifts and crosscuts 7 to 8 feet high was run 19 feet above the bottom of the grizzly level, leaving pillars 15 feet square. Concurrently the tops of the draw raises were belled out, as shown in figure 43. Stub drifts were then run into each pillar to permit the use of long steel, and the pillars were drilled with a fan-shaped round which when blasted completely cut the pillar. The figure shows the stub drifts extending into the pillars at right angles from the undercut drifts; this practice has been modified since the figure was drawn, and the stubs are now run in at a corner of the pillar adjoining a raise. By this change one round less has to be shoveled out by hand.

At Ruth five or six series of raises are completed before undercutting is started. A raise series is a row normal to the haulage drift, as shown in figure 41. The tops of the two lines of finger raises in a raise series are connected by drifts; then crosscuts are driven to connect the tops of each alternate pair of finger raises in the other direction. The next step is to drill and blast the pillars remaining between the crosscuts and drifts, thus allowing the block of ore to cave. In the undercutting workings the openings stand with little timber during the few days that they must remain open. If the ground is heavy undercutting may comprise rounds of vertical and horizontal holes drilled from the top of each finger raise without drifts and crosscuts being run. Little difficulty is encountered at the Ruth mine in starting the ore to cave.

The method of undercutting at Ray, unlike that at other porphyry mines, is a development of the original stope-and-pillar method of mining; it consists of undercutting the block by means of shrinkage stopes. (See fig. 39.) The first series of shrinkage stopes, called locally "undercut stopes", is mined up and across the block to a height of about 43 feet above the grizzly level. They are about 6 feet wide, except that the final round is fanned out to a width of 17 feet. Between the undercut stopes and directly over alternate rows of throat raises, similar but smaller stopes are carried to a height of 35 feet above the grizzly level. The final round is likewise drilled as wide as possible and when blasted breaks into an undercut stope on either side completely cutting off the pillar and allowing the ground above to cave.

At the Old Dominion mine, where the Morenci slide system is used (see fig. 44), undercutting is begun after the inclines are run. The first step is to cut the draw holes, which comprise a series of untimbered raises beginning at the side of the draw-set inclines and extending upward at an angle of 45° for about 9 feet. These raises apex midway between the slides. A cut at least 4 feet long is then taken out directly above the inclines. The next step is to blast another round in the back of the draw holes. Finally the pillars between the draw holes are drilled and blasted. A pillar from apex to apex is blasted down as a unit, and all of it must be cut to reduce the probability of a large section of unbroken ground settling on the slides. There must be enough room to accommodate the swell from the first slough of ore so that no direct weight will be thrown upon the timbers. A draw hole from the slide beyond the pillar being shot down should be completed before blasting. This gives a safe entrance from which to observe the effect of the blast. Usually it is possible to bar down, drill, and blast the back a second time. This would be impossible if this safe entrance was not prepared in advance. The ore is drawn into the slides whence it is pulled through the grizzlies into the ore raises. The flow is controlled by lagging placed across the draw sets in the slides.

The amount of ore broken in development work in side cut-off workings and in undercutting at the Morenci, Inspiration, Miami, and Copper Queen mines is about 10 percent of the total in an average block 200 feet high. Increasing the cut-off workings at Miami from 25 to 50 percent of the area around the block would increase the total amount of ore broken by drilling and blasting to about 14 percent. At Copper Queen a complete cut-off around an average block would increase the amount broken by blasting to about 19 percent. At Ray, where a different method of undercutting is used, about 20 percent of the ore in a block with a 200-foot lift is broken with explosives.

DRAWING PRACTICE

As stated before, the two main objectives in drawing ore when a caving method of mining is used are (1) to draw a maximum of ore with a minimum dilution by waste and (2) to regulate drawing so as to prevent development of damaging weight on the extraction workings. Ore drawing in practice is a compromise between these two objectives.

After the ore is undercut, it is drawn slowly at first until it starts to cave freely and then at the rate that best suits the nature of the ground and the requirements for ore. If drawing is too slow, weight will accumulate gradually on the grizzly drifts, whereas rapid drawing will relieve that pressure. Too rapid drawing, however, may bring the ore down to the chutes in fragments too large to be economically handled. The rate of drawing will also depend upon the readiness of the ore to cave; too rapid drawing may cause dangerous cavities to form above the broken ore; if drawing is too slow the swell of the broken ore will partly support the back and retard caving. Regardless of rate, drawing should be continuous to prevent packing of the ore. The ore should be drawn uniformly, particularly at first, to assure a satisfactory recovery. The contact between the ore and the broken overlying capping should be an even plane, whether horizontal or inclined. If early drawing is not uniform the mass of ore may cave along some plane of weakness, causing chimneys or cavities that may extend into the capping and allow it to mix with the ore. After the main mass of ore has been thoroughly broken and the block has caved to the surface irregular drawing is not as serious as it would have been earlier.

Before drawing is started in a stope vertical sections usually are made which show the expected tonnage and grade from each drawpoint; these are used in controlling the draw. In drawing, the ore is worked through the drawpoints onto the grizzlies, where fragments too large to go through are broken either by hand or by block holing. No bulldozing is done on the grizzlies because it might damage the timber of the grizzly sets. At Ray and some other mines no blasting is done on the grizzlies for the same reason. If the ore is well broken in caving it can be started to run in the drawpoints by barring. Occasionally a large boulder will block a draw raise, in which case it is drilled and blasted. In hard-rock mines the raises are funneled to get large fragments as near the grizzly level as possible so that men will not have to go into the raises to drill holes. Moreover, openings are cut in the sides of the grizzly drifts to connect with the finger raises. At all mines the draw raises will hang up occasionally because large boulders arch over the raise or ore packs between drawing periods. Hung-up chutes that cannot be brought down by barring are blasted with charges of dynamite and electric detonators called bombs.

At Morenci the ore flows through 4- by 6-foot openings in the side of the grizzly drifts to the grizzlies where large pieces are broken up by sledging or block holing. There is plenty of room at the grizzlies for a man to work and step away to avoid a surge of ore. The grizzly bars are 18 inches apart; wide spacing reduced sledging and block holing but necessitated a change in the haulage, hoisting, and coarse-crushing equipment to accommodate large boulders. The advantage of wide spacing is reflected in chute-tapping efficiency, which had increased by 1928 to 120 tons per man-shift from an average of 37 tons per man-shift with 10-inch grizzlies. Before the introduction of wide grizzlies it was occasionally impossible to draw the ore as fast as stope conditions dictated. Drawing can now be increased to 1,200 tons per day from an area of 10,000 square feet whenever it is desirable to do so; this means caving at a rate of 1½ feet vertically per day. The minimum draw in areas that are being held back for purposes of stope control is 12 tons per finger.

At Ray individual blocks are drawn separately, as each block is cut off on all sides by cut-off stopes or finished sections. Undercutting is carried on progressively from one end of the block to the other and is followed immediately by drawing on a limited scale to make room for the expansion of the ground above as it caves. This gradually removes the support from under the main mass of ore and by setting up a cantilever action causes it to slough and break up. As soon as the entire block is undercut the standard procedure of uniform drawing is begun throughout the block. Figure 48 shows a miner working at a drawpoint.

At Miami drawing of ore is started immediately after a stope in the sulphide ore body is undercut. It is usually slow work at the beginning, as the first ground caved generally arrives at the chutes in fairly large pieces. Sometimes the stope back hangs up and drops only a little ore for several weeks, but this is unusual. In 1929, to maintain the normal daily production of approximately 18,000 tons of ore, it was necessary to have 13 or 14 stopes on the drawing schedule. A tapper crew of two men, blasting five to eight times and drawing approximately 400 tons, usually drew from 12 to 15 finger raises per shift. Normally 60 tons was drawn in rotation daily from each chute listed on the draw orders. The regular routine was varied, however, by the presence of capping in a chute; need for repairs; weight on the timbers, which required the chutes to be drawn to relieve this weight whether they were listed to be drawn or not; variation in drawing to meet requirements as to grade of ore; and changes to maintain proper distribution to the haulage level for economical operation of the trains.

Drawing was continued from each chute until the grade of the ore dropped to a point where it was not sufficiently profitable. This usually did not happen until after all of the expected ore had been drawn. The cut-off point was fixed mainly by the minimum profit required, which depended on the cost of operation in the particular stope, market price of the metal, capacity of treatment plant, and total profit requirements. It was also affected by whether or not there was another lift of ore to be drawn from beneath it. If there was, the grade of ore at which drawing stops might be higher, as there would be another opportunity to recover the ore from below

The Ruth ore tends to chimney; when this happens at a drawpoint, caving is started by drilling holes around the base of the pillar from consecutive raises. With the 12½-foot spacing, it is always possible to start caving of the pillar when the ore chimneys, thus insuring maximum recovery of ore. If greater spacing of raises were used, the pillar of ore could not be attacked without considerable danger and expense. Better extraction is obtained by drawing slowly over a large area where possible rather than by drawing hurriedly from a small block, as the plane of contact between the ore and capping is kept at a more uniform angle. In drawing over a large area, however, the broken ore must stand for a longer time before it is won, thus increasing maintenance costs and the tendency of the ore to pack and hang up. A medium should be maintained between the opposing factors.

At Inspiration the area of a block farthest from previously caved ground is undercut first and drawing is begun. If two sides of the block have been mined previously, undercutting is done on a diagonal. About 40 percent of the block is drawn by the time undercutting is completed. By this procedure it is considered that dilution from the completed stopes is kept at a minimum. Drawing is done from as few lines of grizzlies as are necessary to maintain production.

Ore recovery and dilution.—As previously stated, it is impossible to extract all the ore in a block by a caving method or to prevent some dilution with waste. Recovery is usually calculated both in tonnage and metal content. Tonnage recovery is the ratio of the tonnage drawn from a block (extraction) to the tonnage estimated as contained in the block (expectancy). Metal recovery is the ratio of pounds of metal drawn to pounds of copper expected.

The accuracy of recovery calculations depends upon the accuracy of sampling and estimating the ore in the block and recording the grade and tonnage drawn from the block. An apparent extraction of more than 100 percent of the ore and nearly 100 percent of the copper is frequent. This may be due to underestimating the grade and tonnage in the block or to disregarding copper in the capping or other material with which the ore is diluted. For instance, if 1-percent copper is the cut-off point for profitable ore and considerable capping is pulled that runs 0.8 percent copper, the copper in the diluting material may equal that in the ore lost. At most mines all faces around a block are sampled carefully, and a correction is made for grade and tonnage after all development work in the stope has been done preparatory to drawing. A more reliable calculation of recoveries can therefore be made when such figures are used than when the estimate of tonnage and grade is based entirely upon the original drill-hole sampling.

In mining over 40,000,000 tons from completed sections at Ray records indicate that prior to 1929 dilution had approximated 10 percent. At Inspiration, where plant capacity is adequate, ore reserves include material of as low a percentage of copper as can be mined profitably. Under these conditions a minimum efficiency of 75 percent is figured, where efficiency is defined as percentage of copper recovered divided by percentage of total tonnage recovered. The recovery of the estimated tonnage in 1929 was more than 110 percent, with a recovery of over 85 percent of the metal content.

At the Consolidated Coppermines a 97.7-percent recovery of ore and a 97.8-percent recovery of copper were made in mining a block of 2,000,000 tons.

In mining 792,000 tons at the Porphyry mine of the Copper Queen, the tonnage recovery was 138 percent, copper extraction 115 percent, and recovery of grade 86 percent.

At Miami, Maclennan reports: [28]

> Up to date 13 stopes have been completely drawn. Nine of these were original stopes surrounded on all four sides by solid ground and four were pillar stopes, two of which were adjoined by broken waste on two sides and one end and two adjoined waste on one side and one end. These 13 stopes were estimated to contain 11,038,070 tons of ore assaying 1.026 percent copper. They produced 12,710,378 tons, assaying 0.912 percent copper. The extraction of more than 100 percent of the estimated copper content of the ore may be due to drawing some of the partition ore and to copper not included in the estimate, coming from overlying capping or gob.

The percentage extraction was 115.15 of the tonnage, 88.93 of the grade, and 102.40 of the copper.

OPEN STOPES

The open-stope method is used in ground that will stand without support other than pillars or casual stulls over spans the size of stopes being worked. In small ore bodies application of the system may be very simple; in large deposits, however, stoping must be done according to prearranged plans if the best results are to be obtained. Pillars left between open stopes may be mined by other methods. Frequently a choice must be made between open-stope and shrinkage-stope mining. Open stoping is generally preferred in narrow veins, as broken ore is likely to hold up in shrinkage stopes; rows of stulls are placed to hold platforms on which the drillers work. In similar deposits 6 to 20 feet wide shrinkage stoping may prove the most economical method. In wider stopes, however, open stoping with sublevels or inclined raises to provide points for attacking the ore would probably be used, unless the walls required the support afforded by broken ore in the stope.

Ground that can be mined by open stopes does not, of course, require filling as a part of stoping. Small openings usually fill themselves if the walls cave subsequently and present no special hazard. However, large open spaces in a mine nearly always are sources of danger; caving may damage other workings or the surface, and as depth is attained disastrous rock bursts may occur. Worked-out open stopes may stand for years but may cave when the walls have been weakened by other workings or by the failure of pillars. Where failure or caving is gradual, the danger of rock bursts is at a minimum.

[28] Maclennan, F. W., Miami Copper Co. Method of Mining Low-grade Ore Body: Trans. Am. Inst. Min. and Met. Eng., Year Book for 1930, pp. 39–86.

Completed open and shrinkage stopes may provide a convenient place for the disposal of development waste; they are then filled as such material is available. To eliminate danger from rock bursts, some open stopes may be filled with waste specially obtained for the purpose.

In flat or moderately dipping vein or bedded deposits a retreating system analogous to long-wall workings in a coal mine may be used. With such a system, the hanging wall subsides back of the active workings, and no extended areas are left open; thus danger from air blasts is avoided. A long-wall system may also be used in soft, heavy ground that could not economically be held open.

In many mines ore in the upper levels has been mined by open-stope methods. As depth has been attained other methods by which the walls are supported have been found necessary. The method has been used most in Michigan, where it has been used from the surface to a depth of 8,000 feet on the dip.

Sublevel stoping, a variation of the open-stope method, is feasible only in large ore bodies with strong walls. Use of this method brings about economies in stoping costs in wide ore bodies and increases safety of operation. A rather elaborate preparatory program is necessary, however, before a stope can be fully productive. Sublevel stoping is used at Ducktown, Tenn., where the method was an adaptation of one previously employed extensively in the iron mines of the Menominee range in Michigan; later it was introduced at the Flin Flon, Sherritt Gordon, Hidden Creek, and Noranda mines in Canada.

Production data of representative North American copper mines using open-stope methods are shown in table 23. Table 24 contains geological and general data on the same mines; the following pages describe the methods used.

TABLE 23.—*Production data at representative North American copper mines using open-stope methods of mining*

	Conglomerate	Kearsarge	Osceola	Mohawk	Quincy	Seneca
Location	Lake Superior district, Michigan	Lake Superior district, Michigan	Lake Superior district, Michigan	Lake Superior district, Michigan	Lake Superior district, Michigan	Lake Superior district, Michigan
Normal daily production of ore ... tons	2,800	4,000	3,000	2,000	1,500	600
By open stoping ... do	2,100	3,000	2,450	1,500	1,125	450
Total yearly production of ore:						
1929 ... do	841,000	1,270,000	1,009,000	637,000	208,000	139,000
1930 ... do	842,000	1,194,000	899,000	485,000	451,000	228,000
1931 ... do	784,000	241,000	41,000	449,000	[11] 308,000	0
1932 ... do	425,000	151,000	0	217,000	0	0
1933 ... do	493,000	0	0	0	0	0
1934 ... do	459,000	0	0	0	0	0
Total yearly production of copper:						
1929 ... pounds	35,377,000	35,744,000	18,237,000	20,043,000	4,459,000	3,000,000
1930 ... do	[11] 41,000,000	[11 15] 47,000,000	([16])	13,300,000	10,940,000	4,858.000
1931 ... do	[11] 57,810,000	[11 15] 14,517,000	([16])	13,100,000	7,466,000	0
1932 ... do	32,354,000	0	0	8,450,000	0	0
1933 ... do	[18] 33,197,000	0	0	0	0	0
1934 ... do	[19] 32,847,000	0	0	0	0	0
Total production of ore to end of 1930 ... tons	[11] 61,000,000	[11] 43,000,000	[11] 28,500,000	17,244,000	------	------
Total production of ore to end of 1934 ... do	[11] 63,161,000	[11] 44,000,000	[11] 28,541,000	17,910,000	------	------
Total production of copper to end of 1930 ... pounds	3,682,804,000	[21] 1,445,258,000	494,328,000	354,173,000	913,362,000	([21])
Total production of copper to end of 1934 ... do	3,839,012,000	[21] 1,459,775,000	494,328,000	375,723,000	920,828,000	([21])

	Burra Burra	Flin Flon	Sherritt Gordon	Hidden Creek	Bonanza [1]	Noranda	Mary
Location	Ducktown, Tenn.	Flin Flon, Manitoba	Cold Lake, Manitoba	Anyox, British Columbia	Anyox, British Columbia	Noranda, Quebec	Isabella, Tenn.
Normal daily production of ore ... tons	[2] 1,600	[3] 4,500	[4] 1,800	[5] 5,500	280	2,800	[6] 400
By open stoping ... do	1,350	2,180	[4] 1,800	3,600	280	----------	210
Total yearly production of ore:							
1929 ... do	[7] 473,000	0	0	[8] 1,695,000	([8])	428,000	[7] 109,000
1930 ... do	([9])	[10] 284,000	0	1,513,000	88,000	734,000	----------
1931 ... do	([9])	1,101,000	214,000	[12] 1,577,000	([12])	765,000	([9])
1932 ... do	([9])	1,447,000	[13] 158,000	[12] 1,740,000	([12])	919,000	([9])
1933 ... do	([9])	1,608,000	0	[12] 1,533,000	([12])	1,011,000	([9])
1934 ... do	([9])	1,477,000	0	[12] 1,892,000	([12])	1,051,000	([9])
Total yearly production of copper:							
1929 ... pounds	[14] 13,586,000	0	0	[8] 36,746,000	([8])	51,223,000	[13] 2,788,000
1930 ... do	([9])	[10] 2,376,000	([17])	27,714,000	2,837,000	73,509,000	----------
1931 ... do	([9])	31,385,000	14,718,000	[12] 35,236,000	([12])	62,859,000	([9])
1932 ... do	([9])	42,425,000	9,930,000	[12] 38,649,000	([12])	63,013,000	([9])
1933 ... do	([9])	41,373,000	0	[12] 34,460,000	([12])	65,009,000	([9])
1934 ... do	([9])	37,726,000	0	[12] 37,092,000	([12])	70,176,000	([9])
Total production of ore to end of 1930 ... tons	----------	284,000	----------	[20] 16,627,000	([20])	1,445,000	----------
Total production of ore to end of 1934 ... do	----------	5,918,000	372,000	23,369,000	----------	5,192,000	----------
Total production of copper to end of 1930 ... pounds	----------	2,376,000	----------	[20] 584,745,000	([20])	158,349,000	----------
Total production of copper to end of 1934 ... do	----------	155,286,000	24,648,000	730,182,000	----------	419,406,000	----------

[1] Maximum.
[2] Pillars constituting 15 percent of the ore are mined by cut-and-fill.
[3] 2,500 tons daily from underground mine.
[4] Mill capacity.
[5] 5 or 10 percent of ore mined by glory hole.
[6] Pillars amounting to 48 percent of ore mined by other methods.
[7] 1928.
[8] Hidden Creek production for 1929 includes about 100,000 tons from Bonanza, which started in that year.
[9] Production figures not published.
[10] Started production in 1930.
[11] Approximate.
[12] Hidden Creek production for 1931–34 includes ore and copper production from Bonanza.
[13] Underground work. Stopped June 1932.
[14] 1928 total for property.
[15] Includes production for Osceola mine in 1930 and 1931.
[16] Included in production for Kearsarge in 1930 and 1931.
[17] Started production in 1931 with about 15,000,000 pounds.
[18] Also 1,929,000 pounds of copper oxide.
[19] Also 1,790,000 pounds of copper oxide.
[20] Includes production of Bonanza in 1929.
[21] Includes Seneca production.

TABLE 24.—*Geological and stoping data at representative North American copper mines using open-stope methods of mining*

	Conglomerate	Kearsarge	Osceola	Mohawk	Quincy	Seneca	Burra Burra
Location	Lake Superior district, Michigan	Lake Superior district, Michigan	Lake Superior district, Michigan	Lake Superior district, Michigan	Lake Superior district, Michigan	Lake Superior district, Michigan	Ducktown, Tenn.
Kind of deposit	Bedded	Bedded	Bedded	Bedded	Bedded	Bedded	Replacement.
Length of ore body (feet)	10,000	Extensive	Extensive	Extensive	Extensive	Extensive	2,300.
Thickness of ore body (feet)	10–20	18		20	4–20	12	4–180.
Dip of ore body (degrees)	36–38	36	37	38	37–50	32	50–75.
Grade of ore,[3] percent of copper	2.2	1.4	0.9	1.5	1.1	1.1	1.6.[4]
Depth of ore mined (feet)	5,220 [7]	5,000 [8]	4,500 [8]		9,100 [8]	3,050	1,600.
Level interval (feet)	100 [8]	150 [8]	120 [8]	100 [8]	160 [8]		196.
Chute spacing under stopes (feet)		21	25		20		40.
Length of stopes (feet)	100	37	120	100	50		40–340.
Method of attack or advance	Retreating	Retreating	Retreating				Underhand.
Method of handling ore in stopes	Scrapers	Gravity and scrapers.	Gravity and scrapers.		Gravity and scrapers.		Gravity.
Where stopes are silled	On level	10 feet above drift	10 feet above drift	On level	22 feet above level		At sublevels.
Area of stope open at time (feet)	100 by 100	37 by 100			50 by 50		40 by 340.
Kind of support used	Stulls	Pillars	Pillars	Pillars	Pillars		Pillars.
Spacing of supports (feet)	7 by 7 [11]	41	Irregular	100	50		40.[11]
Kind of rounds drilled in stopes	Horizontal, 6-foot slices.	Inclined upward	Inclined upward	Inclined upward	Inclined upward		Downward, 2½- by 10-foot slices.

	Flin Flon	Sherritt Gordon	Hidden Creek	Bonanza	Noranda	Mary
Location	Flin Flon, Manitoba	Cold Lake, Manitoba	Anyox, British Columbia	Anyox, British Columbia	Noranda, Quebec	Isabella, Tenn.
Kind of deposit	Replacement	Replacement	Shoots near contact	Shoots near contact	Sulphide lenses	Replacement.
Length of ore body (feet)	2,600	5,200	300–2,000	1,700[1]		2,000.
Thickness of ore body (feet)	400[1]	15½	50–350	Width 200, thickness 60.	50[2]	150.
Dip of ore body (degrees)	70	45 to vertical		15–20		65.
Grade of ore,[3] percent of copper	2.1[5]	3.5[6]	1.2		5.1	0.8.[4]
Depth of ore mined (feet)			1,330	(9)		900.
Level interval (feet)	(10)	200, 350[8]	100–250			100, 150.
Chute spacing under stopes (feet).		30	50	Branch raises, irregular.	25	100 by 100.
Length of stopes (feet)	500	500	100–300	Large, irregular	45	60.
Method of attack or advance	Retreating	Underhand	Underhand, back, and breast.	Breast and back	Benching from incline raises.	Underhand.
Method of handling ore in stopes	Gravity	Gravity	Gravity	Scrapers and gravity	Scrapers and gravity	Scrapers and gravity.
Where stopes are silled	At sublevels	At sublevels	15–30 feet above drift	On footwall		Top stope.
Area of stope open at time (feet)	200 by 500					60 by 60.
Kind of support used	Pillars	Pillars	Pillars	Pillars	Pillars	Pillars.
Spacing of supports (feet)		120	40 to 60	Irregular	45	60.[11]
Kind of rounds drilled in stopes	Downward	Downward, 8- by 10-foot slices.	Horizontal, vertical, and downward.	Horizontal and vertical.	Upward and downward.	Downward, 10-foot holes.

1 Maximum.
2 Average.
3 Recovery in Michigan mines.
4 Iron and sulphur content utilized.
5 Also 0.08 ounce of gold, 1.28 ounces of silver, and about 3.86 percent of zinc, which is recovered.
6 Also considerable zinc, which was not recovered.
7 9,200 feet on incline.
8 On incline.
9 Only 1 level.
10 Haulage-level intervals are 520 feet and sublevels 40 feet.
11 See text.

OPEN STOPING AS APPLIED TO SMALL ORE BODIES

When the walls will stand without support except for casual stulls, small ore bodies are commonly mined by an open-stope method. Flat, irregular bodies may be stoped as developed; the ore may be broken down in breasts or benches and is usually loaded directly into cars. In steeply dipping veins or beds the broken ore drops into chutes at the bottom of the stopes, and stulls are placed to hold working platforms below the faces. In flat deposits scrapers may be used for removing the ore. The small size of such deposits usually does not permit the use of systematic lay-outs for stoping.

OPEN STOPING AS APPLIED TO FLAT-LYING ORE BODIES

Bonanza.—The method as applied to relatively large flat-lying lenses is illustrated by the practice at the Bonanza mine of the Granby Consolidated Mining, Smelting & Power Co., at Anyox, British Columbia, according to a letter from W. R. Lindsay, general superintendent, Anyox plant, May 20, 1932. The ore is treated in a concentrator about 2 miles away. Ore from the Hidden Creek mine 1 mile away is treated in the same plant. The Bonanza mine is worked through an inclined shaft. The ore body is a flat-lying (15° to 20°) lens in a shear zone in greenstone. The length is 1,700 feet and the maximum width which is along the strike is 200 feet; the thickness varies up to 60 feet. Both the ore and walls are strong.

The capacity of the mine in 1931 was 280 tons per day on one shift. The main haulage drift has been run under the ore body. Branch chute raises extend upward to the bottom of the ore. Open stoping is used.

Rounds are drilled with 140-pound drills mounted on tripods. The blasted ore is scraped to the ore chutes. Fairly strong dikes cut the ore body and are left as pillars; the pillar arrangement, therefore, is irregular.

OPEN STOPING AS APPLIED TO EXTENSIVE BEDDED DEPOSITS

Michigan open-stope practice.—Open stoping as applied systematically to extensive bedded deposits is exemplified by the Michigan practice. For many years virtually all of the ore from the Michigan copper mines was mined by the open-stope method, but in recent years greater depth or the need for sorting has forced its partial replacement by methods in which the walls are supported. During 1929 about 25 percent of the production was won by methods other than open stoping. According to Crane [29] open-stope methods generally are used in lodes with moderate and relatively flat dips. In lodes with steep dips, cut-and-fill and shrinkage methods are largely used. Ninety percent of the copper in the district has come from six ore bodies.[30] One of these is the Calumet & Hecla in the Conglomerate lode, and the other five are in the Baltic, Isle Royale, Kearsarge, Osceola, and Pewabic amygdaloid lodes, respectively.

Conglomerate mines of Calumet & Hecla Consolidated Mining Co.—The method used on the Conglomerate lode is the best example of

[29] Crane, W. R., Mining Methods and Practice in the Michigan Copper Mines: Bull. 306, Bureau of Mines, 1929, p. 187.

[30] Vivian, Harry, Deep Mining Methods, Conglomerate Mine of the Calumet & Hecla Consolidated Copper Co.; Inf. Circ. 6526, Bureau of Mines, 1931, 20 pp.

the application of the open-stope method to an extensive bedded deposit of moderate dip. As depth was attained the method of mining was changed from advancing to retreating. At the Calumet & Hecla the thickness of the lode or bed ranges from 10 or 12 feet at the outcrop to 20 feet on the 20th level, which is 8,100 feet below the surface on the dip.[31] The dip averages 37°; the ore body is remarkably persistent on both the strike and dip and contains no barren portions. The lode material and hanging wall are hard and stand well. The footwall, however, contains a layer of soft rock which may be thrust upward into the workings if weight becomes concentrated upon it. To prevent this condition and air blasts a change was made in 1909 from an advancing to a retreating system of mining. The retreating system of mining is shown in figure 50.

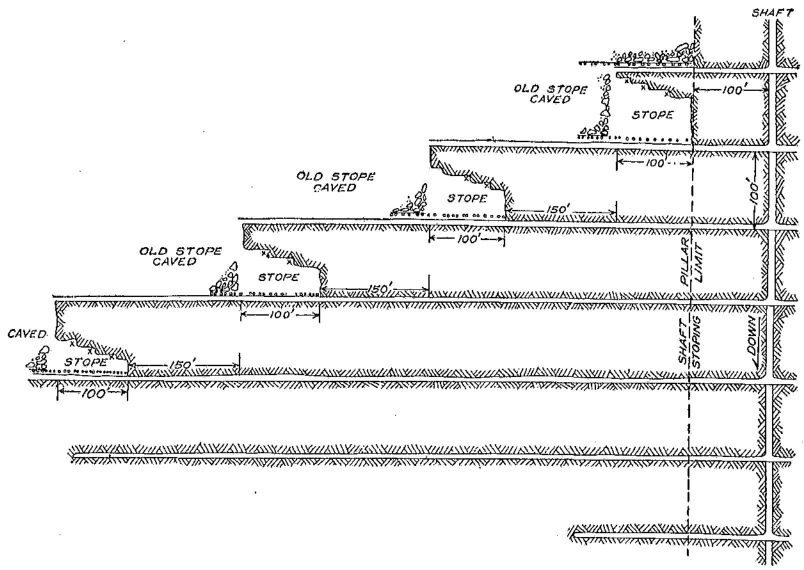


FIGURE 50.—Retreating stoping system, Conglomerate lode.

Shafts are sunk on the lode, and levels are run at 100-foot intervals. Above the 81st level 100-foot pillars were left on each side of the shafts; below the width will be 200 feet. To start a stope a 6-foot horizontal slice in the back of the drift, beginning at the end nearest the shaft, is extended toward the previously completed stope, which is usually well caved down by the time the first slice reaches it. Each successive slice thereafter is started from the side next the caved stope and extended toward the shaft (see fig. 50); thus the short raises are avoided which would be necessary if all slices were started at the shaft side. Three or four slices may be worked at a time in a stope. Eighty-five percent of the broken ore runs to the bottom of the stope as it is blasted. It is loaded by a scraper from the drift floor into cars of 3¾ tons capacity. This same scraper is used to clean the stopes. The lower 40 feet of a stope is timbered with double batteries of stulls. Four rows of 18- to 24-inch stulls in pairs, with 7 feet between pairs

[31] Vivian, Harry, Deep Mining Methods, Conglomerate Mine of the Calumet & Hecla Consolidated Copper Co.: Inf. Circ. 6526, Bureau of Mines, 1931, 20 pp.

and 7 feet between rows, are placed horizontally across the stope; the first row is placed 16 feet from the drift floor. Above this height the stulls are set singly and at irregular intervals.

Kearsarge.—The Kearsarge lode has been profitable over a continuous length of nearly 6 miles; in 1931 it was the largest producer in the district. During this year six shafts, sunk on the lode, were being operated by the Calumet & Hecla Consolidated Mining Co.[32] Operations were suspended in January 1932.

The dip in the active mining area is 36°. The level interval has been standardized at 150 feet on the incline. Pillars 200 feet wide are left on both sides of the shafts. Figure 51 shows the method of mining.

From the end of the drift at the boundary of the mining area a series of short raises is run upon the footwall on 21-foot centers to a height of approximately 10 feet. These serve to open the stopes and are used later as ore chutes. The ore is silled out by driving a drift from the boundary raise toward the shaft. This drift breaks into the tops of the successive raises, which are cut away to form funnel-shaped openings. The haulage drift is protected by the pillars between the raises.

Stope sections 37 feet wide are taken from the boundary to the level above. A 5-foot pillar known locally as a rib pillar is left between each section. The pillars are divided into sections by holes about 25 feet apart which allow passage between stope sections and permit free movement of air.

Osceola.—In 1929 the production of copper in the Osceola amygdaloid lode was exceeded in the district only by the Calumet conglomerate and the Kearsarge and Baltic amygdaloid lodes. Operations at the Osceola were suspended in May 1932. Three and one-half miles of the Osceola lode belongs to the Calumet & Hecla Consolidated Copper Co.; ore is being produced from four shafts.

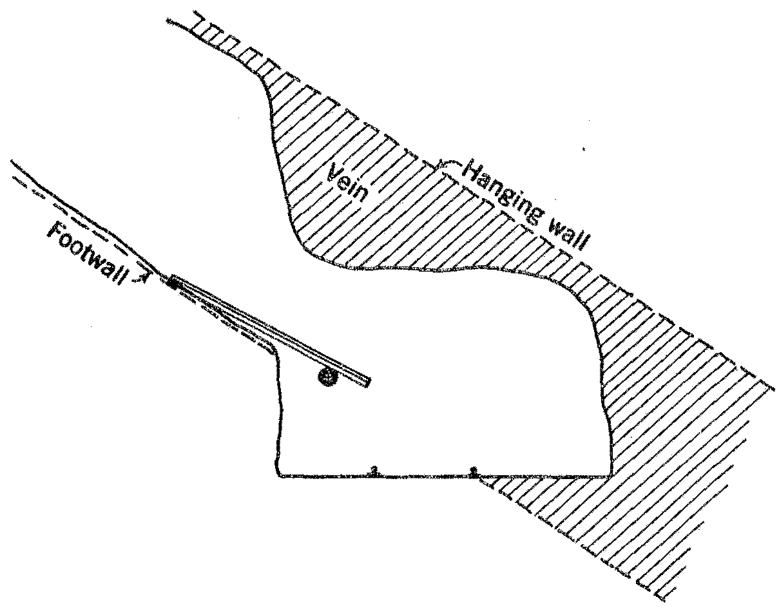
According to Potter [33] mineralization in the Osceola lode is very erratic. The most persistent copper-bearing horizon is along the hanging wall, but in many places rich pockets of heavy copper occur where irregular benches of fragmental amygdaloid extend into the underlying footwall traprock.

The shafts are located in the lode, preferably a few feet from the hanging wall or just under the lode, which has an average dip of 37°. Pillars 110 feet wide are left on either side of the shafts, which are 9 by 22 feet in section. With greater depth the pillar width probably will be increased. Levels are at 120-foot intervals along the dip.

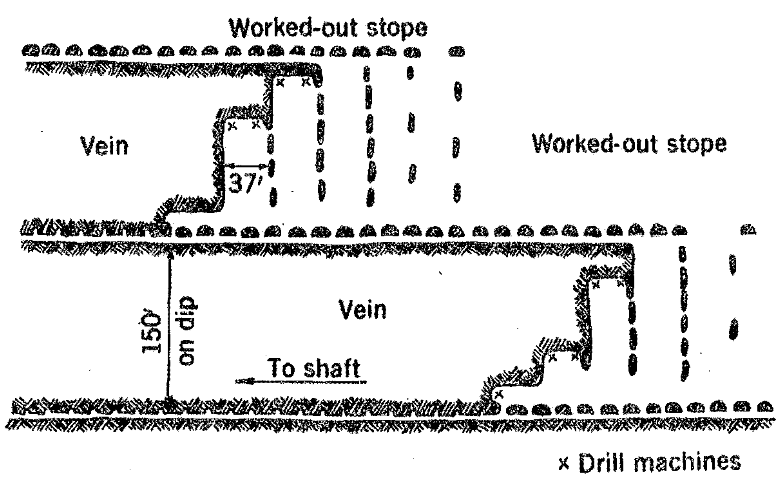
On account of the irregular occurrence of the copper any prearranged method of attack is impracticable. Open stopes with pillars and without timbers are the general rule. The location and size of pillars are determined according to the judgment and experience of the operating staff; these pillars consist usually of barren or low-grade material. Figure 52 shows a section and plan of a typical stope. The ore is loaded into the tramming cars by gravity or scrapers.

[32] Potter, Ocha, Mining Methods in the Amygdaloid Lode Operations: Min. Cong. Jour., vol. 17, October 1931, pp. 482–486.

[33] Potter, Ocha, and Richards, Samuel, Mining Methods in the Amygdaloid Lodes—Osceola Lode Operations: Min. Cong. Jour., vol. 17, October 1931, pp. 487–490.



SECTION ACROSS DRIFT THROUGH CHUTE AND BOTTOM STOPE



IDEAL PLAN OF RIB PILLAR STOPING

FIGURE 51.—Open-stope method used on Kearsarge lode.

Mohawk.—The mine of the Mohawk Mining Co. is on the Kearsarge amygdaloid lode. It is operated through three shafts. The Mohawk mill has a capacity of 2,800 to 3,000 tons daily. The method of stoping is similar to that at the Kearsarge mine of the Calumet & Hecla Consolidated Copper Co. Stopes are 100 feet long and are separated by 15-foot pillars on the dip.[34] A 15-foot pillar is left at the top of the

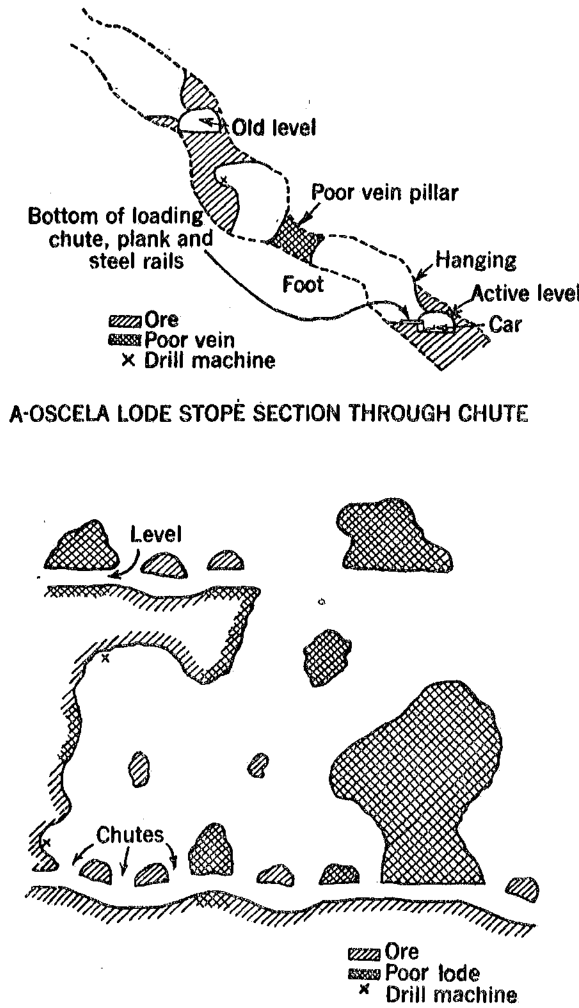


FIGURE 52.—Method of stoping on Osceola lode.

stope until the level above is finished. A pillar 13 feet in diameter is also left just above the drift in the middle of each stope. Stoping operations in the original property in 1931 comprised only mining pillars.

Quincy.—The mine of the Quincy Mining Co. is on the Pewabic amygdaloid lode. The thickness of the lode ranges from 4 feet in the

[34] Peele, Robert, Mining Engineers' Handbook, p. 578.

lower levels to 20 feet in the upper workings. The shafts are sunk on the footwall of the lode, which dips 50° at the surface but flattens out to 37° at a depth of 4,000 feet. The deepest shaft is 9,100 feet on the dip.

Serious damage has occurred in the mine from air blasts. In 1932 stopes were filled after completion to prevent further crushing of shaft pillars, which are 250 feet wide. Stopes are silled about 22 feet above the level and are started 50 feet wide. Two pillars with ore chutes between are left above the drift.[35] A 50-foot pillar is left between each stope. About 20 feet of the pillar is cut away near the middle; the lower segment is 62 feet and the upper segment 57 feet long. Both segments are cut to a point on the ends facing each other.

SUBLEVEL STOPING

Probably the greatest improvement in open stoping at copper mines in recent years has been the adaptation to copper mines of the sublevel principle of mining, which was developed in the iron mines. Where the physical conditions of the ore and wall rock are favorable, this method has largely replaced underground glory holes and shrinkage stoping.

The method retains the favorable feature of glory-hole work in that long, downward vertical rounds can be drilled and a large tonnage broken per machine shift; moreover it overcomes the hazard to the men of working under large unsupported backs and on long slopes, and less hand shoveling is required to clean the benches before drilling.

Sublevel stoping has advantages over shrinkage stoping in that the ore can be drawn as soon as it is broken and operations can be concentrated in fewer stopes to obtain a required tonnage. Sublevel-stoping costs usually are less than shrinkage stoping costs in ground of the same kind.

The effect is virtually the same if spiral raises or a series of inclined raises are used instead of sublevels to provide points for attacking the ore. Although these modifications cannot strictly be classed as sublevel stoping, they are grouped under this heading for the purposes of this paper.

One of the best examples, and the first, of straight sublevel stoping in copper mining is the practice at the Burra Burra mine of the Tennessee Copper Co. at Ducktown, Tenn. The methods used at the Sheritt Gordon mine at Cold Lake, Manitoba, and the Flin Flon mine at Flin Flon, Manitoba, were patterned after practice at Burra Burra.

At the Hidden Creek mine, Anyox, British Columbia, the ore is broken from spiral raises. At the Noranda mines, Noranda, Quebec, the ore is attacked from a system of parallel inclined raises.

Burra Burra.—The Burra Burra is the principal mine of the Tennessee Copper Co. A smaller mine, the Eureka, adjoins the Burra Burra and is operated mainly for the sulphur in the ore. About 60 percent of the output from both mines goes directly to the smelter, and 40 percent is concentrated before smelting. The roaster gases at the smelter are used for the manufacture of sulphuric acid. The ore is a massive pyrite containing copper sulphides. The ore body has a maximum length of 2,300 feet, ranges from a few feet to 180 feet in thickness, and dips 50° to 75°. The immediate wall rocks are

[35] See footnote 34.

very hard and stand well for spans of more than 100 feet where the ground is not fractured as a result of folding. The ore is hard and tends to break in large, angular blocks.

The Burra Burra mine is worked from two shafts 2,200 feet apart, measured along the strike of the lode.[36] Thirteen levels have been

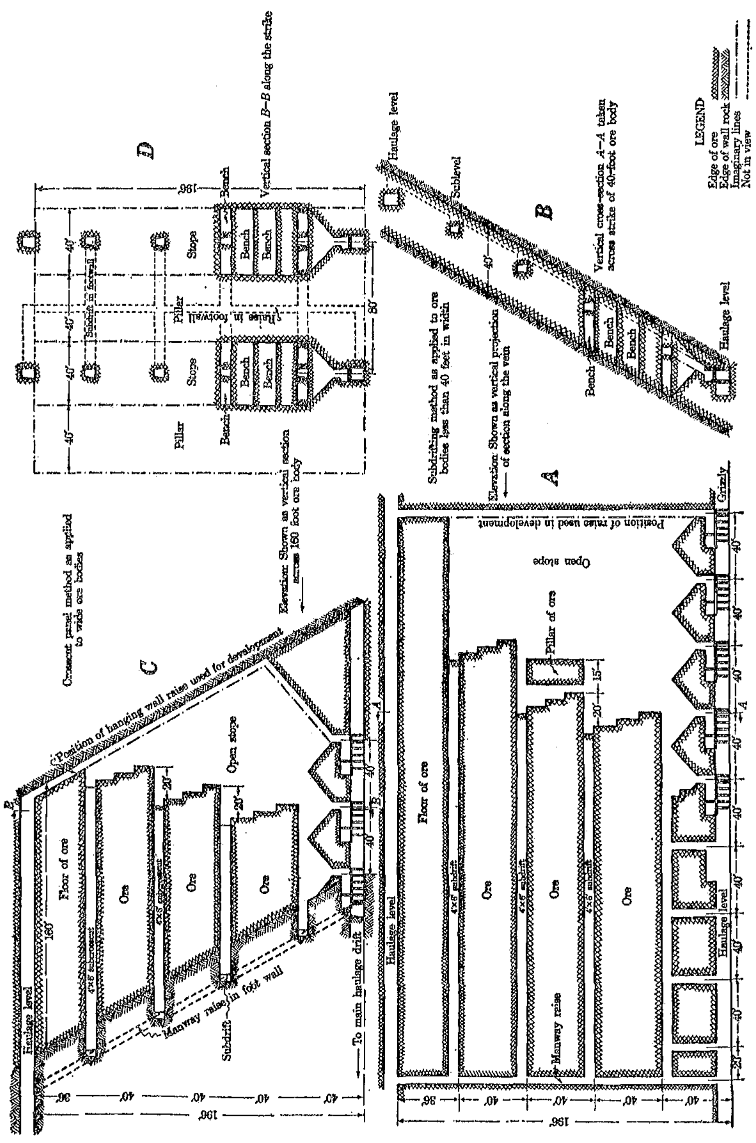


FIGURE 53.—Methods of developing and mining sublevel stopes, Burra Burra mine, Tennessee Copper Co., Ducktown, Tenn.

developed. Haulage drifts below no. 10 level are 196 feet apart vertically.

Where the ore body is less than 40 feet wide sublevel stopes the full width of the ore are run longitudinally. (See fig. 53.) Pillars between

[36] McNaughton, C. H., Mining Methods of the Tennessee Copper Co., Ducktown, Tenn.; Inf. Circ. 6149, Bureau of Mines, 1929, 17 pp.

stopes are usually left in narrow parts of the vein. The block to be mined is developed by raises driven on the footwall to the level above, usually at the ends of the block. Sublevels are then driven at 40-foot vertical intervals to connect with the end raises. At 40-foot intervals 4- by 5-foot "pull raises" are driven on the footwall to the first sublevel. Grizzly chambers are constructed immediately over the drift at the "pull raises." The draw holes are funneled out at the top before stoping begins.

The total development in a representative block was 2,210 linear feet; as 206,000 tons of ore were mined from this block, 96 tons were stoped per foot of stope development work. If the ore that is recovered in the pillars through the same workings is included, 110 tons are produced per foot of development work.

Stoping is begun on the second sublevel at one end of the block by slabbing off the sides of the subdrift to expose the walls; a footwall raise is then benched out the full width of the ore. Benching proper begins by cutting a 5-foot slab from wall to wall at the height of the sublevel and extending the cut downward 6 feet. A second 5-foot cut is then taken as before at the subdrift level, after which the first bench is extended downward to the top of the sublevel below. The ore is broken by successive downward slices retreating from one end of the stope to the other.

Until 1930 the practice had been to maintain two benches on the ends of the blocks of ore between sublevels, as shown in figure 53. Elimination of one of the benches has reduced the hazard from falling ground and minimized the amount of hand shoveling on benches after blasting. Bench holes are drilled 4 to 5 feet apart along the face with a burden of 2½ feet. After the initial 6-foot cut, rounds are drilled 10 feet deep.

Beginning on the next sublevel above, the process is repeated. The work on the lower level is kept far enough ahead of that above that the men are always working under the protection of solid ground.

Where the ore body is wide haulage drifts are run in the footwall; crosscuts extend to the hanging wall on 40-foot centers. The sublevel stopes are 40 feet wide across the ore body with 40-foot pillars between stopes. These stopes are developed from crosscuts on the haulage level in the same manner that the longitudinal stopes are opened from the drifts. (See fig. 53.) The figure shows two benches on the end of each block; the practice has been changed as in the longitudinal stopes, and only one bench of a block is worked at a time.

The ore runs by gravity to grizzly chambers; the grizzly openings are 20 by 20 inches. Large fragments are partly broken by the fall to the bottom of the stopes. Moreover, the practice of blasting narrow cuts (2½ feet) tends to keep the number of large boulders to a minimum.

In addition to the 40-foot vertical pillars between stopes, 40-foot level pillars and 30-foot crown pillars are left in the open stopes. In narrow veins, where the ground stands well, the ore left in pillars at the bottom of a stope is drilled, broken, and loaded into tramcars by portable scrapers. The stope is then filled and the crown pillar mined by an inclined cut-and-fill method. The filling for each cut comes from the filled stope above. In wide stopes the part of the horizontal pillar above the level is first taken out by an inclined cut-and-fill method; the filling comes from caved waste in the stope. The crown pillar below the haulage level is mined later, and the ore is broken by long slabbing rounds drilled from stations on the footwall. The ore is drawn out at the bottom of the stope. Slabbing is continued until the pillar fails, and waste from above follows the ore into the stope.

First the vertical pillars are partly undercut 50 feet above the level, and the ore is blasted into the draw raises of the adjoining stope. Holes up to 120 feet deep are then drilled from stations cut into the footwall raise at the middle of the pillar. Beginning at the bottom, the holes are blasted with 0.1 pound of powder to the ton of ore. The ground breaks well, but some large blocks cave beyond the holes. About 85 percent of the total ore is mined by open stoping and 15 percent by the cut-and-fill method.

About one-fourth of the ore in a bottom pillar between the haulage level and the first sublevel is removed during stope development. If the ore removed in development is credited to the sublevel method, about 70 percent of the ore in the stope section is mined by sublevel stoping and 30 percent in extracting pillars.

In wide sections of the ore body, where the 40-foot stopes alternate with 40-foot pillars, the total proportion of ore in pillars would be $0.30 \times 0.50 + 0.50$, or 65 percent.

If it is assumed that in narrow sections of the ore body (see fig. 53) 40-foot pillars are left between 340-foot stopes and the ore is broken out on the strike between walls, the proportion of ore left in pillars for later extraction would be $0.30 \times 0.89 + 0.11$, or 38 percent.

With an extraction of 60 percent of the pillar ore, the over-all extraction in the wide part of the ore body would be $0.60 \times 0.65 + 1 \times 0.35$, or 74 percent. In the narrow part of the ore body the over-all saving would be $0.60 \times 0.38 + 1 \times 0.62$, or 85 percent. Actual extraction is about 5 percent higher than estimated. The average extraction at the mine is about 85 percent.

Flin Flon.—The main ore body at the Flin Flon mine is exposed at the surface; it has a maximum width of 200 feet below the pit. About two-thirds of the ore was produced from an open pit and one-third from underground operations to the end of 1935. During the first half of 1936 the normal daily tonnage was 2,445 tons from the underground workings and 2,083 tons from the open pit. The underground mine is open to a depth of 1,170 feet; to 1934 stoping was above the 650-foot level. A sublevel method of stoping is used, according to a letter from R. E. Phelan, general manager, Hudson Bay Mining & Smelting Co., Winnipeg, Manitoba, August 28, 1936. Haulage levels are 520 feet apart and sublevels 40 feet at 40-foot intervals. Stopes are 500 feet long and as wide as the ore body. A scraping level is run 16 feet above the haulage adit. Chutes along the scaper drift are 30 feet apart; stopes are started above this level. Starting at a raise in the center of the stope each sublevel is successively widened to the walls with drills mounted on columns. The ore between sublevels is then drilled with jackhammers, retreating both ways from the central raise. The ore falls into the chutes and is scraped into cars.

Sherritt-Gordon.—The mine of the Sherritt-Gordon Co., Ltd., is at Cold Lake, Manitoba. The ore bodies comprise lenses of massive and disseminated sulphides in gneiss. The mine is worked through three shafts.[37] The ore is treated at a concentrator at the mine. Concentrates are shipped to the Flin Flon smelter, about 30 miles

[37] Canadian Mining and Metallurgical Bulletin, vol. 33, August 1930, p. 245.

away. Production was started in 1931 at about 500 tons daily; operations were suspended in June 1932.

The Burra Burra sublevel method of mining was used at the beginning but was soon modified so that it bore little resemblance to the original method. Later the system was again modified where the dip of the vein was less than 45°.[38]

A block 1,000 feet long between the 500-foot level and the surface was the first to be developed. The thickness of the ore mined ranged from 3 to 50 feet. The average thickness in 1931 was 12.5 feet and in 1932, 12.1 feet. Stoping was begun in the middle of the block. A drift 8 feet high was run on the 500-foot or third level along the footwall contact; in the center of the ore body the width of the drift was 15 feet and at the ends 7 feet. Drifts 4 by 6½ feet in section were also run in the footwall on the first and second levels; a subdrift was run under a surface pillar. Box holes or chute raises were put up from the haulage drift at 30-foot intervals. Raises 5 by 5 feet in section were run from the box holes on the footwall at 120-foot intervals to the surface or to the sublevel below the surface pillar. The raises driven up from the third level were connected by short crosscuts with the drifts on the first and second levels. The box holes at the haulage level were funneled out until they connected with each other, thus silling out the stope.

The next step was to start a bench on one of the raises at the second level by cutting around the top of the raise until both walls were exposed; the bench was 8 feet wide on each side of the raise. The bench was then mined downward to the level below. Holes were drilled with pluggers on the corners of 2-foot squares; after the initial cut, rounds were 10 feet deep. The ore from the benches and raises dropped to the box holes, whence it was drawn into 80-cubic-foot Granby-type cars. Successive slices were carried down in like manner 50 to 55 feet from the center lines of the raise. A rib pillar 10 to 20 feet long was left between each stope section. Manways were maintained in the original raises after stoping operations were begun. After the faces of the stope on the second and third levels had retreated 30 to 40 feet back from the raise a stope was started at the top of the raise on the first level and carried back in the same manner as on the sublevel below. The ore fell through the open stope to the box holes. No grizzlies were used. Men working on the benches wore safety belts. In some instances the pillars were stoped out and the hanging wall was left unsupported; in others the pillars were left and the ore was lost.

When the dip was less than 45° the method was modified; the ore was pulled down the slope as much as 200 feet by scrapers. Later, it was found more economical to wash the ore down the slopes with water.

The ore is crushed at the shaft before it is hoisted.

Hidden Creek.—The Hidden Creek mine of the Granby Consolidated Mining, Smelting & Power Co. is at Anyox, British Columbia, on Granby Bay of Observatory Inlet.[39] The ore bodies are of two types: (1) Irregular lenses of heavy pyrite with some copper and (2) chalcopyrite and pyrrhotite occurring in joint planes and impregnating

[38] Brown, E. L., general superintendent, Sherritt-Gordon Mines, Ltd., Mining Methods and Costs at the Sherritt Gordon Mine: Canadian Inst. Min. and Met. Eng., vol. 36, 1933, pp. 468-494.
[39] Data on Hidden Creek mine supplied by W. R. Lindsay, general superintendent, Anyox plant.

the rock. The wall rocks are hard and stand well over relatively long spans. The mine is operated through an adit and a 550-foot shaft. The ore is treated in a concentrator at the mine. In 1931 virtually all the ore produced came from open stopes, although in the past shrinkage stopes 200 feet wide, 300 feet long, and 200 feet high had been worked.

The method used is similar in principle to sublevel stoping at the Burra Burra mine of the Tennessee Copper Co. At Hidden Creek, however, the ore is broken from spiral raises rather than from sublevels. This method is adaptable to fairly large ore bodies where the ground is strong and does not slough. The ore body to be mined by this method is opened by one or more drifts; the number depends on the width. Pillars 40 to 60 feet thick are set off between each block of stoping ground. A manway raise is run in the center of the pillars to the level above. Chute raises are put up on 50-foot centers. A floor pillar 15 to 30 feet high is left at the bottom of the stope. Grizzly chambers are cut on three sides of the chute raises just above the chutes. The grizzly spacing is 18 inches. Manway raises are run midway between pairs of chutes, and connections are made to the grizzly chambers by short drifts, or a grizzly drift may be used to afford access to the chambers.

After the chute raises have been cross-connected, large raises 12 feet or more wide by 9 to 10 feet high are driven from them on a slope just sufficient to allow the broken ore to run; the raises are swung around on a spiral, similar to a winding staircase on an exaggerated scale; the central well down which the broken ore falls is an unobstructed opening. The main width of the spiral, along the outer edge of which is a trail, is completely under cover. The grade of the trail is such that a man can walk up it. The manways in the pillars are connected to the spiral as required. If width and other conditions permit, a second interlacing spiral may be driven across the stope. The ore is broken from these preliminary cuts by back stoping, breast stoping, and benching. Supporting braces or arches of ground are left at critical points in the stope and recovered at the finish by blasting rounds previously drilled. The stopes are drawn empty each day. Large slabs of ore falling down the stope are largely broken up. Secondary blasting, however, is nearly as expensive as primary blasting. Much ore is brought down by caving many of the ribs. This ore is broken by secondary blasting both in the stopes and in the grizzly chambers.

A variation of the spiral-stope method called the long-slope method has been adopted for mining lower ends of ore bodies where the ore limits are not definitely known.[40] This method requires the usual preliminary development and consists essentially of driving parallel inclined stopes (+38°) through the ore body parallel to the strike of the ore in narrow ore and across the strike in wide ore, leaving a parallel inclined brace over the back of each stope of such thickness that it may be mined as one bench with 20-foot holes.

Starting at the top of the desired ore section, the initial stope (10 feet high and up to 80 feet wide) is mined to the established limits, and the back is slabbed, arched, and trimmed to give maximum strength and safety. The bottom is then benched with drifter benches, until further benching seems dangerous. As benching is the cheapest

[40] McNicholas, F. S. Long-slope Mining at Anyox: Eng. and Min. Jour. vol. 133, November 1932, pp. 567–568.

method of breaking ground at Anyox, an effort is made to obtain a maximum of such work. In some ground, benching may be carried safely 70 or 80 feet from the back.

After benching is completed, a 30-foot bottom is left under the stope, and another parallel stope is driven and benched in like manner.

The process is repeated until the entire block of ground is prepared, and a series of parallel inclined braces is left. These braces are then mined, starting at the bottom of the prepared section, by drilling 20-foot holes downward from tripods at the top of the lowest inclined brace and blasting electrically in sections, or if the brace is weak the entire brace is drilled and blasted in one simultaneous electrical blast.

The method is flexible, but as the height and width of each stope depend upon the character of the ground the exact interval between stopes cannot be anticipated. At all times an effort is made to carry benching as far as possible with safety. In other words, the amount of benching that may be done in the upper stope must be determined before the next lower one is started. A vertical leeway of about 10 feet in the location of the succeeding understope results, as one cut (10 feet) may or may not be taken off the back as conditions require. A total of 90 tons is broken per machine-shift with the long-slope method compared to 76 tons with the spiral stope, and there is a corresponding reduction in cost.

Pillars and braces left in completed stopes are broken in large single blasts. Waste rock from the walls of the stopes falls on top of the ore and follows the ore down as it is drawn. A large blast comprising 58 tons of explosive broke 500,000 tons of ore and 300,000 tons of waste rock at a cost of 6.13 cents per ton of ore in 1934.[41] The area was roughly 550 feet long, 500 feet high, and 100 to 250 feet wide. The explosive was loaded in powder drifts in thick pillars and in long drill holes in other accessible parts of the area to be blasted.

Noranda.—The Horne mine of the Noranda Mines, Ltd., is in the Rouyn district of Quebec. The ore consists of lenses of mixed sulphides. A typical smaller lens would be about 50 feet wide and a few hundred feet long. The important "H" ore body, which is over 300 feet wide in places, consists of massive sulphides; it contains good values in gold. The ore bodies first discovered were at the surface. The mine is worked through a number of vertical shafts. Production of ore started at the end of 1927.

The mining method used is an adaptation of the open-stope system developed under the direction of Ernest Hibbert [42] at the Mother Lode mine, now part of the holdings of the Granby Consolidated Mining, Smelting & Power Co., Ltd., at Greenwood, British Columbia. The system is locally called the Hibbert method; it is similar in principle to sublevel stoping except that inclined raises are used instead of subdrifts.

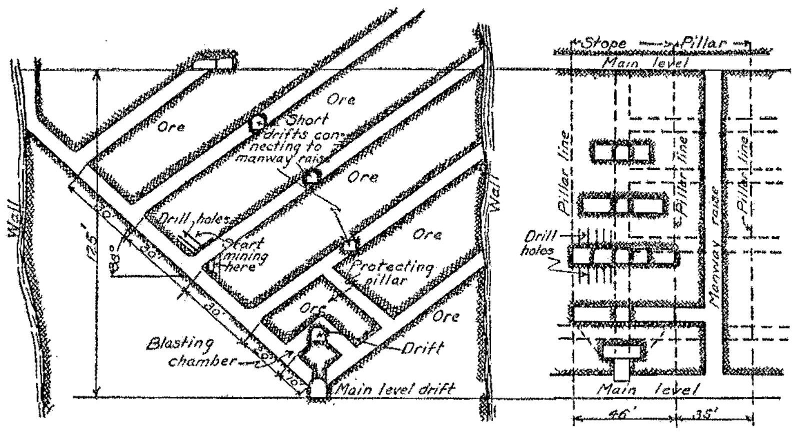
In ore bodies less than 50 feet wide that are of relatively small importance a series of raises are driven from level to level in the ore along the strike at an inclination of 42°. From these raises 6- by 7-foot branch raises are driven at right angles about 35 feet apart.

[41] Healy, R. L., and McNicholas, F. S., Breaking Half a Million Tons of Ore in One Blast With 58 Tons of Powder: Min. and Met., May 1935.

[42] Hubbell, A. H., The Noranda Enterprise, Part 1, The Horne Mine: Eng. and Min. Jour., vol. 126, Sept. 1, 1928, pp. 331–334.

After the branch raises have been holed through to the level above they are slabbed out the full width of the ore body. The benches of ore between the slabbed-out inclines are broken by both overhand and underhand mining. The method of stoping is similar to that used at the Hanover-Bessemer mine.[43]

Where the ore body is more than 50 feet wide, it is laid out for stoping in 46-foot stopes separated by 35-foot pillars. Chutes are built 27 feet apart in a drift along the strike; from these chutes inclined raises are driven at right angles to the strike. (See fig. 54.)[44]



A. LONGITUDINAL PROJECTION B. TRANSVERSE SECTION

FIGURE 54.—Open-stope method in wide ore bodies, Noranda.

Where the dip of the deposit deviates appreciably from the vertical the first incline is run up the footwall. Branch raises are run as shown either to the hanging wall or to the level above. Stopes are developed by connecting the inclines from two adjoining chutes and slabbing them out to a width of 46 feet. At the third chute the incline with its branches is in the center of a pillar section and is left 7 feet wide. The fourth and fifth series of inclines are then connected for the next stope. The benches are mined by drilling up and down holes and blasting the ore into the incline, which soon becomes a large open stope. The ore runs down to a grizzly station above the haulage level. After the stopes have been mined the pillars are lagged over and the stopes filled. Pillars will then be extracted by cut-and-fill stoping. In the lower "H" ore body the sublevel system is used and is reported to have certain advantages over the inclined method.

UNDERGROUND GLORY HOLE

Mary.—The underground glory-hole method is illustrated by the practice at the Mary mine of the Ducktown Chemical & Iron Co., Isabella, Tenn. The ore occurs in lenses in a hard country rock that stands well for spans of 100 feet if the back is arched.[45] The main ore

[43] Kniffin, Lloyd M., Mining and Engineering Methods and Costs of the Hanover-Bessemer Iron & Copper Co., Fierro, N. Mex.: Inf. Circ. 6361, Bureau of Mines, 1930, 20 pp.

[44] After Jackson, Chas. F., Mining Ore in Open Stopes, Central and Eastern United States: Inf. Circ. 6193, Bureau of Mines, 1931, p. 7, fig. 10.

[45] Kegler, V. L., Mining Methods of the Ducktown Chemical &Iron Co., Mary Mine, Isabella, Tenn.: Inf. Circ. 6397, Bureau of Mines, 1931, 9 pp.

body is 2,000 feet long, averages 150 feet in thickness, and dips 65°; it is mined for its copper, sulphur, and iron. The roaster gases at the smelter are used in the manufacture of sulphuric acid.

The ore is hoisted through a vertical shaft; two other shafts are used for ventilation and supplies. The upper level interval is 100 feet and the lower 150 feet. Haulage drifts are driven through the wall rock to the ore body, where drifting follows the center of the vein. Crosscuts are driven from hanging to foot wall on 100-foot centers. All

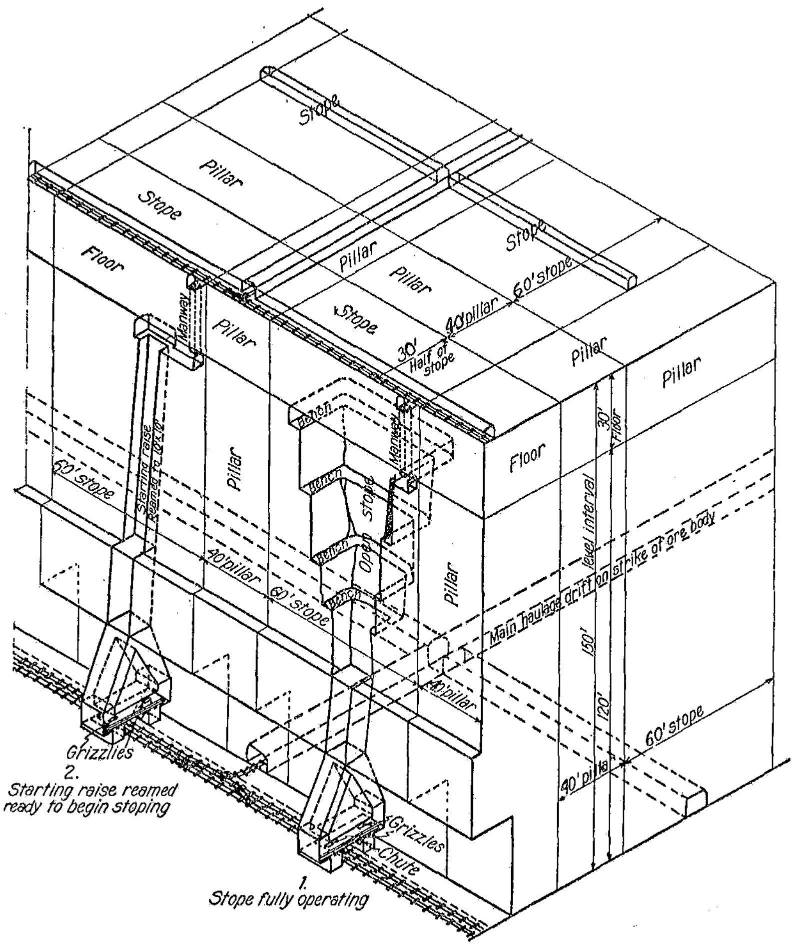


FIGURE 55.—Open-stope method, Mary mine.

stope development work is in ore. After exploration has defined the strike of an ore body it is divided into 60-foot chambers and 40-foot pillars as measured along the strike. (See fig. 55.) Raises are driven in the chambers from points near the footwall to the level above. An offset is made in the raise 30 feet below the upper level to facilitate stoping and permit the upper 30 feet to be used as a manway.

In preparation for mining, the crosscuts are slabbed to a width of 14 feet and the raises enlarged to a diameter of 10 feet from the offset to the level below. Stoping is begun by reaming out a bench at the top of the enlarged raise. Benches are then carried down through the stope by underhand mining. The main disadvantages of this method of mining are the hazard from working under a wide back and the necessity of cleaning off the benches by hand preparatory to drilling. Figure 55 shows grizzlies and chutes at the bottom of the raise. In old stopes the ore is loaded by hand into cars from the floor of the crosscuts. After a series of open stopes is completed the pillars are robbed. However, enough pillars are left to prevent surface subsidence.

SQUARE-SETTING

Square-setting is used as a principal stoping method only in heavy ground; filling usually follows closely the removal of the ore. With this method much timber is required and usually less ore than with other methods is broken per man-shift. It can be used profitably only in relatively rich ore. Square-setting, however, is widely used as an auxiliary method to other stoping methods, particularly in the extraction of pillars. Any stoping method in which the excavation is timbered with square-sets is termed "square-setting."[46] Four variations in the method are recognized: Horizontal sections, inclined (rill) sections, upward vertical sections, and downward vertical sections. The direction of attack depends mainly on the size and shape of the ore body, the intensity and direction of the ground pressure, the amount of sorting necessary, and the daily demand for tonnage.

The timber in square-set stopes must withstand the ground pressure until the stopes are filled or a section is finished. In wide deposits, where the pressure is from the sides, horizontal sections one to six sets wide are generally taken across the ore, beginning at the footwall side. When the greatest pressure is from the top, the sections are taken up vertically. Loose or running ore usually is mined in vertical sections, beginning at the top. The character of the ore and walls governs the amount of ground that can be kept open at a time. In narrow veins, sections of 100 feet or more may be mined from one level to the next; a horizontal or rilled back may be carried. Two or more floors as long as the stopes may be open at a time. On the other hand, in wide deposits, where the ground is very heavy, sections may be only one or two sets wide, and the sets may be filled as soon as placed.

Small sections can be completed in less time than large ones. Decreasing the size of sections frequently has eliminated trouble from wall pressure. Sometimes a small section may be worked through between levels without filling, whereas a large one would have to be filled as stoping progressed. The rate of working out a stope should be such that the timbers are relieved as pressure develops; that is, when the floor begins to take weight the floor above should be started to relieve the one below. Where the demand for tonnage requires rapid mining of the ore, larger sections are frequently more practical than they would be otherwise.

[46] Gardner, E. D., and Vanderburg, W. O., Square-set System of Mining: Inf. Circ. 6691, Bureau of Mines, 1933, 68 pp.

Horizontal sections usually are preferred to rill sections where much sorting of ore is required; moreover, the need for close sorting may make horizontal sections preferable to vertical slices. The amount of sorting required governs to a considerable extent the stoping details and is a factor in choice of the method. Filling usually is brought into stopes from raises from the level above; occasionally, however, it is obtained from crosscuts and inclined raises run into the walls or is sorted from the ore.

Tables 25 and 26 show production by square-setting and give details of its application at principal North American copper mines. The following pages describe square-set stoping at representative mines.

ANACONDA MINES

Square-setting has been the principal method of stoping in mines at Butte, Mont.; moreover, the method has had its chief application in these mines.

The mines of the Anaconda Copper Co. at Butte have been among the largest producers of copper in North America. They are served by many shafts, but each mine is worked as a unit. During 1929, 17 individual mines were being worked, and in 1930, 22 mines were listed as having produced ore.

The ore occurs in a number of vein systems in granite. The thickness of the individual ore shoots ranges from 4 to 100 feet. The granite wall rock is fractured and partly decomposed and usually requires support as soon as the ore in a stope is removed. Moreover, most of the ore is structurally weak and will stand unsupported only over relatively short spans. As upper levels and the surface must be maintained, top slicing or caving methods are impracticable. About 80 percent of the ore is mined by square-setting and 20 percent by cut-and-fill methods. The latter are used in relatively narrow ore bodies where the wall rock stands comparatively well.

The manner of attacking a given ore body depends mainly on the width of the ore. In narrow veins the ore usually is mined by taking horizontal sections along the strike of the vein; in some places, however, inclined or rill sections have been used. Figure 56 shows a typical example of mining a narrow vein by horizontal sections at Butte.[47] The method of framing the sets at Butte is shown in figure 57*B*. Framing commonly used in the Southwest is shown in figure 57*A*.

Filling in excess of that sorted from the ore is drawn from a waste raise through temporary chutes placed at each floor as stoping progresses upward. The waste is allowed to run from the raise into the first few sets and leveled off by hand. For filling more distant sets the waste is drawn into cars and transferred to the place needed. Subsequent blocks are mined without leaving a pillar between; the next stope usually is well under way before the first one is finished. Ground conditions determine how closely one block can follow another.

The general method of mining is the same in wide veins as in narrow ones; the main difference is in the length of the stope section. In moderately wide veins raises are put up on the footwall. If the vein is wide enough to require drawpoints elsewhere than at the footwall drift, crosscuts are run toward the hanging wall at the chute sets, or a hanging-wall drift may be run. Raises are spaced according to

[47] After Bicknell, H. L., The Butte System of Square-Set Mining: Compressed Air Mag., April 1922, p.113.

TABLE 25.—*Production and geological data at 10 representative North American mines using square-set method of mining*

Mine	Anaconda	United Verde Extension	United Verde (underground)	Magma	Copper Queen (Limestone Division).	Denn	Colorada	Frood	Matahambre	Old Dominion
Location	Butte, Mont.	Jerome, Ariz.	Jerome, Ariz.	Superior, Ariz.	Bisbee, Ariz.	Bisbee, Ariz.	Cananea, Sonora	Sudbury, Ontario	Matahambre, Cuba	Globe, Ariz.
Normal daily production of ore (tons):										
Total	10,000	1,200	3,000	830	1,000	400	1,500	[1] 3,000	1,300	1,800
By square-setting	8,000	1,200	700	500	550	350	180	[2] 800	400	300
Total yearly production of ore (tons):										
1929	—	361,000	[3] 1,788,000	270,000	460,000	106,000	[4] 895,000	200,000	362,000	[2] 220,000
1930	—	302,000	[3] 986,000	252,000	280,000	76,000	([5])	903,000	369,000	[2] 243,000
1931	—	183,000	[3] 242,000	232,000	([5])	([5])	([5])	1,069,000	[2] 330,000	—
1932	—	242,000	0	145,000	([5])	([5])	([5])	514,000	[2] 145,000	0
1933	—	186,000	0	145,000	([5])	([5])	([5])	953,000	[2] 219,000	0
1934	—	201,000	0	265,000	([5])	([5])	([5])	1,868,000	[2] 152,000	0
Total yearly production of copper (pounds):										
1929	281,452,000	59,068,000	[3] 144,700,000	38,235,000	42,559,000	10,700,000	[4] 58,827,000	[4] 88,880,000	31,500,000	18,943,000
1930	185,325,000	40,817,000	[3] 72,500,000	31,884,000	29,106,000	9,000,000	[4] 42,425,000	[4] 125,328,000	34,109,000	17,597,000
1931	171,244,000	24,591,000	[3] 19,900,000	28,399,000	([5])	([5])	[4] 41,873,000	[4] 110,740,000	29,778,000	11,348,000
1932	97,208,000	37,752,000	0	21,706,000	([5])	([5])	[4] 36,820,000	[4] 74,620,000	13,066,000	0
1933	92,721,000	33,197,000	0	19,713,000	([5])	([5])	[4] 51,793,000	[4] 141,500,000	19,746,000	0
1934	64,567,000	26,136,000	0	32,023,000	([5])	([5])	[4] 60,431,000	[4] 200,410,000	13,650,000	0
Total production of ore (tons):										
To end of 1930	—	2,700,000	—	2,380,000	—	[6] 228,000	[7] 21,000,000	[4] 26,700,000	—	—
To end of 1934	—	3,512,000	—	3,167,000	—	—	—	[4] 32,974,000	—	—
Total production of copper (pounds):										
To end of 1930	[8] 10,102,000,000	608,000,000	[3] 1,960,000,000	285,182,000	[9] 2,530,000,000	[6] 25,200,000	[4] 1,106,318,000	[4] 970,000,000	—	[10] 600,000,000
To end of 1934	[8] 10,528,000,000	729,676,000	[3] 1,980,000,000	387,023,000	[11] 2,775,000,000	[12] 162,000,000	[4] 1,297,235,000	[4] 1,497,270,000	—	[13] 611,000,000
Kind of deposit	Veins in granite	Sulphide lenses	Massive sulphides	Vein in diabase	Replacement deposit in limestone	Replacement deposits in limestone	Pipe in quartz porphyry	Deposit in shear zone	Lenses	Vein
Size of ore body (feet):										
Length	—	500	1,100	1,200–1,500	—	Up to 500	300	5,000	Up to 275	350
Thickness	4–100	300	300	[14] 25	—	[14] 125	[15] 20–60	40–200	Up to 30	100
Average grade of ore (percent copper)	—	6.44	5.0	6.06	4.9	5.5	4.0	[16] 4.5	4.5	—
Dip of ore body	Steep dips predominate	—	60	45–80	—	90	90	65	42–45	45–90
Character of walls	Soft to firm	Swelling to firm	Loose to firm	Weak and crushed	Fairly strong	Fairly strong	Strong	Strong	Weak hanging wall	Weak

[1] Mine in 1930 being equipped to handle 8,000 tons daily.
[2] Estimate.
[3] Estimated total of underground and open-cut mines.
[4] Total of all mines of company.

[9] All divisions of Copper Queen mine, 1880–1930, inclusive.
[10] Total for mine, 1904–30.
[11] All divisions of Copper Queen mine, 1880–1933.
[12] 1906–33, inclusive.

[5] Figures not available.
[6] 1926–31, inclusive.
[7] 1901–29, inclusive; all mines of company.
[8] District.
[13] Total for mine, 1904–34.
[14] Average.
[15] Where square-set method is used.
[16] Also 2.5 percent of nickel and $4 per ton in precious metals.

TABLE 26.—*Stoping data at 10 representative North American copper mines using square-set method of mining*

	Anaconda	United Verde Extension	United Verde (underground)	Magma	Copper Queen (Limestone Division)
Depth of ore mined (feet)	4,000	1,500	3,000	3,000	1,800.
Level interval (feet)	100, 200	100	150	125, 150	100.
Spacing of raises to level above (feet)	25–75	33	20–50	1 to a section	20–50.
Spacing of chutes under stopes (feet)	25–40	22	20–50	15–25	20–50.
Horizontal area of sections (sets)	Various	1 to 3 by 10	3 by 7, 2 by 10	14 feet by width of vein	
Method of advance of sections	Horizontal and rill	Vertical	Horizontal and rill	Downward	Vertical and horizontal.
Number of floors open at one time	2	2	1 horizontal, 2 rill	3	
Position of sill floor	Top of drift	Top of drift	Top of drift	Floor of drift	
Type of sets	Cap butting	Cap butting	Cap butting	Cap butting	Cap butting.
Size of sets (feet and inches)	5–4 by 5–4 by 7–9	5–6 by 5–6 by 7–2	5–6 by 5–6 by 7–2	5–0 by 5–0 by 6–8	5–0 by 4–7 by 7–10.
Kind of stope rounds drilled	Vertical and inclined		Horizontal	Underhand	Horizontal.
Method of handling ore in stopes	Shoveling and slides	Gravity with slides	Shoveling and wheelbarrows.	Gravity	Gravity and shoveling.
Method of placing filling	Gravity and cars	Gravity and cars	Gravity and cars	do	Gravity.
Source of waste filling	Sorting and development	Development	Surface and development	Surface and development	Old fills and development.
	Denn	**Colorada**	**Frood**	**Matahambre**	**Old Dominion**
Depth of ore mined (feet)	2,300	1,500	3,000	1,800	
Level interval (feet)	100	125	200	100, 130, 150	100, 200.
Spacing of raises to level above (feet)	30 or 35	30	1 to a stope	1 to a stope	30.
Spacing of chutes under stopes (feet)	20–30	30	17½–22	50	15.
Horizontal area of sections (sets)	5 feet by width of vein	3 feet by width of vein	45 feet by width of vein	Size of ore body	15 feet by width of vein.
Method of advance of sections	Vertical	Vertical	Horizontal	Horizontal	Horizontal.
Number of floors open at one time	Varies	2	2	1	2.
Position of sill floor	Top of drift	11 feet above top of drift	30 feet above rail	14 feet above drift	Top of drift.
Type of sets	Cap butting	Post butting			Cap butting.
Size of sets (feet and inches)	5–0 by 5–0 by 7–10	5–3 by 5–3 by 7–4			5–0 by 5–0 by 7–10.
Kind of stope rounds drilled	Horizontal	Horizontal	Horizontal	Horizontal	Horizontal.
Method of handling ore in stopes	Gravity with slides	Shoveling and wheelbarrows.	Shoveling and slides	Shoveling and wheelbarrows.	Shoveling and wheelbarrows.
Method of placing filling	Gravity	Scrapers, shoveling		Hydraulic (sand) shoveling and scrapers.	Slides, wheelbarrows, and shoveling.
Source of waste filling	Development and waste raises.	Surface and development	Surface and development	Development and mill tailing.	Development and waste raises.

the length of the stope, distance between levels, nature of the ground, use to which the raise is to be put, and requirements for ventilation. In heavy ground the raises may be 25 feet apart; in ground that stands better the distance may be as much as 75 feet, although that appears too great a distance for efficient ventilation in wide stopes. The length of the section is governed by the spacing of the waste raises. The floors are taken out one at a time, advancing from a raise toward the hanging wall. The ore is broken and removed and the timber placed in much the same manner as in narrow veins. In the heaviest ground only space for one set is broken at a time. If the ground stands well, room for two or three sets may be blasted out in one round.

UNITED VERDE EXTENSION

The United Verde Extension mine [48] is at Jerome, Ariz. The ore body was roughly 300 by 500 feet in horizontal section at the 1,400-foot level where it was largest; it tapered to a thin tail below the 1,700-foot level. The mine is worked through two vertical shafts. Because of the heavy, massive nature of the ore, its richness, and the necessity for mining it so that no subsidence would occur to generate enough heat to cause a fire, only the square-set system of mining has been used in the main sulphide lens. Complete extraction of the main ore body has been possible with practically no dilution. Moreover, square-setting allowed careful prospecting of the walls, which resulted in the finding of many small, rich lenses of ore that otherwise would have been missed.

The plan and section of a typical stope are shown in figure 58. The stope sections are usually three sets wide in fairly solid ore and two sets wide in heavier ground. If the ore is badly fractured it is sometimes removed in slices a single set wide. Slices are carried upward to the level above; they are usually from 10 to 20 sets long. Ore chutes are placed about every fourth set; alternate chutes have manways beside them. By so spacing chutes and leaving slides with grizzlies in adjoining sets, as shown in figure 58, shoveling of ore into chutes is virtually eliminated.

Pillars, usually about six sets wide, were left to protect the main haulageways; these became badly broken after standing for several years but were later mined successfully by the method shown in figure 59. A small vertical square-set cut is taken up through the center of the pillar, the top of the set is tied across with timber stringers, then the sides are sliced downward.[49]

UNITED VERDE

The United Verde underground mine at Jerome, Ariz., is worked through vertical shafts and an adit. All ore is either hoisted or dropped to the adit (1,000-foot) level for transportation to the company smelter or concentrator at Clarkdale, 4 miles in a straight line from the mine. The massive sulphide ore generally is very hard and dense and stands well over large areas without timber support, except for an occasional bulkhead where a slab is loose or where the ore body is weakened by an andesite dike. This condition permits mining about

[48] D'Arcy, R. L., Mining Practice and Methods at the United Verde Extension Mining Co., Jerome, Ariz.: Inf. Circ. 6250, Bureau of Mines, February 1930, 11 pp.
[49] D'Arcy, R. L., work cited.

65 percent of the ore by cut-and-fill methods. Of the total output, 27 percent is extracted by square-setting.

The cut-and-fill stopes, as described on page 228, are 30 to 160 feet wide and 60 to 200 feet long; pillars 30 to 40 feet wide are left between them.[50] Pillars are also left at the tops of these stopes. Square-setting is used to extract the vertical pillars between cut-and-

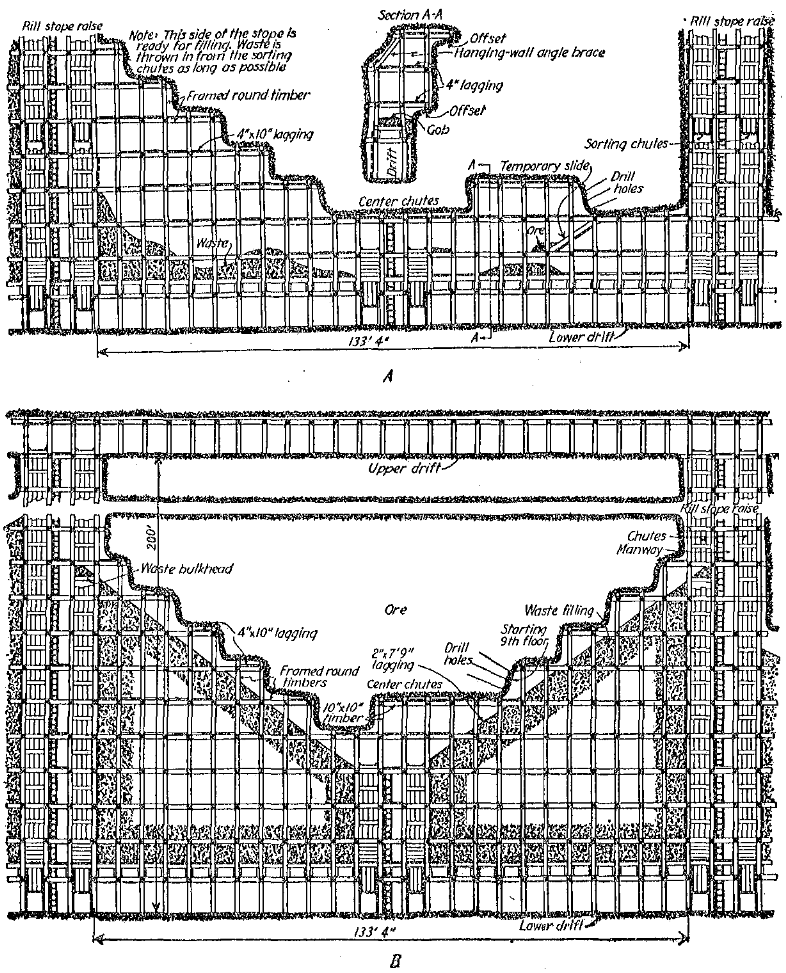


FIGURE 56.—Timbered rill stope, Butte: *A*, Early stage of stoping; *B*, stope completely developed.

fill stopes, in the crown pillars, and in some areas in the main stopes where the ground is heavy.

The size of square-set stopes varies considerably. In the vertical pillars the sections are necessarily small, rarely exceeding 36 sets in area. No more than two floors are opened at a time, and the fill is kept as close behind the shovelers as possible The fill is introduced

[50] Quayle, T. W., Mining Methods and Practices at the United Verde Copper Mine, Jerome, Ariz.: Inf. Circ. 6440, Bureau of Mines, 1931, 31 pp.

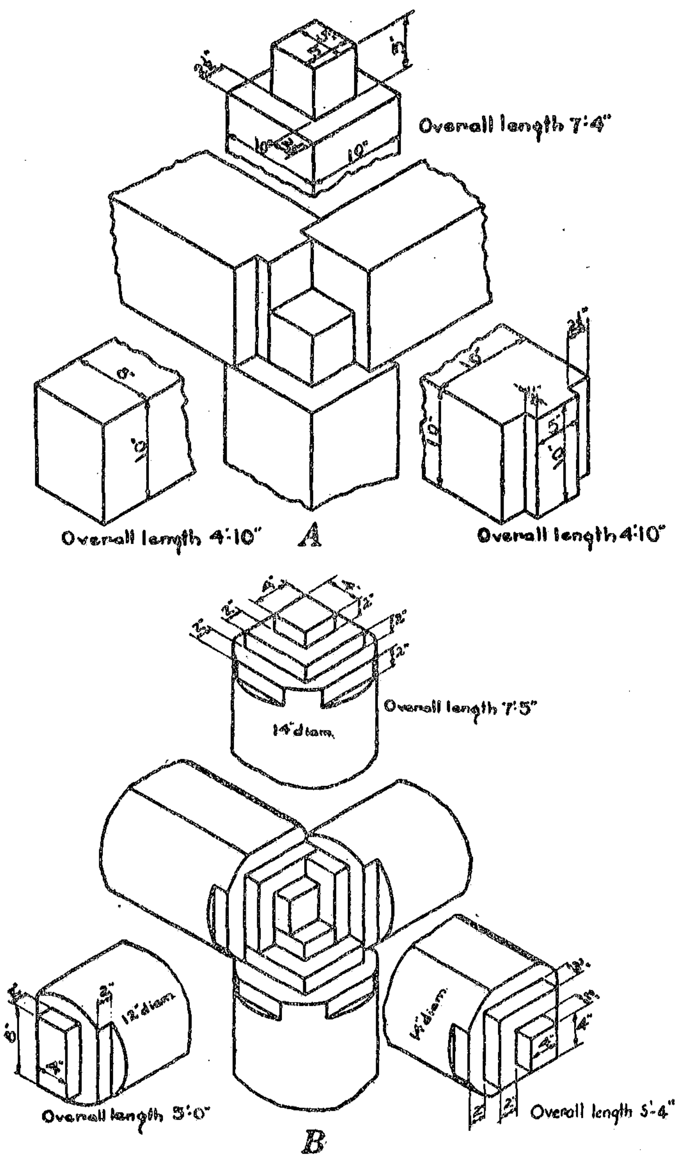


FIGURE 57.—Methods of framing square-sets: *A*, Cananea post butting joint; *B*, Rocker, or Anaconda, ca butting joint.

through a raise to the level above and distributed in the stope by narrow end-dump cars running on sectional track.

The crown pillars are mined by a rill method of square-setting. Stoping is started from a raise at the center of the cut-and-fill stope below. (See fig 60.) Four rows of sets on the second and two on

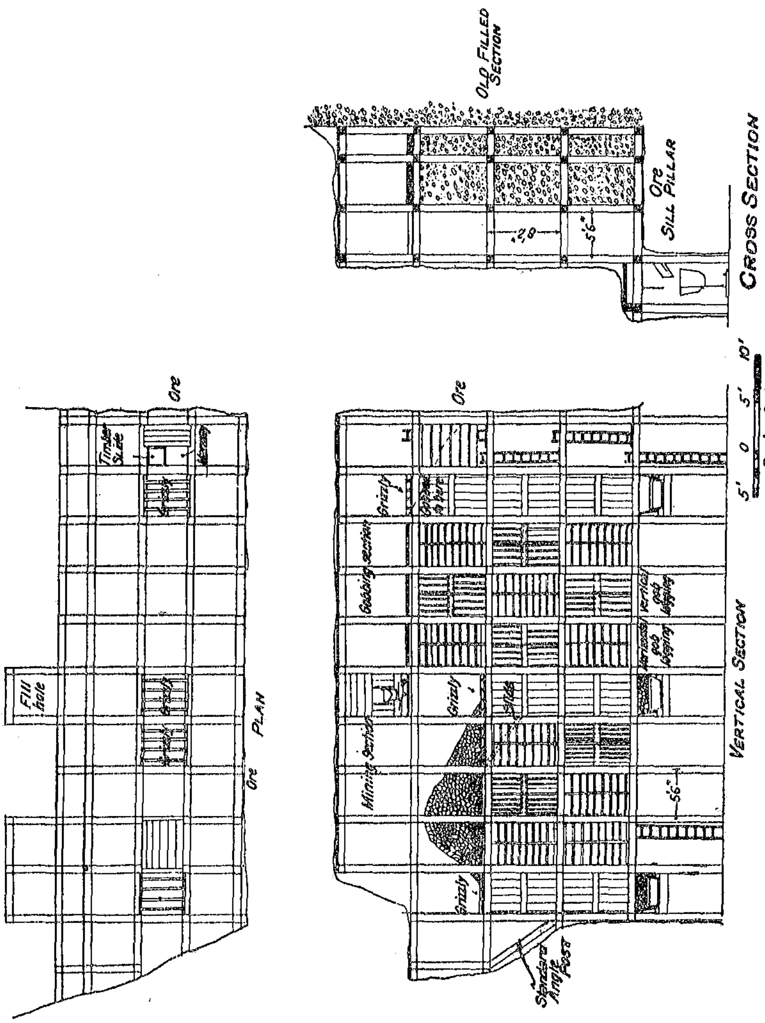


FIGURE 58.—Typical square-set stope, United Verde Extension mine.

the third floor usually are taken out before any fill is introduced. After the incline is established, stoping is started at the bottom of each new cut and progresses upward to the crest. One side of the rill is mined while the other side is being filled.

MAGMA

The Magma mine of the Magma Copper Co. is at Superior, Ariz. The ore is treated in a concentrator or sent directly to the smelter, both of which are near the mine. The ore occurs in a vein in diabase; the vein dips about 80° in the lower, active part of the mine. The ore is relatively hard, but the diabase wall rock is fractured and will not

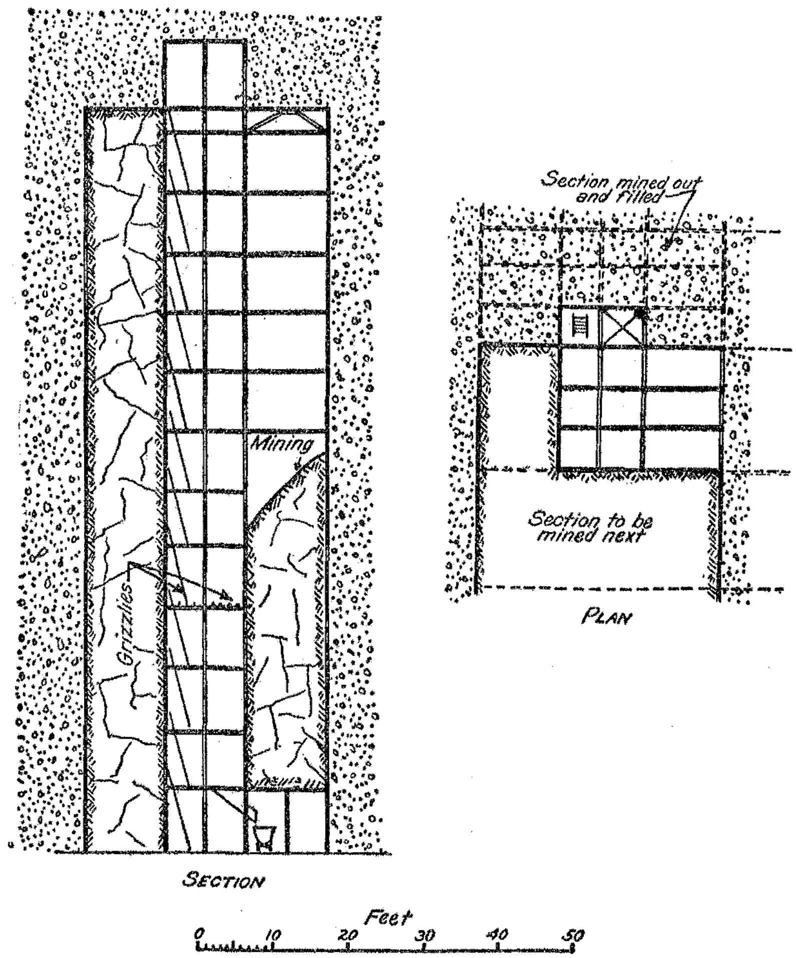


FIGURE 59.—Method of mining pillars, United Verde Extension mine.

stand without support. The main ore shoot is 1,200 to 1,500 feet long on the lower levels, averages about 25 feet in width, and has been mined from the surface to and below the 3,000-foot level. The mine is worked through six vertical shafts. In ore shoots more than 15 to 20 feet wide a combination rill-stope and pillar system is used.[51]

[51] Snow, F. W., Mining Methods and Costs at the Magma Mine, Superior, Ariz.: Inf. Circ. 6168, Bureau of Mines, 1929, 32 pp.

Rill stopes 16 feet wide and extending from wall to wall are first taken up to the level above, as described on page 234, and pillars 14 feet wide are left between. The pillars are then mined by a modified form of the Mitchell slice method. The top of the pillar is mined out and a line of segment sets placed immediately below the sill floor of the filled stope above. (See fig. 61.) The pillar is then mined downward, and a slope of about 40° down toward the footwall is maintained; the broken ore runs by gravity into the square-set raise that was used

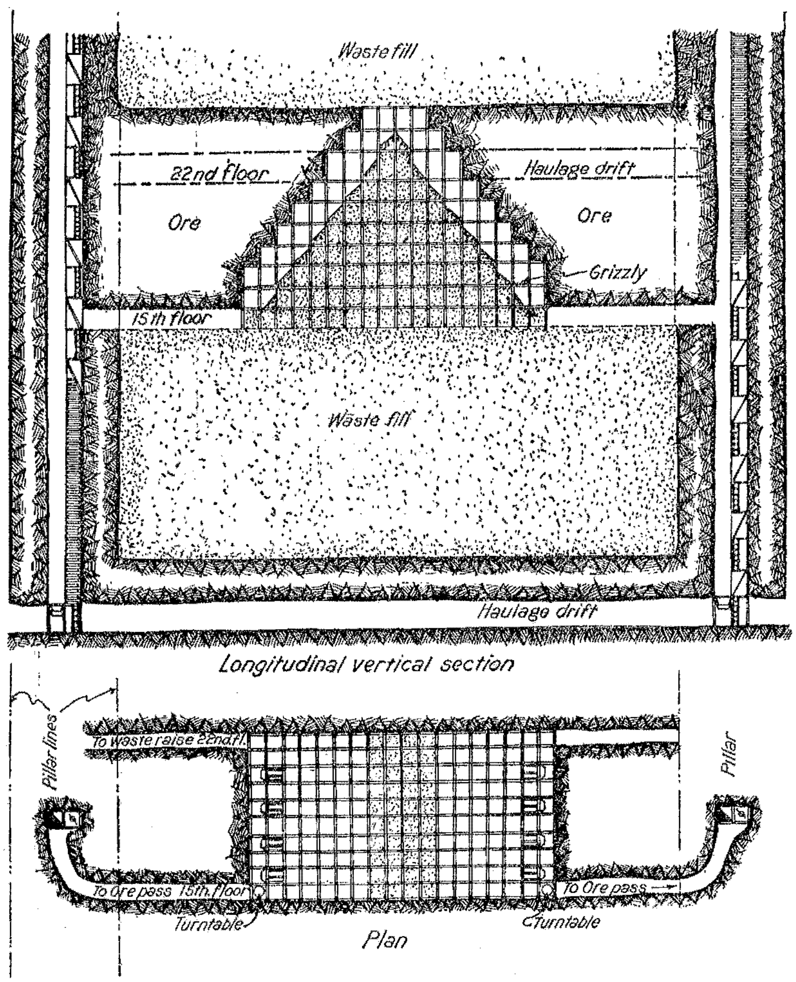


FIGURE 60.—Inclined square-set method of mining horizontal pillar, United Verde mine.

in the adjoining rill stope. As stoping progresses stringers are placed across the stope between corresponding posts previously placed in the rill stopes. Only enough ground is broken at one time to make room for one stringer.

The bottom of the rill stopes and unusually heavy ground is also mined by square-setting. About 60 percent of the mine output in 1930 was produced by this method.

LIMESTONE DIVISION, COPPER QUEEN

The Copper Queen mine of the Phelps Dodge Corporation, before the acquisition of the Calumet and Arizona mine, was worked as two divisions, the Limestone and the Porphyry.

The ore of the Limestone Division comprises principally sulphides and occurs as irregular, lenticular bodies in the limestone. The ore

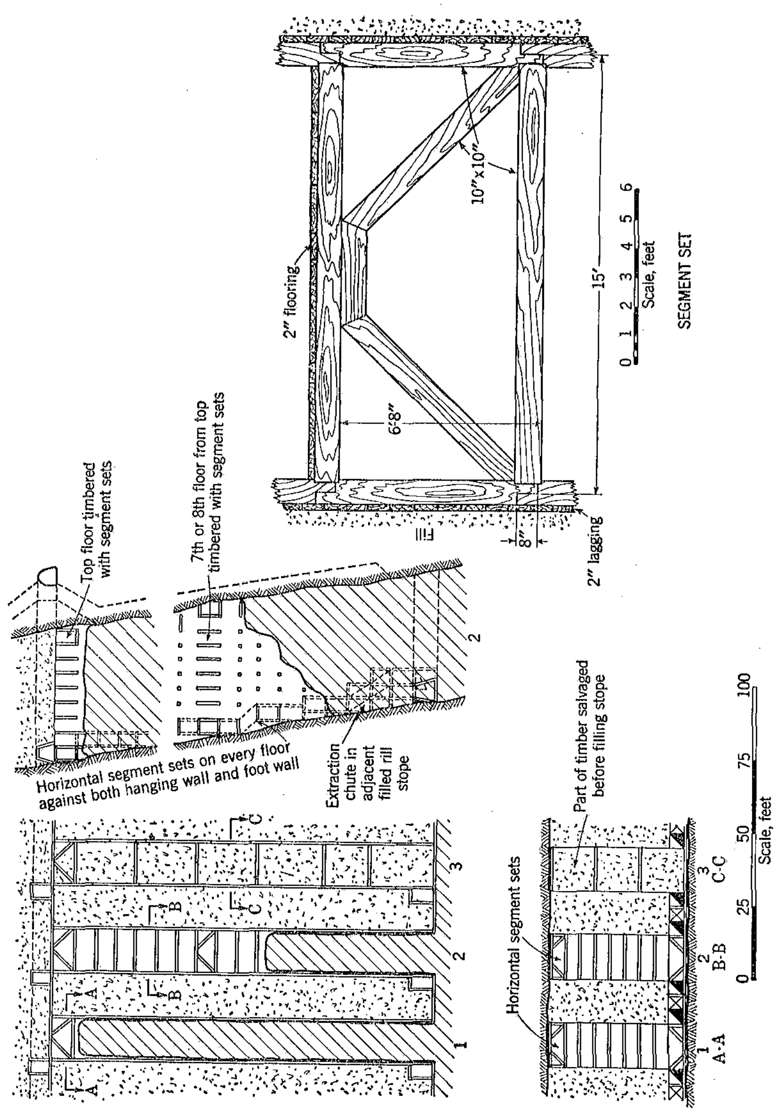


FIGURE 61.—Method of mining pillars in rill-stope-and-pillar system, Magma mine.

in the Porphyry Division consists of disseminated copper sulphides in porphyry.

The deposits in the Limestone Division have been worked mainly by square-setting, top slicing, and the inclined cut-and-fill method.

The method selected depends principally upon the structure of the ore and the need for selective mining. In general, the wall rock stands well. Where the ore will stand, a cut-and-fill method is used. Although the ore is hard to break, planes of weakness extend through the ore body in many places which permit masses, sometimes weighing hundreds of tons, to settle or fall where the back is not well supported. In such places either a regulation square-set or a top-slicing method is used.

Where the ore bodies are relatively weak and thick enough, top slicing is generally used. Square-setting used as an alternative to top slicing is described in the section on top slicing (p. 243). Fifty-five percent of the ore in this division (1932) was stoped by square-setting, 40 percent by top-slicing, and 15 percent by cut-and-fill stoping.

DENN

The Denn mine of the Shattuck-Denn Mining Corporation is at Lowell, Ariz., in the Bisbee mining district. It adjoins the Campbell Division of the Copper Queen group of mines. The ore comprises massive sulphides, and the ore body occurs in limestone. The ore and wall rocks are fairly strong, but large blocks break away if a back is left unsupported.

The mine is worked through a vertical shaft. The mining system is shown in figure 62. Preparatory to mining a drift is run in the foot wall about 20 feet from the ore; crosscuts are then extended to the handing wall on 40-foot centers. The ore body is laid out in stopes five sets wide over the crosscuts, with 15-foot pillars between. A raise is extended through each stope section from the crosscuts to the level above for ventilation and bringing in waste filling.

After the stopes on both sides of a pillar have been mined and the filling has settled, the pillar is mined by a method similar to that used at Magma. The pillar stopes at the Denn mine are filled only where necessary to prevent subsidence. This method of mining pillars was developed at Bisbee and is named the "Mitchell slice" system after the mine foreman who devised it.

The total cost of mining the ore body from 1926 to 1930 was $6.01 a ton.

COLORADA

The mines of the Cananea Consolidated Copper Co. are at Cananea, Sonora, Mexico. Most of the production during the last few years has been obtained from the Colorada ore body, which is a large pipe of sulphide.[52] In part (200 feet) of the ore body between the 500- and 600-foot levels, the walls were solid, but the massive sulphide ore was so blocky and shattered that it had to be mined by square-setting. In 1929 about 12 percent of the ore from the mine was produced by this method.

The width of the ore averages 60 feet on the 600 level and 20 feet on the 500 level. A drift was run in the center of the ore on the 600 level and timbered with sill-floor square-sets. Two-compartment raises were driven from it to the 500 level at intervals of 30 feet along the vein. These raises were on the dividing line between 15-foot stope sections and were offset as they progressed to keep them as nearly as possible in the center of the ore.

[52] Catron, William, Mining Methods, Practices, and Costs of the Cananea Consolidated Copper Co., Sonoro, Mexico: Inf. Circ. 6247, Bureau of Mines, 1930, 41 pp.

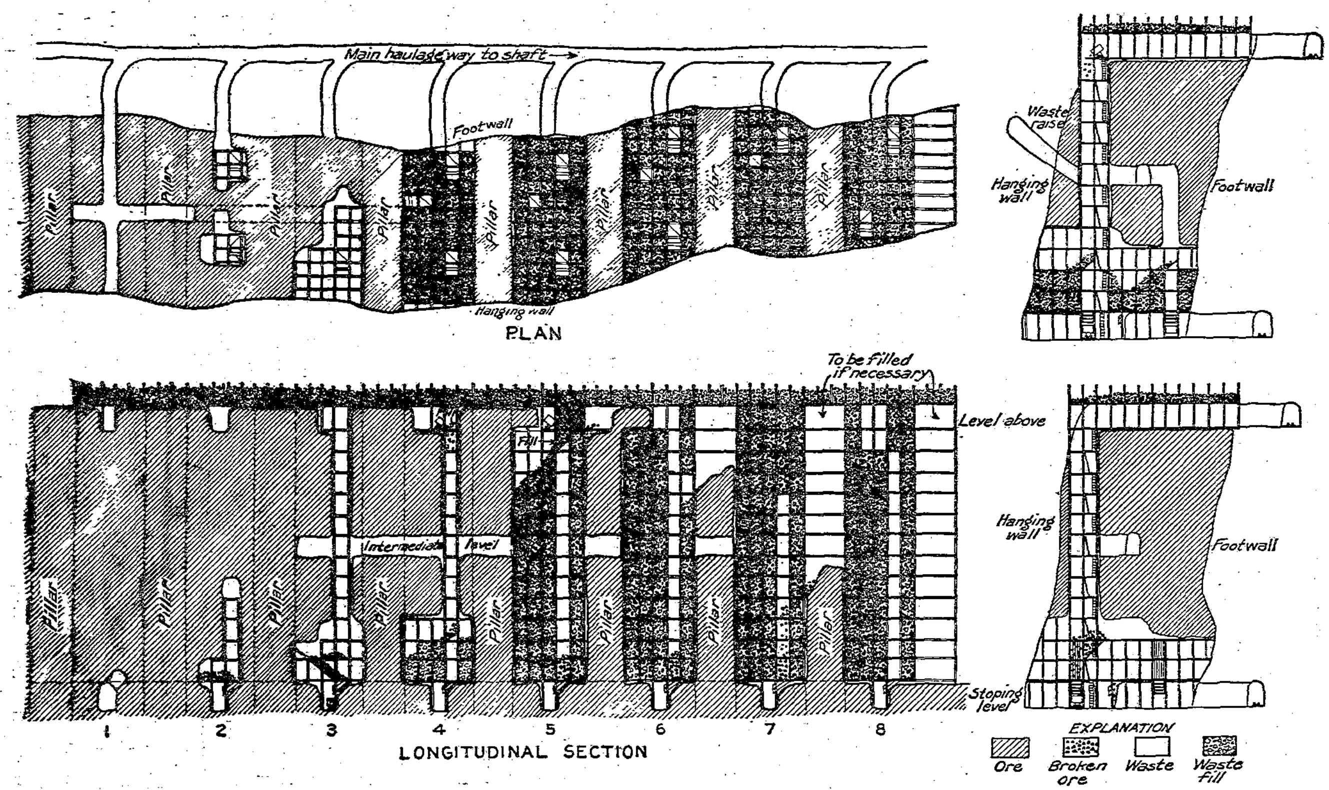


FIGURE 62.—Method of mining at Denn mine, Bisbee district, Arizona.

Eleven feet above the floor of the 600 level a 15-foot section was mined across the ore body in both directions from the raise and timbered with standard 10- by 10-inch square-sets.

The next floor was mined by starting at the raise and blasting the ore down onto the first floor, whence it was shoveled into the chute. Fill was then run in from the level above and a temporary floor laid over it. While one side of a raise was being filled, the ore was being broken on the other, thus speeding the work and leaving as little ground as possible—never more than two floors. Each section was mined vertically to the level above; mining of adjoining sections followed closely in proper order.

FROOD

The Frood mine of the International Nickel Co. is near Sudbury, Ontario. The ore is mined both for its copper and nickel content; it is treated at a concentrator and smelter at Copper Cliff. The ore body is of higher grade at depth than near the surface. At the lower levels it consists of massive sulphide 40 to 200 feet wide. The mine is worked mainly through a new vertical six-compartment shaft, 3,045 feet deep. The level interval is 200 feet. Every other level is a haulage drift. The ore body contains rock inclusions which must be sorted, and as the upper levels must be preserved close filling is required.[53] The ground is not heavy and does not cave, but small slabs as much as 8 inches thick and a few feet square drop out of the back without warning. The horizontal cut-and-fill method was the main one used, but horizontal jointing and the weight of the massive sulphide ore have necessitated a change to square-setting in most places.

Square-set stopes are 45 feet wide and run across the ore body with 35-foot pillars between. To eliminate the fire hazard of the timber as much as possible two intermediate methods were tried in 1930 and 1931.[54] The first was a semishrinkage method in which a row of standard square-sets was carried up along each rib pillar and tied to it by rock bolts and wire cable. Four or five cuts of the stope were then mined by shrinkage stoping; simultaneously the square-sets were carried up with the back and tied to the pillars. The ore was then drawn and replaced by fill and the cycle repeated.

The second method was a modified form of standard square-setting. The square-set stope was divided longitudinally into three sections by two rows of standard square-sets. (See fig. 63.) These were connected to each other and the pillars by rectangular sets in which the girts were removed in order to create a firebreak between the standard sets and the pillars. Neither of these intermediate methods provided support enough for the back, and they were finally abandoned in favor of standard filled square-set methods with little increase in cost.

MATAHAMBRE

The Matahambre mine of the American Metal Co., Ltd., is on the north coast of Cuba. The ore bodies occur in large, irregular pipes, usually lenticular in cross section. The dip ranges from 42° to 45°.

[53] Mutz, H. J., Mining the Frood Ore Body at Depth: Eng. and Min. Jour., vol. 130, Nov. 10, 1930, pp. 445–462.

[54] Ontario Department of Mines, Twenty-five Years of Ontario's Mining History: Bull. 83, 1932, pp. 27–28.

The hanging wall, comprising thin-bedded shales, will not stand if left unsupported for a height of more than 10 feet.[55] The ore body is as much as 275 feet long and about 30 feet wide. The mine is operated

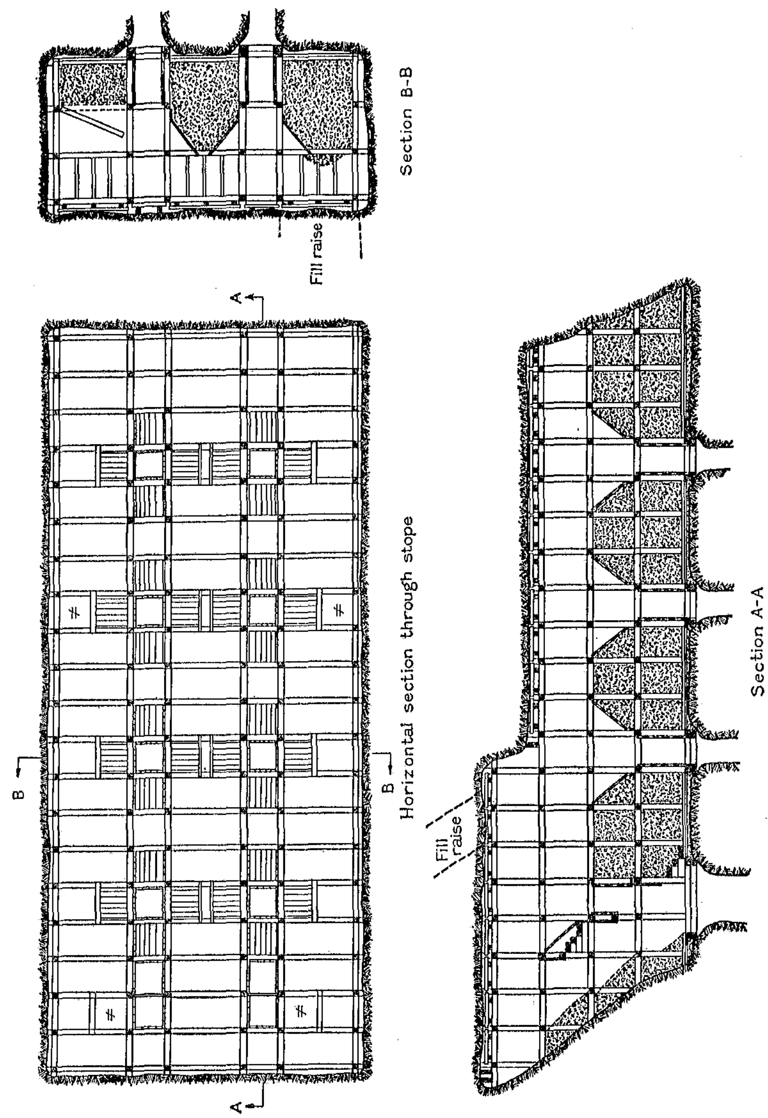


FIGURE 63.—Square-setting, Frood mine.

from a vertical three-compartment shaft; ore is produced from 13 levels.

Cut-and-fill stoping, as described on page 226, is the principal method used. In some of the ore bodies the ground is so fractured that square-sets are required to hold the stopes open. The upper 15 or 20

[55] Richert, G. L., Mining Methods at Minas de Matahambre, Pinar del Rio, Cuba: Inf. Circ. 6145, Bureau of Mines, 1929, 18 pp.

feet of all cut-and-fill stopes is also mined by square-setting. During 1928 about 30 percent of the ore produced was mined by square-setting. All sets are filled as soon as a cut is completed. This permits carrying the stopes up past the level and removing the ore from the level pillar of the stope above.

OLD DOMINION

The Old Dominion mine at Globe is operated through a vertical shaft. Three other shafts are maintained for ventilation and emergency exits. Haulage levels are 200 feet apart vertically, and intermediate mining levels are midway between them. When a level is opened a drift is first run in the hanging wall about 40 feet from the footwall of the vein. Crosscuts are then run through the vein at 240-foot intervals. After an ore body has been proved an extraction drift is run on the footwall in the vein.

Square-setting is used for mining relatively high grade ore in areas where the overlying country rock must be supported. In prospecting ground that is to be mined by this method raises are run on the footwall at intervals that are multiples of 30 feet so that they will fit into a stoping scheme that allows one raise for every two 15-foot sections. The sections are run the width of the vein. The opening of more ground than 15 feet at one time is unsafe in nearly all square-set stopes.[56] Usually not over two floors are taken out before filling.

The broken ore is generally shoveled or wheeled into the chutes. In wide stopes slides may be used or another chute carried up toward the hanging wall by keeping a set open and lined off.

Filling is obtained from development work or waste raises in the foot wall. The foot-wall side of the stope usually is kept two or more sets higher than the hanging-wall side. Slides are used to deflect the waste toward the hanging wall; however, part of the waste is placed by wheelbarrows or hand shoveling. The wheelbarrows are filled through a temporary chute built in the square-sets of the floor above. Where the stope is more than eight sets wide a raise may be run into the hanging wall to obtain filling for the hanging-wall side of the stope.

BOLEO

The method of mining a flat bed at the mine of the Campagnie du Boleo at Boleo, Baja California, Mexico, does not come under any of the classifications in this paper but approaches square-setting more nearly than any other. The ground is extremely heavy, and close timbering and back filling are practiced. A short-wall method of mining is used. Successive cuts of ore are taken along the strike, and the space made by the previous cut is filled with waste stripped from the ore; mining and filling are done contemporaneously.[57] The back at the working face is held up by regular rows of stulls placed normal to the back.

The bed containing the ore dips about 15 percent (8° or 9°). The ore is about 2 feet thick and can be broken readily with a pick. It is covered by soft, wet clay, which is under hydrostatic pressure.

[56] Shoemaker, A. H., Mining Methods at the Old Dominion Mine, Globe, Ariz.: Inf. Circ. 6237, Bureau of Mines, 1930, 21 pp.

[57] Bellanger, M., Mining Copper in Baja California: Eng. and Min. Jour., vol. 132, Nov. 9, 1931, pp. 394-397.

The footwall is either hard conglomerate or soft, sandy tuff. (See fig. 16, p. 84.) The ore is developed by winzes driven 200 meters (656 feet) apart from haulage drifts. It is blocked out by drifts from the incline 60 to 70 meters (196.8 to 229.6 feet) below the haulage level. The average vertical distance between levels is 20 meters (65.6 feet).

A water temperature of 120° F. and slow oxidation of the sulphide ore make good ventilation necessary.

After an area of ore is blocked out and the ore sampled the exploratory workings, although timbered, are abandoned as it is impossible to hold them open any length of time. Extraction inclines and drifts are then run in the footwall 6 meters (19.7 feet) below the bed in a formation that will stay open. The levels are 150 meters (492 feet) apart and the inclines 200 meters (656 feet) center to center. Sublevels are run 30 meters (98.4 feet) apart from the inclines.

Stoping operations commence at the upper end of a block at the incline and are advanced at right angles to the incline. The ore is broken from the face by pick and shoveled by hand onto a shaker conveyor running parallel to the incline. This conveyor empties onto a second one in the subdrift, which in turn discharges onto a third one in a footwall incline. The third conveyor deposits its load into a loading chute at the haulage level.

After the upper stope has reached an appropriate distance a second one is started in the next lift below. Belt conveyors used in the second lift discharge on the conveyor in the subdrift above. The lower two lifts are mined in the same manner as the upper two. The stopes are advanced 60 meters (196.8 feet) from the incline and then abandoned. Further advance is impractical on account of the expense of maintaining the subdrifts. The timber must be replaced twice in a 60-meter (196.9-foot) advance. When the mined-out area along the whole incline has reached a uniform distance of 60 meters (196.9 feet) stoping is discontinued and started in the next incline 200 meters (656 feet) distant. The next stope advances toward the abandoned workings. The method of advance is directly opposed to that practiced in long-wall mining in coal mines, as the pressure ahead is much greater than that behind the working face. Keeping a drift open ahead of a working face is practically impossible. The change of equilibrium in the ground due to stoping is felt as far as 100 meters (328 feet) ahead of the stope face.

Although only 2 feet of the bed is pay ore, 6 feet is mined. Waste is thrown back of the conveyors and fills the mined-out area. Surplus waste is hoisted.

Stope timbering in normal ground comprises a series of stulls spaced 1.3 meters (4.27 feet) apart in straight rows. Caps and girts are used on the stulls, and the back is tightly lagged. When necessary temporary stulls are placed 30 centimeters (1 foot) ahead of the last row in the working face. Heavy ground is supported by cribs of 4- by 4-inch pine. To protect the men from falls top lagging is carried close to the face.

The shaker conveyor consists of a series of 9-foot sections coupled with eyelets and bolts. The driving mechanism and small motor can be placed under one of the troughs of the conveyor or at the elevated end. Three men can easily move a 30-meter (98.4-foot) conveyor in less than 2 hours. The 20-inch belt conveyors used in the subdrifts and the 26-inch belt conveyor used in the inclines are built in 6-foot sections and are equipped with automatic couplings that facilitate lengthening or shortening the conveyors according to requirements. The conveyors have an over-all height of 18 inches and are driven by 5-, 8-, or 10-horsepower, fully enclosed motors. The return belt is protected by spill plates. The 20-inch belts are of 4-ply and the 26-inch of 5-ply rubber-impregnated cotton fabric with rubber-coated faces. Maintenance cost is $0.15 per ton of ore mined.

All stoping is done on contract; a section of a face 2.2 meters (7 feet) long is allotted to each contractor. Each face advances 1.1 meters (3.6 feet) per shift. The output per man-shift was 2,126 kilograms (2.34 tons) of ore. The cost of mining a square meter with conveyors was 3.59 pesos. The daily rate of pay for miners is 5.95 pesos.

The accident rate has been reduced from 6.28 per 1,000 man-shifts in 1927 to 2.22 in 1930 and 0.68 in March 1931.

BRITANNIA

The mines of the Britannia Mining & Smelting Co. Ltd., are at Britannia Beach, about 20 miles from Vancouver, British Columbia. The stock of the company is held by the Howe Sound Co. The ore is treated in a flotation mill at the portal of a lower haulage level. The mines were developed through adits; the mining levels are connected by shafts. The haulage system comprises an adit on the 2,700 level and another on the 4,100 level, connected by a rock raise. The lower adit is being extended for a new rock raise 14,000 feet from the portal.[58] The ore is crushed to minus 5 inches at the rock raise.

The ore bodies occur in a shear zone about 7,000 feet long. The dip ranges from 65° to 70°. The principal mines of the company are the Fairview, Bluff, Victoria, and Empress. Most of the ore is won by shrinkage and glory-hole methods. Square-setting, however, is the chief method used in the Victoria and Empress mines. The walls of the veins in these two mines are generally weak and the ore is blocky. The width of the veins ranges from one to six sets. A rill cut-and-fill method is used in some of the narrower veins, but the method has been found to be only slightly cheaper than square-setting.

The square-set stopes are silled out at the level. Stope sections are 36 feet long and the width of the vein. Two-compartment raises are maintained at either end of a section; the raise next to the unmined ore of a finished section serves the next one mined. Four floors are taken out in a stope at a time, then the space is filled. This is repeated until the level above is reached. Flat holes are drilled and break to the shoveling floor below. Slides are used in the stope to reduce shoveling of the ore and to fill the stopes with waste brought from above through the raises. The corners of the stopes which cannot be reached by the waste slides are not filled. Timbermen place the square-sets, line the chutes, and install the gob lining in the sections. A crew in a large stope consists of one miner, one timberman and helper, and two muckers. Square-sets are 6 feet in horizontal section and 7 feet 6 inches high; the posts and caps are of 10 by 10 and the girts of 6 by 10 timber.

[58] Brennan, C. V., Mining Operations at the Property of the Britannia Mining & Smelting Co., Britannia Beach, British Columbia: Inf. Circ. 6815, Bureau of Mines, 1935, 37 pp.

Filling is of major importance; the best results are obtained when not too much ground is opened at a time. Glacial material, which covers bedrock to a depth of about 100 feet, is used chiefly for filling. It contains considerable clay with sand and boulders; it sets very tightly, especially when wet, with virtually no shrinkage. At the Victoria mine, two waste raises on the east and west ends of the ore body connect with waste glory holes. One miner is employed steadily for the two glory holes. He keeps the glory holes filled from coyote holes and blasts the larger boulders. The waste is drawn through grizzly chambers on the top level of the mine. The grizzly bars are set 10 inches apart. Considerable plugging is necessary on the grizzlies, and two or more men are employed at each one. Waste is drawn into cars for distribution at each of the mining levels or bypassed to the level below.

SHRINKAGE STOPING

Shrinkage stoping is used for mining deposits with strong walls that dip usually at least 50°. The method is intermediate between open stoping and cut-and-fill stoping. It grades into undercut block caving when the ore is broken from workings above the back of the stope. The ore in shrinkage stopes partly supports the walls, and the method can be used in some places where open-stope mining would not be practicable. The method cannot be used where sorting in stopes is necessary. Shrinkage stoping appears to be best adapted to mining deposits 6 to 15 feet thick, but ore shoots as thin as 3 feet and as wide as 75 feet are mined successfully by this method. The stopes reach a maximum height of 200 feet. Where the dip approaches the angle of repose shorter intervals are desirable, because the ore tends to hold back in a stope. Moreover, exceedingly high stopes tie up relatively large quantities of ore during mining. The manner of attacking the ore varies less with shrinkage stoping than with most other underground methods. Manways are built up through the broken ore or maintained in pillars at about 100-foot intervals.

In relatively narrow veins, stopes may be taken from level to level without leaving pillars, except to protect the upper level. Usually, however, pillars are left at the top of the stopes above haulage levels and between stope sections. In large deposits sections are taken across the ore body, and pillars are left between sections.

The length of stopes and width of pillars are governed by ground conditions. In good ground stopes may be as much as 500 feet long, with 50-foot pillars between; in weaker ground stopes may be only 50 feet long, with 50-foot pillars. Pillars may be blasted and pulled with the ore in the stope sections, blasted and recovered after the adjoining stopes have been emptied, mined by some other method after the shrinkage stopes are filled, or left in place.

Where the ore breaks into large fragments grizzly chambers protected by small pillars generally are provided above the haulage levels.

Tables 27 and 28 list the principal copper mines using shrinkage stoping in North America, production data, character of the deposit, and stoping details. Subsequent pages describe the methods at representative mines.

TABLE 27.—*Production data at representative North American copper mines using shrinkage methods of mining*

	Walker	Engels	Eighty-five	Verde Central	Kennecott [1]	Isle Royale	Britannia	Creighton	Levack	Garson	Beatson
Location	Spring Garden, Calif.	Engelmine, Calif.	Valedon, N. Mex.	Jerome, Ariz.	Kennecott, Alaska	Houghton, Mich.	Britannia Beach, British Columbia	Creighton Township, Ontario	Levack, Ontario	Garson, Ontario	Latouche, Alaska
Normal daily production of ore (tons):											
Total	1,600	1,300	300	350	350	1,700	7,000	[2] 5,000	[2] 3,000	[2] 1,600	1,500
By shrinkage	1,600	1,300	225	350	350	1,700	[3] 5,000	[3] 4,500	3,000	1,600	1,500
Total yearly production of ore (tons):											
1929	458,000	395,000	59,000	[4] 93,000	80,000	515,000	1,920,000	1,177,000	369,000	246,000	453,000
1930	519,000	[5] 177,000	81,000	[4] 93,000	58,000	510,000	2,152,000	862,000	[6] 0	278,000	445,000
1931	432,000	0	(7)	0	56,000	353,000	2,022,000	301,000	0	210,000	0
1932	35,000	0	0	0	31,000	58,000	809,000	97,000	0	56,028	0
1933	0	0	0	0	0	0	623,000	383,000	0	0	0
1934	0	0	0	0	0	0	786,000	823,000	0	0	0
Total yearly production of copper (pounds):											
1929	15,032,000	11,000,000	[3] 3,000,000	4,335,000	[3] 18,060,000	10,864,000	41,972,000	(8)	(8)	(8)	[3] 11,000,000
1930	15,776,000	[5] 4,073,000	[3] 4,000,000	4,280,000	[3] 16,270,000	10,659,000	44,294,000	(8)	0	(8)	[3] 9,500,000
1931	12,850,000	0	(7)	0	[3] 12,320,000	7,731,000	27,944,000	(8)	0	(8)	0
1932	1,070,000	0	0	0	[3] 5,380,000	1,403,000	11,497,000	(8)	0	(8)	0
1933	0	0	0	0	0	0	7,990,000	(8)	0	0	0
1934	0	0	0	0	0	0	11,326,000	(8)	0	0	0
Total production of ore to end of 1930 tons	[3] 2,750,000	4,693,000	[3] 1,400,000	186,000	----------	[9] 12,849,000	15,000,000	[2] 18,000,000	----------	----------	[3] 6,000,000
Total production of ore to end of 1934 tons	3,217,000	4,693,000	----------	186,000	----------	13,260,000	19,240,000	19,604,000	----------	----------	6,000,000
Total production of copper to end of 1930 pounds	[3] 105,000,000	159,500,000	----------	8,600,000	[3] 925,000,000	265,292,000	418,000,000	(8)	(8)	(8)	[3] 200,000,000
Total production of copper to end of 1934 pounds	118,920,000	159,500,000	----------	8,600,000	943,000,000	274,426,000	476,757,000	(8)	(8)	(8)	200,000,000
Kind of deposit	Shoots in shear zone	Shoots in shear zone	Vein	Vein	Replacement	Bedded	Shoots in shear zone	Shoots in shear zone	Shoots in shear zone	Shoots in shear zone	Shoots in shear zone
Length of ore body feet	200–1,000	830	1,600	Varies	150–1,000	----------	300–800	500–1,000	Up to 700	Up to 700	800
Width of ore body do	[10] 15	[11] 100	[10] 5	Varies	Up to 100	[10] 9	10–250	50–300	Up to 400	Up to 100	[11] 340
Dip of ore body degrees	55–75	80	80	80–90	40–90	56	65–70	45	35–55	55	60–70
Grade of ore percent of copper	1.7	1.5	2.9	2.6	13	1.0	1.0	[12] 4.6	----------	----------	1.3
Depth of ore mined feet	----------	2,000	2,000	1,930	----------	[13] 5,000	----------	2,530	700	1,400	----------

[1] Including Mother Lode.
[2] Tonnage hoisted; considerable waste is removed by sorting in rock house.
[3] Approximate estimate.
[4] Started January 1929; closed October 1930.
[5] Closed July 1930.
[6] Not operated 1930–35.
[7] No available figures.
[8] Total for company in square-set table.
[9] 1901–30.
[10] Average.
[11] Maximum.
[12] Also 5.8 percent nickel (grade of all ores smelted in 1929).
[13] On incline.

TABLE 28.—*Geological and stoping data at representative North American copper mines using shrinkage methods of mining*

	Walker	Engels	Eighty-Five	Verde Central	Kennecott [1]	Isle Royale
Location	Spring Garden, Calif.	Engelmine, Calif.	Valedon, N. Mex.	Jerome, Ariz.	Kennecott, Alaska	Houghton, Mich.
Level interval (feet)	150	200	150	150	150 and 200	100.[2]
Spacing of raises to level above (feet).	90–500	90–120	100	1 or 2 to a stope		
Spacing of chutes under stopes (feet).	50	30	10	25	25–35	18.
Spacing of manways (feet)	100	90–120	100	100		
Length of stopes (feet)	50–500	90–120	100	100	560 [3]	
Maximum area of stope open at one time (feet).	15 by 500	50 by 120	5 by 100	10 by 100		
Where stopes silled	15 feet above top of drift.	18 feet above top of drift.	At drift	At drift	25–30 feet above drift	At level.
Method of advance or attack	Flat back		Stepped back	Stepped back		
Kind of rounds drilled	Horizontal	Horizontal	Vertical	Vertical	Horizontal	Horizontal.
Grizzly chamber used	Yes	No	No	Yes	No	No.

	Britannia	Creighton	Levack	Garson	Beatson	Hornet
Location	Britannia Beach, British Columbia	Creighton Township, Ontario	Levack, Ontario	Garson, Ontario	Latouche, Alaska	Mathewson, Calif.
Level interval (feet)	200	120	120	100 and 200		
Spacing of raises to level above (feet).	1 to a stope		1 to a stope		40 by 60	180.
Spacing of chutes under stopes (feet).	33⅓	15	16 by 50		34 by 40	25.
Spacing of manways (feet)	300	75	1 to a stope			180.
Length of stopes (feet)	300	Up to 250	Up to 400	Up to 100	70–270	150.
Maximum area of stope open at one time (feet).		50 by 250	100 by 400		70 by 270	80 by 150.
Where stopes silled	30 feet above level	25 feet above level	25 feet above level	25 feet above level		25 feet above level.
Method of advance or attack	Stepped back	Flat back	Flat back	Flat back	Stations in raises	Powder drifts.
Kind of rounds drilled	Horizontal	Horizontal	Horizontal	Horizontal	Fanned	
Grizzly chamber used	Yes	No	No		Yes	Yes.

[1] Including Mother Lode. [2] On incline. [3] Average.

WALKER

The Walker mine of the Walker Mining Co., a subsidiary of the Anaconda Copper Co., is about 9 miles from Spring Garden, Calif., a station on the Western Pacific R. R. The ore is milled in a 1,600-ton flotation plant at the mine; the concentrates are transported to the railroad by aerial tramway. Operations were suspended from February 1932 until 1935.

The ore bodies are persistent shoots in a mineralized zone. The wall rock is hard and generally stands well. The ore bodies are admirably adapted to shrinkage stoping.[59]

The mine is worked through a 10,000-foot adit and an interior, three-compartment inclined shaft. Level intervals above the adit are 150 feet, except the first, which is 180 feet to allow for a grizzly level 30 feet above the adit. Below the adit, haulage levels are 500 feet apart with two working levels between, according to a letter from H. A. Geisendorfer, general manager, Walker Mining Co. Main ore chutes on the haulage levels are placed on 50-foot centers, and each is connected by a raise to a grizzly chamber in a subdrift 30 feet above. The grizzly chambers are protected by small pillars above the grizzly drift, and each is connected to the stope by two finger raises that tap the stope at about 25-foot intervals.

In good ground stopes may be 400 to 500 feet long with 50-foot pillars between, in which are two-compartment raises that serve later in mining the pillars. In poor ground stopes may be only 50 feet long with 40-foot pillars between, likewise provided with raises. In the long stopes small, cribbed manways are carried up every 100 feet. Horizontal pillars are left at each working level; the one below is about 25 feet and the one above about 15 feet high.

Stope rounds consist of flat holes drilled with mounted drifter-type drills. During mining on the upper levels one-third of the ore (to allow for the swell) was drawn and transferred to the main ore chute. When back stoping was completed the level pillars were broken, beginning at the end farthest from the shaft; the ore in all the stopes above the main adit was then drawn directly to this level. Boulders are broken in the grizzly chambers.

In open ground pillars of good ore between stopes are partly recovered after the stopes are drawn by trimming them down. Pillars of relatively low grade ore are not mined. Where the hanging wall is weak and the ore high grade, pillars are extracted by the square-set method. Only a few stopes, however, have been worked by this method. In 1932 one section of the ore body lying next to a fault was being prepared for mining by a cut-and-fill method.

ENGELS

The mines of the Engels Copper Mining Co. are at Engelmine, Calif. The normal output from the two mines, which are about 3 miles apart, is about 1,000 tons a day. The ore is treated in a concentrator at Engelmine. The ore occurs in shoots in shear zones and is similar to that at the Walker mine. The size of the shoots varies; the largest was 830 feet long and 100 feet wide. The mines are worked through adits and interior shafts. The level interval in the lower part of each mine is about 200 feet.

[59] Young, George J., Anaconda's Walker Mine: Eng. and Min. Jour.-Press, vol. 117, May 3, 1924, p. 725.

In the main ore body stopes are 90 to 120 feet long with 20-foot pillars between; they are laid out across the ore body.[60] The pillars coincide vertically with those above and below. Manways are installed in the pillars, and crosscuts give access to the stopes every 25 feet.

Ore chutes are 30 feet apart on the footwall. First the chute raises are driven 14 feet above the rail; standard chutes are installed, and the raises are extended to a height of 32 feet and belled out to connect with one another, which completes the first or cutting-out stage of stoping. An 18-foot pillar is left between the top of the drift and bottom of the stope.

Mining stops within 20 feet of the next level until the stope above is emptied. When the stopes are emptied of ore the vertical pillars are mined, and most of the resulting broken ore is drawn from the stope chutes. The firmness of the walls makes this method of mining possible.

The mine was closed in 1930.

EIGHTY-FIVE

The Eighty-Five mine of the Phelps Dodge Corporation (formerly a part of the Calumet & Arizona holdings) is at Valedon, 3 miles from Lordsburg, N. Mex. The ore is shipped to the smelter at Douglas, Ariz., where its high silica content makes it valuable as a flux. The ore occurs in a more or less continuous shoot in a fissure vein with an average dip of 80°. The ore shoot is about 1,600 feet long and has been opened to a depth of 2,000 feet. The mine is worked through a vertical shaft with levels at 150-foot intervals. The ore is stoped by the shrinkage and cut-and-fill methods. Shrinkage has gradually superseded cut-and-fill stoping, and in 1930 about 85 percent of the output came from shrinkage stopes.

In preparation for mining by shrinkage the ore is first opened by drifting.[61] If the ore is of high grade an extraction drift is run in the footwall so that pillars need not be left to hold the level open. Stoping begins at the end of the ore body nearest the shaft, where a raise has been driven to the level above. Stopes are 100 feet long. Chutes are built on 10-foot centers. (See fig. 64.) Pillars are sometimes left above the drifts where the walls are weak or the vein is unusually wide.

To permit a steady production one stope is prepared for mining, one is mined, and a third is drawn at the same time. As the ore in a stope is drawn, waste from the hanging wall sloughs off and follows the ore down. To prevent excessive dilution the ore must be drawn evenly.

The mine was closed at the end of 1931.

VERDE CENTRAL

The Verde Central mine is at Jerome, Ariz. It was formerly operated by the Verde Central Mines, Inc., a subsidiary of the Calumet and Arizona Mining Co., but was sold to the United Verde Copper

[60] Nelson, W. I., Mining Methods and Costs at the Engels Mine, Plumas County, Calif.: Inf. Circ. 6260, Bureau of Mines, 1930, 22 p.

[61] Youtz, R. B., Mining Methods at the Eighty-Five Mine, Calumet & Arizona Mining Co., Valedon, N. Mex.: Inf. Circ. 6413, Bureau of Mines, 1931, 26 pp.

Co. in 1931. Production at the Verde Central mine began January 1, 1929, and operations were discontinued late in 1930.

The ore was concentrated at the mine, and the concentrates were shipped by truck to Jerome then by rail to the United Verde Extension smelter at Clemenceau, about 10 miles away.[62] The ore occurs in shoots in a vein with very firm walls. No ore has been found above the 600-foot level. The mine was worked through a three-compartment shaft 1,930 feet deep, with levels 150 feet apart.

The ore was developed to the 1,750-foot level before stoping was begun. A sequence of stoping the various ore shoots was followed, so that the average hoisting distance would be between the 1,000- and 1,200-foot levels (the "center of gravity" of the ore reserves), that an average mill feed would be obtained, and that ore would not have to be trammed through worked-out sections of the mine. As waste was disposed of in worked-out stopes some development work was deferred until a nearby stope was finished.

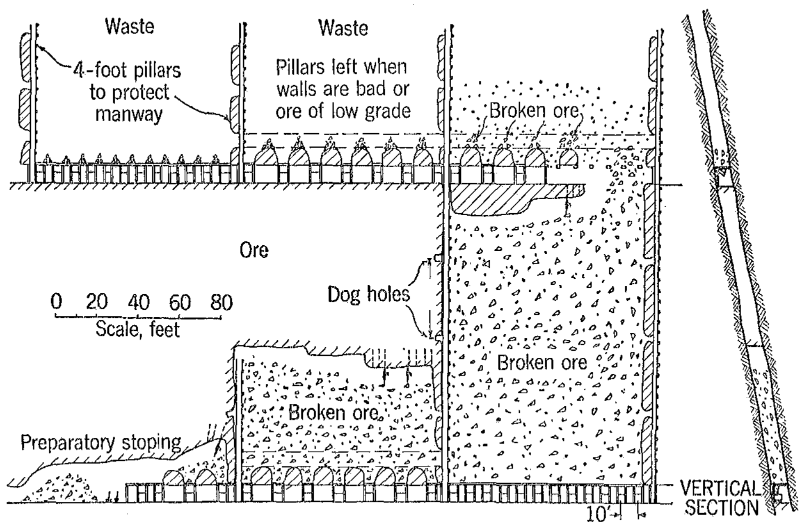


FIGURE 64.—Shrinkage stopes, Eighty-Five mine.

The method of stoping is shown in figure 65. A horizontal pillar, generally a little higher than the width of the vein, was left at the top of a stope to support the walls during final drawing of the broken ore. After the stope was filled the crown pillar was removed. Boulders were blasted at the blasting chamber. The chamber timbers, although badly worn, lasted the life of a stope.

KENNECOTT

The Kennecott mines of the Kennecott Copper Corporation are on Copper River at Kennecott, Alaska, 120 miles from Cordova on the sea. The principal mines are the Bonanza and Jumbo. The high-grade ore is shipped to the smelter at Tacoma, Wash.; the low-grade

[62] Dickson, Robert H., Mining Methods and Costs of Mining Copper Ore at the Verde Central Mines, Inc., Jerome, Ariz.; Inf. Circ. 6464, Bureau of Mines, 1931, 13 pp.

ore is treated in a concentrator and leaching plant at Kennecott. The mines are worked through adits and interior inclined shafts and are all connected underground. In 1931 mining of the main ore bodies was being completed, and the main production was derived from pulling pillars and other clean-up operations. The mines were closed in 1932 and reopened in 1935.

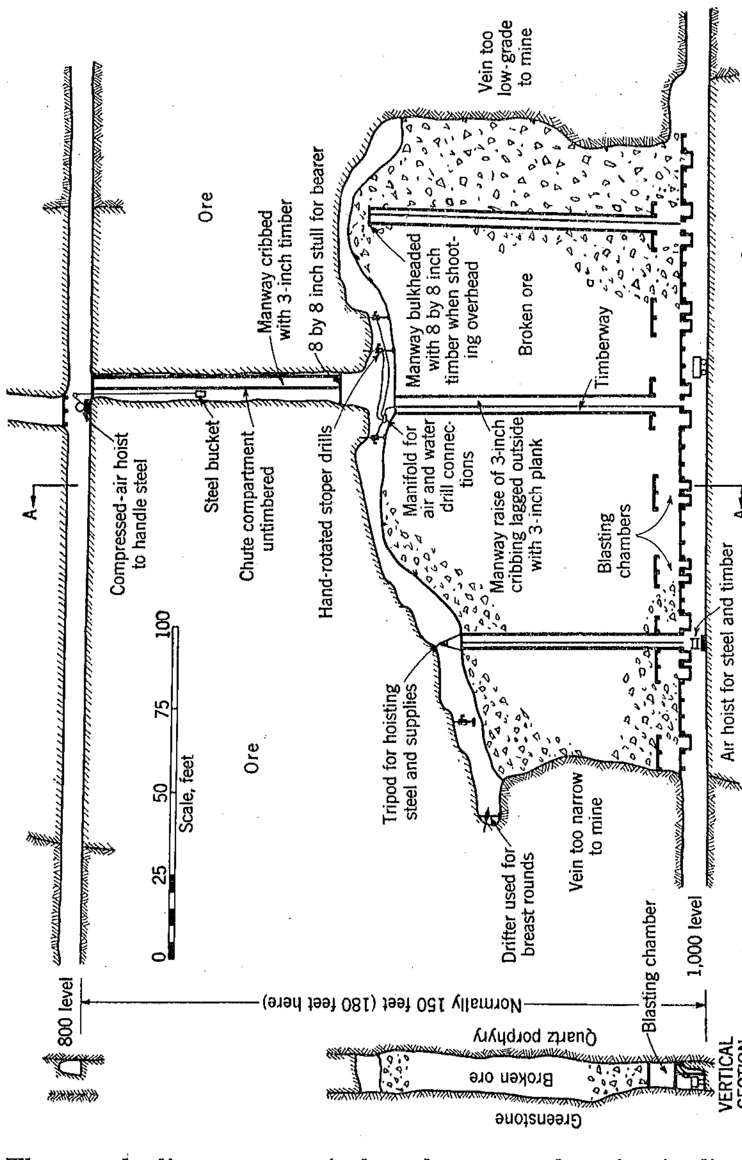


FIGURE 65.—Longitudinal section through stope, Verde Central mine.

The ore bodies were typical replacement deposits in limestone. Their width ranged from a few to 100 feet and their length from 150 to 1,000 feet. Some ore bodies contained both high- and low-grade ore. The dip was usually almost vertical but varied down to 48°.

Figure 66 shows the plan and projection of a typical ore body and the method of stoping.[63] Crosscuts were driven from an inclined shaft to an ore body. Drifts were run preferably in the high-grade ore. Chute raises were 25 to 30 feet apart. Where the ore would not run down the footwall, branch raises were extended to the stope. The stopes were silled 25 to 30 feet above the level.

Access to a stope was through empty chute raises, special manway raises in the country rock at one side of the stope, or raises from the level above; the last were usually connected through before the stope was advanced very far upward. Stoping limits depended solely on the grade of the ore; there were no hanging-wall or foot-wall contacts.

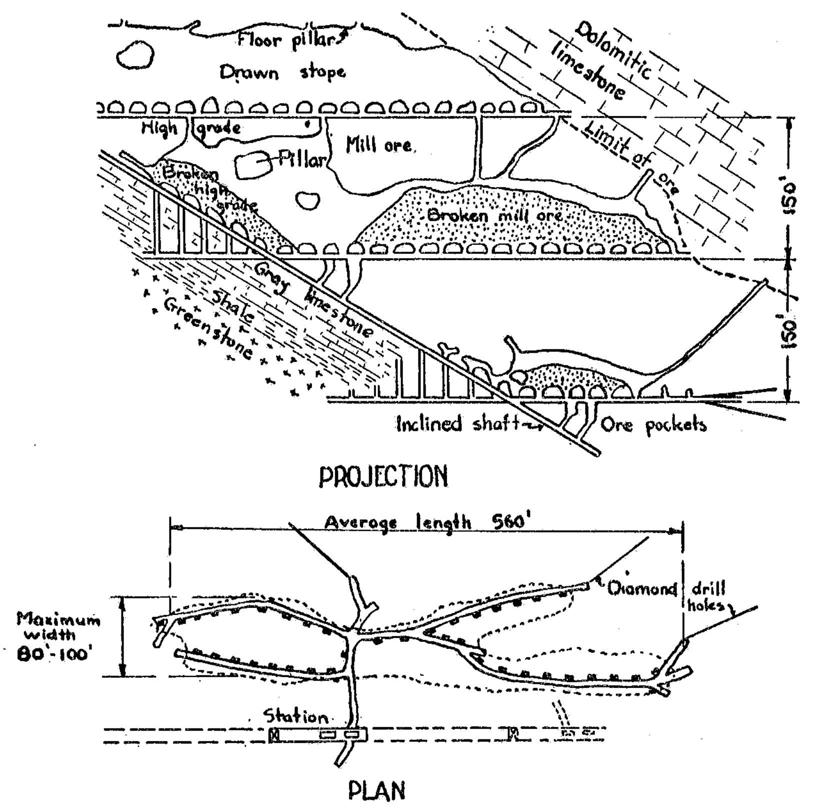


FIGURE 66.—Shrinkage stope at Kennecott.

The nature of the ore was such (chalcocite and copper oxide veinlets and impregnations in limestone) that the grade was readily determined by inspection. A high-grade section of the ore body that was of sufficient size was mined by shrinkage and completely drawn out. Concentrating ore was then stoped and the void left by the extraction of the high-grade ore filled.

[63] Birch, Stephen, Geology and Mining Methods of Kennecott Mines: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 499-511.

ISLE ROYALE

The Isle Royale mine, belonging to a subsidiary of the Calumet & Hecla Consolidated Mining Co., is at Houghton, Mich. The mine is opened for nearly 3 miles along the strike of the Isle Royale lode.[64] The average thickness of the ore is 9 feet and it dips 56°. The ore and walls are fairly strong and firm over certain areas, although in places cross fissuring and faulting weaken the walls. The judicious use of pillars, however, keeps dilution of the ore by wall waste at a minimum.

The mineralization of the lode is not uniform. Where possible, lean spots in the lode are left as pillars. Occasionally blocks as much as 100 feet long are too low-grade to mine. Selective mining is used where possible.

The mine is operated through three shafts sunk on the lode to an inclined depth of nearly 5,000 feet. To a depth of 2,500 feet stoping advanced from the shafts to the property boundaries. At this depth the cost of maintaining the haulage level became so great that a change was made to a retreating system.

A form of shrinkage stoping adapted to the retreating system is used. The drifts that form the haulage levels are driven to the boundaries, where stoping begins. After the face on one level is progressing toward the shaft, stoping is begun on the level below. Usually five or six levels are worked at one time. As the top level of the group is finished, stoping is started on a new one below.

To begin a shrinkage stope the drift is made as wide as the ore for a distance of about 100 feet. If the lode is over 12 feet wide, drift sets are placed on 6-foot centers and ore chutes are built in every third set; if not, stulls are used instead of sets.

The method of stoping is shown in figure 67.[65] The face is carried to the level above at an angle of 30° to 35°, except in heavy ground, where the face is steepened so that a given section can be mined out and abandoned more quickly. The irregularities of the ore body can be followed better and the drilling machines set up more easily on the flatter slopes. Only enough ore is drawn off at this stage to give proper working space for the miners. As virtually all the broken material must be hoisted, care is taken not to break rock that is below mining grade. Exploratory raises are run to determine the copper content of the ore before it is broken. The footwall is mined first; then if there is enough ore on the hanging wall it is also broken. Casual pillars of poor rock are left for support.

When stoping operations have advanced far enough ahead the ore remaining in the stope is drawn. That left between the chutes is dropped into the drift and loaded by hand. Sorting at the surface raises the grade from 0.8 or 0.9 percent to 1.1 percent of copper.

BRITANNIA

The mines of the Britannia Mining & Smelting Co., Ltd., are at Britannia Beach, British Columbia. Shrinkage or a combination shrinkage and glory-hole method are used in the Fairview and Bluff

[64] Wohlrab, A. H., Shrinkage Stoping in Amygdaloid Lodes: Min. Cong. Jour., October 1931, pp. 491–495.
[65] After Crane, W. R., Mining Methods and Practice in the Michigan Copper Mines: Bull. 306, Bureau of Mines, 1929, p. 63.

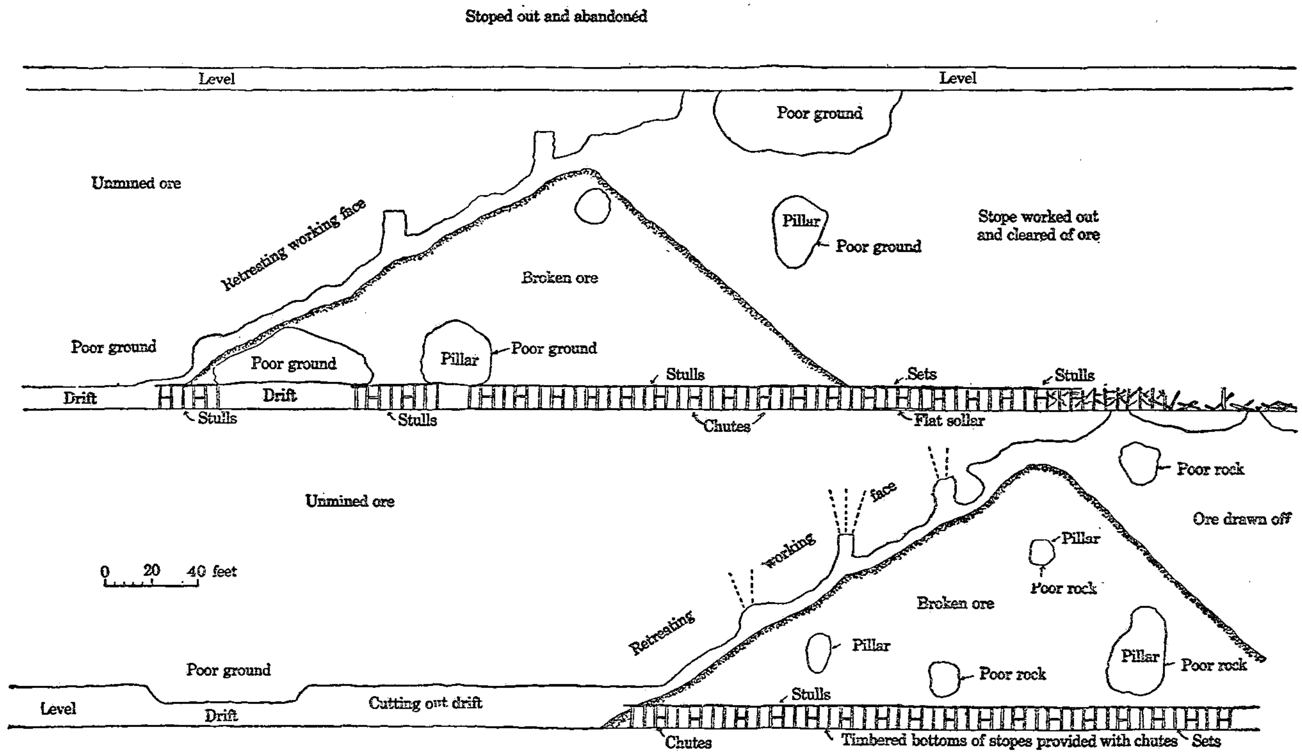


FIGURE 67.—Shrinkage stoping with retreating face, Isle Royale mine.

mines.[66] Shrinkage stopes are used in moderate-size ore bodies in the lower levels of the Fairview mine. The ground stands well over the 12 or 15 feet usually mined. A raise is driven to the level above before stoping begins; a pillar is left to support it. Chutes are 33 feet apart; a stope connects the chute raises 25 feet above the drift. After a flat back is formed, entrance to a stope is through the service raise. Stoping details are shown in table 28.

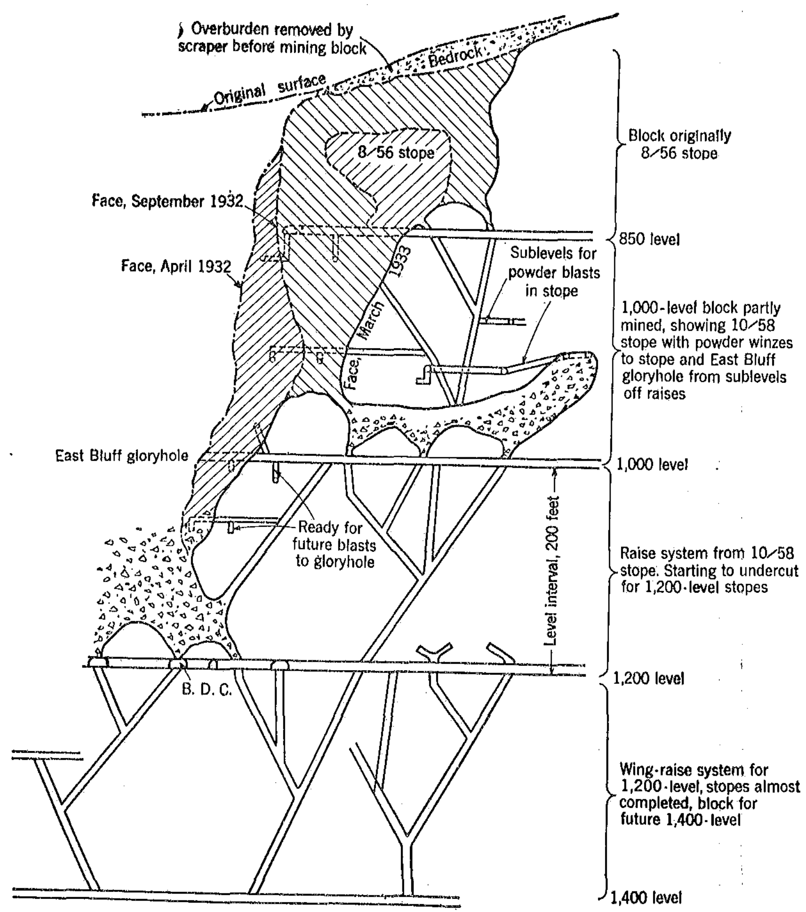


FIGURE 68.—Ideal section showing application of the Britannia method of stoping to East Bluff ore body.

The Britannia method, developed by C. V. Brennan, is employed in the wide ore bodies of the Bluff and Fairview mines. It is essentially a shrinkage method in which the ore is broken by powder drifts and was designed to meet the necessity for low costs and increased safety of operations. Figure 68 gives an ideal section showing the application of this method. The block of ore extended from below the 1,200-foot level to the surface at the 700-foot level and lay east of the old Bluff glory hole. Intermediate levels were 125 and 325 feet

[66] Brennan, C. V., Mining Operations at the Property of the Britannia Mining & Smelting Co., Britannia Beach, British Columbia: Inf. Circ. 6815, Bureau of Mines, 1935, 37 pp.

below the 700-foot level. On the east end 9,000 cubic yards of overburden were removed by a scraper and 75-horsepower hoist before underground stoping began. Stopes are laid out to conform with the size and shape of the ore body, with pillars between sections. The blocks vary in width up to 200 feet and are completely undercut. The block is usually undercut from foot wall to hanging wall by belling out raises from grizzly chambers as in large standard shrinkage stopes. One or more service raises, depending upon the size, are necessary for each block.

A grizzly chamber or an alternate level for drawing chutes is first established. In handling the ore from the large-scale stoping operations, grizzly chambers have been indispensable. Here most of the secondary breaking is done. The chambers are usually laid out on 70-foot centers, and the first row is placed as near the footwall as conditions will permit. The raise representing the throat of the chamber points up the footwall and permits drawing in that direction. (See fig. 68.) Each grizzly bar consists of three sections of English rails bolted together as shown in figure 69. The spacing depends upon the character of the rock and the length of drop from the grizzly to the transfer chute. East Bluff rock, which shows little tendency

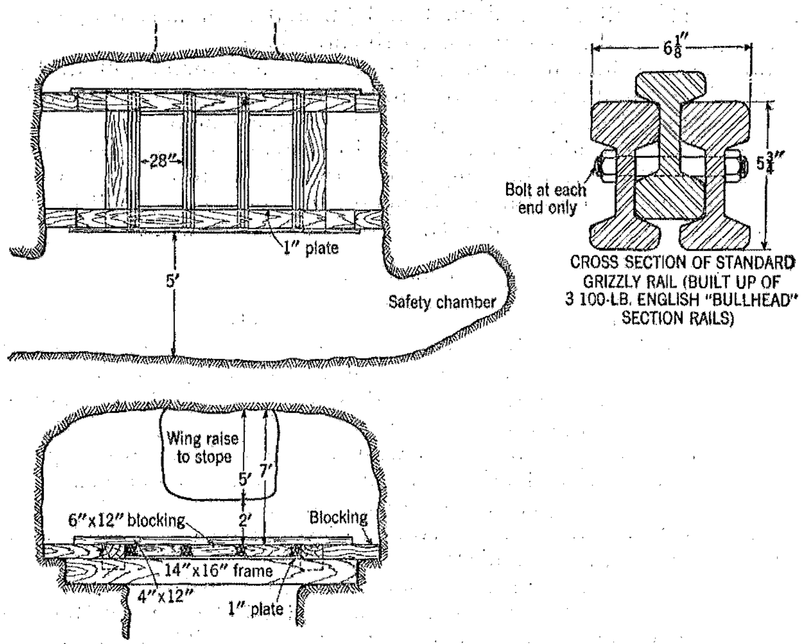


FIGURE 69.—Plan and section of standard bulldoze chamber, Britannia mine.

to shatter in its fall down the raise, is passed through a 26-inch grizzly, whereas the more-sheared and softer ore of the Fairview section, which drops 500 feet, is passed through openings up to 36 inches wide. The ore goes through branch raises to the haulage level.

Powder drifts are run from the service raises. The first lift generally is 25 or 30 feet above the undercut. It is customary to open this sublevel by a drift on the footwall and define the position of the hanging wall by short diamond-drill holes. The powder pockets are in crosscuts from the footwall drift and are placed to give the most effective results. They are excavated by sinking a few feet in the bottom of the crosscuts. The ore broken in the last two or three rounds is saved for stemming.

Each lift is blasted down in stages, retreating lengthwise of the stope toward the service raise. Work proceeds on several lifts at the same time, which gives a stope a rilled appearance. The quantity of explosive needed for a blast is important; it is based on judgment and experience.

In East Bluff ground, where the walls stand well, no control of drawing is necessary, but in the softer ground just enough ore is drawn to make room for the next blast. Large blocks of ore in the throat raises are drilled with stopers, and boulders on the grizzlies are drilled with pluggers. A chamber crew comprises one miner and a mucker; they wear safety belts. The safety chamber permits a safe retreat of the inside man if a run of ore occurs. Forty-percent-strength gelatin explosive is used for all blasting work in the chambers. In the East Bluff ore body one-half pound of explosive is used at the grizzlies per ton of ore mined. An average of 0.16 man-hour per ton is required to get the ore through the grizzly compared with 0.11 man-hour for stoping. The pillars are drilled and blasted after active operations are completed and are drawn with the stope ore.

The method has flexibility, leaves no permanent pillars to be extracted by auxiliary methods, and provides for a constant retreat from worked-out areas. It requires close supervision.

In the hard ground of the East Bluff losses of ore have not exceeded 10 percent. The softer hanging wall of the West Bluff, however, caves readily and may dilute the ore to such an extent that it will be below commercial grade before an entire stope can be drawn.

CREIGHTON

The Creighton mine of the International Nickel Co. of Canada, Ltd., is in the Sudbury district of Ontario. The mine has been operated for over 30 years. The ore was mined to a depth of 300 feet by open-pit methods. Below this depth open-stope methods were first used, then shrinkage stoping was adopted. Owing to increasing irregularities in the deposit, weak wall rock, and the need for space to dispose of waste, shrinkage methods will not be economical in the future and will be replaced by cut-and-fill and square-set methods. So far (1930), however, only a comparatively small tonnage has been mined by these two methods.[67]

The main ore body is a contact massive pyrrhotite extending from the surface to level 20; a second ore body extends from level 16 to level 28 and a third from level 18 to and below level 50. The ratio of copper to nickel in the ore ranges from 1: 3.5 to 3.5 : 2.

The mine is worked through four inclined shafts. Main haulage drifts are run parallel to the ore body. (See fig. 70.) Where the width of the ore warrants, crosscuts are turned off on 75-foot centers. Shrinkage stopes 50 feet wide and as much as 250 feet long are laid off across the ore body, with 25-foot pillars between them. The crosscuts are on the center lines of the pillars. The manner of laying off the stopes and the method of stoping are shown in figure 70,

[67] Parker, Ralph D., Mining at Creighton; Eng. and Min. Jour., vol. 130, Nov. 10, 1930, pp. 437-443.

Chute raises are run every 15 feet on alternate sides of the crosscuts. Footwall raises also are run on the center lines of the pillars at the contact to explore the footwall and serve as manways. Ore passes are driven in the footwall on the pillar lines. Branches from the main ore passes are taken off at such intervals that chutes from the branches when connected with the stopes will not have a greater horizontal span than 30 feet.

A series of 8- by 10-foot raises is driven in the footwall between all main haulage drifts, at a minimum angle of 65°. When back stoping has been completed in two adjoining stopes, the intervening pillar is drilled from drifts and raises and blasted electrically.

LEVACK

The Levack mine of the International Nickel Co. of Canada, Ltd., is the only operating mine on the north range of the Sudbury district, Ontario. It was the main producer of the Mond Nickel Co. before it was taken over by the International Nickel Co. The ore is shipped to the Coniston smelter.

Stopes 100 feet wide with 40-foot pillars between are laid out across the ore body.[68] The present practice is to leave a 10- or 12-foot rib pillar midway between the main pillars. (See fig. 71.) This prevents scaling of the back. The rib pillar is cut off at the hanging wall and footwall as mining progresses upward and is drawn when the stope is emptied.

Preparatory to stoping, a drift is run through the long axis of the ore body. Two parallel crosscuts 50 feet apart are run from the drift under each stope section from foot wall to hanging wall. Chute raises are put up at 16-foot intervals on alternate sides of the crosscuts to the stope floor, which is 25 feet above the level. Next the stope is silled by making a cut 7 feet high; as this work progresses the chute raises are funneled out. Where ground conditions permit, an 8-foot section is taken out up the footwall, and all cuts are started from this opening. All stopes are entered from a small manway at the footwall. In stopes in which no rib pillar has been left half of a breast (50 feet) is drilled and blasted at a time. The breast between the rib pillar and a main pillar is blasted as one round. About 80 tons are broken for each machine-drill shift; about 250 men underground produce 2,500 tons daily. Cuts or benches are usually 18 feet high. Rounds comprise 12-foot horizontal holes. Where the footwall flattens so that the ore will not run to the chutes in the main crosscuts, chutes are run from subdrifts or branch raises.

GARSON

The Garson mine of the International Nickel Co. of Canada, Ltd., is in Garson Township, Ontario. The length of the ore zone ranges from a few feet to 700 feet and the width from a few feet to 100 feet. The ore bodies are irregular in size and shape. All the ore is mined by shrinkage. The backs are strong, and areas 100 feet wide by several hundred feet long will stand open. Haulage drifts are run under the ore bodies. Ore on flat footwalls is pulled down when the stopes are emptied.[69]

[68] Sharpe, A. L., Levack Mine Practice: Eng. and Min. Jour., vol. 130, Nov. 10, 1930, pp. 452-453.
[69] Hall, Oliver, Mines and Mining Operations (International Nickel Co. of Canada, Ltd.): Eng. and Min. Jour., vol. 130, Nov. 10, 1930, pp. 435-437.

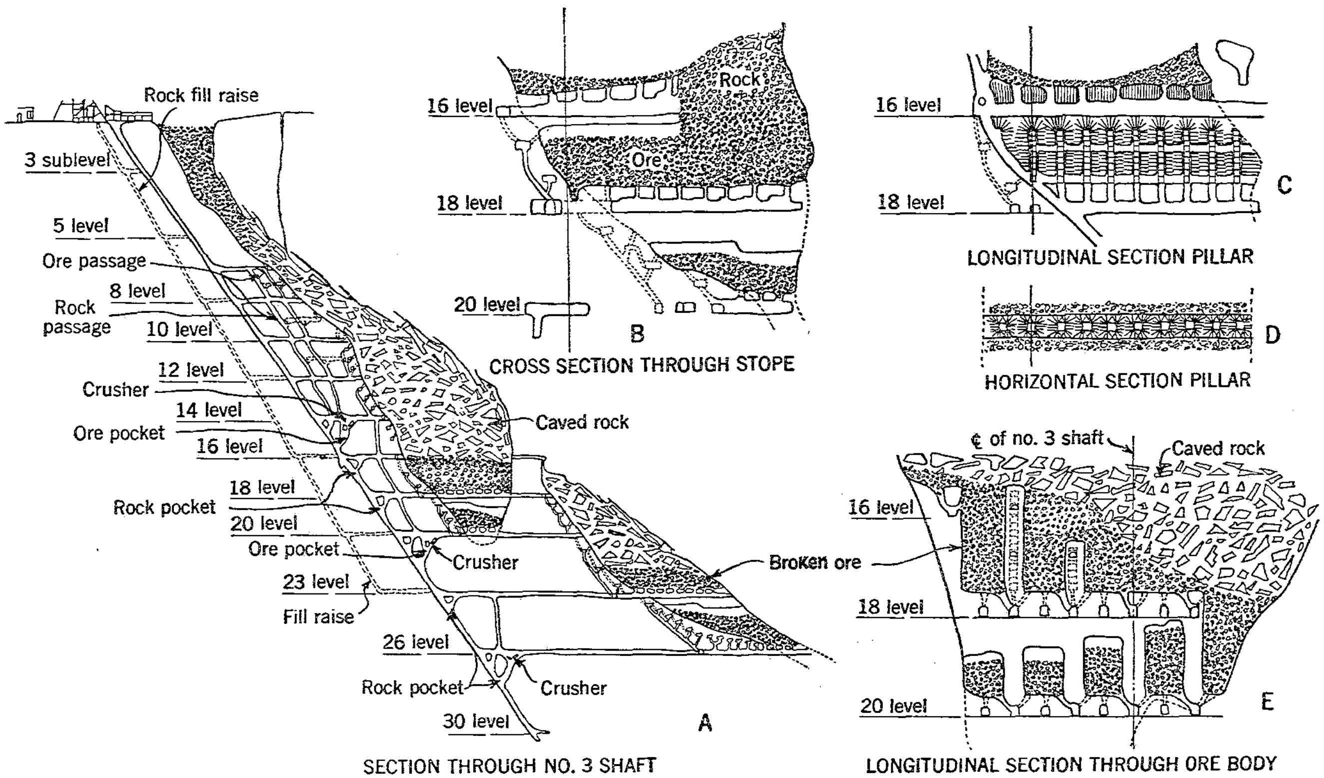


FIGURE 70.—Shrinkage stopes, Creighton mine.

BEATSON

The Beatson mine of the Kennecott Copper Corporation is on tidewater at Latouche, Alaska. Rich ore was shipped directly to the smelter at Tacoma, Wash. Concentrating ore was treated in a 1,500-ton plant at the mine. The ore occurred in a shear zone. The largest ore body was roughly lenticular, 800 feet long, and a maximum of 340 feet wide. The wall rock and ore were hard and stood well.

A glory-hole method of mining was used first at this mine, but after 1923 virtually all the ore was mined by the so-called Latouche system.[70] This is classified here as a form of shrinkage stoping, because the broken ore was used to support a weak hanging wall and only the necessary amount of "swell" was drawn during mining.

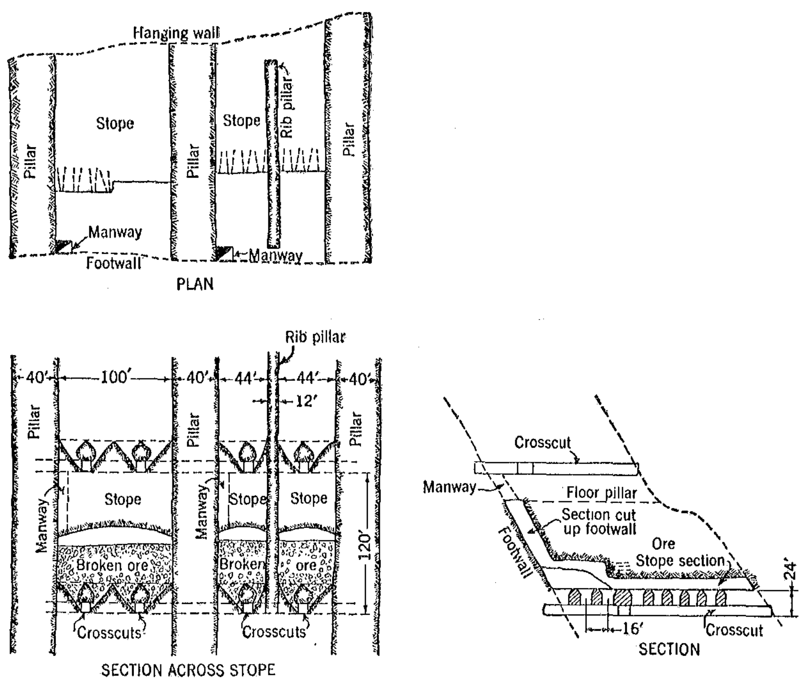


FIGURE 71.—Shrinkage stoping, Levack mine.

The mine was worked through an adit 970 feet long and an interior three-compartment shaft 300 feet deep. Levels were spaced at 100-foot intervals. The stopes were 70 feet wide and alternated with 30-foot pillars along the strike; the shortest stope between foot wall and hanging wall was 70 feet long and the longest 270 feet. The stopes were prepared for mining much as in the undercut block-caving method, that is, by running branch gathering raises and undercutting; however, the ore was too strong and hard to cave of its own weight and had to be broken down by explosives. The mining level was at the top of the block being mined. After the ore was broken it was

[70] Presley, Be Van, The Latouche System of Mining as Developed at the Beatson Mine, Kennecott Copper Corporation, Latouche, Alaska: Trans. Am. Inst. Min. and Met. Eng., vol. 76, 1928, pp. 11–52.

drawn through grizzlies on the grizzly level, 150 feet below the mining level. From the grizzlies the ore dropped through branch raises to the haulage level. The grizzly chambers were 24 feet long, 8 feet wide, and 7½ feet high. Two lines of grizzlies were used in the stopes, and one line was used in the pillars.

The stopes were undercut by widening inclined connecting raises from adjoining grizzly chambers until they met similar raises from the next row of chambers. On the pillar side the widening was stopped at the pillar line. The undercutting raises were branched to form protecting pillars over the grizzly chambers.

After the undercutting raises were completed and before the stope was undercut two lines of mining raises were run 15 feet into each stope section parallel to the pillar lines. A line of raises was also run on the center line of the pillars. These raises were parallel to the dip of the ore body and extended to the mining level above at 40- to 60-foot intervals. Stoping was started by cutting a bench around the raise nearest the footwall, about 28 feet above the undercut. From this bench rounds consisting of holes 30 to 40 feet deep were then drilled to break down the back of the stope. The same was done in the next raise and so on across the stope; the process was repeated at a point higher up until the surface was reached. The pillars were mined in the same way after the stopes on both sides were completed.

The ore bodies were exhausted and the mine was closed down at the end of 1930.

RAY

Shrinkage stoping and the caving of pillars is used (1929) at Ray to mine relatively high-grade small bodies of ore on the fringe which are too high above the motor levels or are not thick enough to warrant the development necessary for the use of a straight caving method.

Six or eight laterals are driven through a block on 25-foot centers. Shrinkage stopes on 25-foot centers and at right angles to the laterals are then carried up to the capping.[71] These stopes are made as wide as the ground will permit to keep the width of the pillars to a minimum. In starting the shrinkage stopes, raises are put up on a 45° slope from both sides of the laterals until they intercept to form the starting cut of the stopes. As soon as the shrinkage stopes are completed the pillars between them are undercut. The final undercut round in a pillar breaks into the shrinkage stopes on either side. The ore from the stopes is drawn into small cars and trammed by hand to transfer raises whence it is loaded into motor trains on the haulage level. The method is locally termed the "hand-tramming method" and was the predecessor of the block-caving method described on page 152.

HORNET

The Hornet mine of the Mountain Copper Co., Ltd., is at Mathewson, Shasta County, Calif. This company has been one of the most important producers of copper in California. During recent years, however, the production of copper has dropped. The Hornet ore body of massive pyrite is about 200 feet in width, is of undetermined length, and has a dip of 60°. It occurs in a shear zone in rhyolite. The ore is mined both for its copper and sulphur; it contains 0.7 percent copper and 45 to 48 percent sulphur.

[71] Thomas, Robert W., Mining Practice at Ray Mines, Nevada Consolidated Copper Co, Ray, Ariz.: Inf. Circ. 6167, Bureau of Mines, 1929, 27 pp.

The ore is conveyed by aerial tramway about 2½ miles from the mine to a concentrator, where it is treated prior to being shipped to an acid plant. Most of the waste rejected at the mill is wall rock which is caved with the ore.

The copper is leached from the calcines after the ore is used for making sulphuric acid. Normal production is about 400 tons per day. The mine is worked through adits.

The method of mining is an adaptation of shrinkage stoping; part of the ore in the back of the stopes is broken by blasting and the remainder caves. The method closely resembles undercut block caving as used in the oxide ore body at Miami (see p. 161); however, no attempt is made at the Hornet mine to guide the caving along predetermined boundaries. The method has been called powder-drift stoping or induced caving. The ore body is laid off along the strike of the vein in 80- by 150-foot stopes with 30-foot pillars between.

The first haulageway is driven lengthwise in the ore body near or on the footwall.[72] Manway raises 180 feet apart are then carried up the footwall in the pillars and connected by a drift 25 feet above the haulage level. Next, ore passes are put up on 25-foot centers from the haulage drift to the level of the grizzly floor on a line 30 feet to the side of the haulage drift, and 15-foot crosscuts are driven from the grizzly level drift midway between each pair of ore passes; from the end of each crosscut two branch entries are driven at an angle of about 30° to intercept the top of the ore-pass raises, where grizzly chambers are cut. A runway is cut on each side of an ore pass to protect the men working in the chamber. The crosscuts serve as entries to the grizzlies.

The top of each ore-pass raise is decked with grizzlies made of 90-pound rails. The grizzlies have a 20-inch elevation in the rear which causes the oversize to slide well to the front. This inclination reduces the blocking of the grizzlies by boulders to about half of that when the grizzlies are horizontal. A notable feature of the blasting chambers is the use of the Y connections, as a result of which pipe and light lines in the main grizzly drift are not exposed to the blasting. From the rear of the grizzlies (which are 30 feet from the footwall) bell-shaped mill holes are extended upward into the ore. These are on 25-foot centers parallel to the footwall.

Stoping is started by connecting the row of belled-out mill holes. A stope 80 feet wide and 150 feet long is then undercut by room-and-pillar mining; the rooms are carried up a slope in both directions from the mill holes and across the stope. The width of the pillars depends upon the physical condition of the ground. After the rooms are finished the pillars are drilled and blasted, beginning at one end of the stope. This leaves what would appear to be an ordinary 80- by 150-foot shrinkage stope with the back about 25 feet above the bell-shaped openings.

The next step is to drive powder drifts from one or both of the manways in the pillars at the ends of the stope at a height of 20 or 30 feet above the back, depending upon the nature of the ground to be broken. These drifts are about 55 feet long from the manway or 40 feet from the edge of the pillar. At the end of the drifts two 10- or 15-foot crosscuts are driven to form a tee. The stope below is then drawn down to make space enough for the ore to be obtained from the powder-drift

[72] Data furnished by M. J. Murphy, mine superintendent, Mar. 1, 1930.

blast. The crosscuts at the end of the drift are loaded with equal amounts of 40-percent gelatin dynamite (100 to 135 50-pound cases). The explosive is taken from the boxes, and each pile is connected by three rows of cartridges placed side by side with the joints broken. Two detonators of 18-foot capped fuse are placed in each pile, and the charges are stemmed with the broken ore from the last round in each crosscut. The usual procedure is to blast alternate powder drifts on opposite ends of the stope. Powerful lights are used to inspect the condition of the stope after each blast, and the alternate drifts are placed to give the best results. From 20,000 to 30,000 tons usually are obtained from each of these blasts. Each foot in height of the stope corresponds to about 1,000 tons of ore, therefore about as much additional ore caves as is broken by a blast. In the first stope, caving induced by the first blast is carried up 200 feet to near the top of the ore body. Full advantage is taken of all faults and structural features in breaking the ore.

Two men work at each grizzly; considerable secondary blasting is necessary.

Good ventilation at the grizzlies is provided so that the men can return directly to this work after block-holing or bulldozing charges are fired.

No timber is used in the stopes which reduces the fire hazard. After the ore is broken it is kept in motion, and seldom more than 40,000 tons is in a stope at a time; no fires have resulted from the caving.

The main advantage of powder-drift shrinkage over straight shrinkage stoping is the large reduction in primary drilling and blasting costs and the safer working conditions. The drillers are not exposed to falls of rock from a large back or to being carried down when the shrinkage stope is drawn. No barring down or propping up of dangerous looking slabs or rigging to reach the back for drilling is required. Ventilation or pipe lines are not needed in the stope proper. The disadvantages are that selective mining or stope control is out of the question and dilution is unavoidable.

CUT-AND-FILL STOPING

In a cut-and-fill method of stoping, relatively thin slices are taken successively from the lower surface of a block of ore; after the broken ore from one slice is removed the space it occupied is filled with waste material before the next slice is mined. Casual support by stulls or cribs may be used to hold up weak sections of the back until they are ready to be blasted.

The method is used under conditions intermediate between those suitable to shrinkage stoping on the one hand and square-setting on the other. Through variations the cut-and-fill method merges into shrinkage stoping, open stoping, or square-setting. The method is flexible in that if heavy ground is encountered a change can be made readily to square-setting. Should the walls of stopes prove stronger than expected a number of cuts may be taken before filling, which would be in effect a semishrinkage method.

Where the cut-and-fill method is applied to wide ore bodies it is common practice to mine regular sections across the ore body between pillars, which are later stoped by square-setting or top slicing.

Cut-and-fill stoping can be divided into two general classes—the horizontal cut-and-fill and the inclined cut-and-fill, or rill, method.[75] In the former the back and filling are kept practically horizontal. In the latter the back and filling are maintained parallel to each other and usually at the angle of repose of the waste material used for filling.

The principal copper mines in North America using the cut-and-fill method of mining, together with production data, are shown in table 29. The character of the deposits and stoping data for these mines are shown in table 30. Brief descriptions of the methods used at these mines follow.

[75] Johnson, C. H., and Gardner, E. D., Cut-and-fill Stoping: Inf. Circ. 6688, Bureau of Mines, 1933, 59 pp.

TABLE 29.—*Production data for representative North American copper mines using cut-and-fill method of mining*

	Pilares	Matahambre	Champion	United Verde (underground)	Frood	Campbell	Colorada	Magma
Location	Pilares, Sonora	Matahambre, Cuba	Painesdale, Mich.	Jerome, Ariz.	Sudbury, Ontario	Lowell, Ariz.	Cananea, Sonora	Superior, Ariz.
Normal daily production of ore (tons):								
Total	3,000	1,300	1,500	3,000	[1] 3,000	900	1,500	830
By cut and fill	2,300	900	1,500	2,000	[2] 2,200	[3] 900	600	330
Total yearly production of ore (tons):								
1929	835,000	362,000	447,000	[2] 800,000	200,000	[4] 578,000	[5] 525,000	270,000
1930	646,000	369,000	[2] 450,000	[2] 350,000	903,000	[4] 415,000	----	252,000
1931	[2] 600,000	330,000	[6] 405,000	(7)	1,069,000	(7)	----	232,000
1932	0	145,000	291,000	(7)	514,000	(7)	----	145,000
1933	0	219,000	204,000	(7)	953,000	(7)	----	145,000
1934	0	152,000	241,000	(7)	1,868,000	(7)	----	264,000
Total yearly production of copper (pounds):								
1929	42,070,000	31,500,000	20,661,000	[2] 72,000,000	[8] 88,880,000	[4] 55,586,000	[8] 58,827,000	36,517,000
1930	28,704,000	34,109,000	20,000,000	[2] 30,000,000	[8] 125,328,000	[4] 41,345,000	[8] 42,425,000	31,559,000
1931	26,994,000	29,778,000	[6] 17,721,000	(7)	[8] 110,740,000	(7)	[8] 41,873,000	28,761,000
1932	0	13,066,000	12,189,000	(7)	[8] 74,620,000	(7)	[8] 36,820,000	22,052,000
1933	0	19,746,000	12,167,000	(7)	[8] 141,500,000	(7)	[8] 51,793,000	19,628,000
1934	0	13,650,000	13,930,000	(7)	[8] 200,410,000	(7)	[8] 60,431,000	31,647,000
Total production of ore to end of 1930 tons	14,650,000	----	16,338,267	----	[8] 26,700,000	[9] 12,900,000	[8] 22,000,000	2,380,000
Total production of ore to end of 1934 tons	15,250,000	----	17,479,000	----	[8] 32,974,000	(7)	----	3,166,000
Total production of copper to end of 1930 pounds	759,940,000	----	532,384,832	[10] 2,050,000,000	[8] 970,000,000	[9] 1,250,000,000	[8] 1,106,318,000	282,000,000
Total production of copper to end of 1934 pounds	786,934,000	----	588,392,000	----	[8] 1,497,270,000	(7)	[8] 1,297,235,000	384,000,000
Kind of deposit	Tabular	Lenses	Bedded	Massive sulphides	Deposits in shear zone	Massive sulphides	Pipe in quartz porphyry	Vein in diabase
Length of ore body feet	Irregular	Up to 275	8,000	1,100	5,000	500	300	1,200-1,500
Width of ore body do	Irregular	Up to 30	[11] 17	300	40-200	50-250	200	[11] 25
Grade of ore percent of copper	3	4.5	[12] 2.25	5	4.5	[4] 5	4	7
Dip of ore body degrees	90	42-45	70	60	65	25-90	90	45-80
Character of walls	Weak	Hanging, weak	Hanging, weak	Weak to strong	Strong	Strong	Strong	Weak and crushed
Weight of ore cubic feet per ton	12	1,011	11	8 — 10	9	9	8 — 12	10½

[1] In 1930; mine being equipped to produce 8,000 tons daily.
[2] Estimate.
[3] No pillars mined in 1929 or 1930.
[4] Total, Calumet and Arizona mine.
[5] Estimated; total all mines 895,000 tons.
[6] Includes Baltic production.
[7] Production not published.
[8] All mines of company.
[9] Total Calumet and Arizona mine to end of 1930.
[10] Estimated total, underground and open-cut mines.
[11] Average.
[12] Recovery.

TABLE 30.—*Stoping data at representative North American copper mines using cut-and-fill method of mining*

	Pilares	Matahambre	Champion	United Verde (underground)	Frood	Campbell	Colorada	Magma	Anaconda
Location	Pilares, Sonora	Matahambre, Cuba	Painesdale, Mich.	Jerome, Ariz.	Sudbury, Ontario	Lowell, Ariz.	Cananea, Sonora	Superior, Ariz.	Butte, Mont.
Depth of ore mined (feet)	1,900	1,800	4,000	3,000	3,000	2,300	1,500	3,000	4,000.
Level interval (feet)	100, 133	100, 130, 150	94[1]	150	200	100	125	125, 150[2]	100, 200.
Spacing of raises to level above (feet).	30 or 50	100, 150	200	1 or 2 to a stope	1 or 2 to a stope	45 to 50	40 by 75	1 to a section	112.
Spacing of chutes under stopes (feet).	30	50	200	16½ or 22	22 or 27½	10	40	do	112.
Width of sections (feet)	Width of ore body	Width of ore body	Width of lode	30 to 160	45	45 to 50	30	16	Width of ore body.
Length of sections (feet)	Length of ore body.	Length of ore body.		60 to 200	Width of ore body.	Width of ore body.	50 to 160	15 to 60	56.
Where stopes silled	At level or 20 feet above.	14 feet above level.	At level	21 to 25 feet above level.	30 feet above level.	10 feet above level.	11 or 22 feet above level.	At level	At level.
Inclined or horizontal cuts.	Both	Horizontal	Both	Horizontal	Horizontal	Inclined	Horizontal	Inclined	Inclined.
Height of cuts (feet)	8	6 to 7	3½ to 4	7	7	12	11	7	8.
Kind of drills used in Stopes.	Drifters	Jackhammers and some stopers.	Drifters	Drifters	Drifters	Drifters	Drifters	Stopers	Stopers.
Method of handling ore in stopes.	Shoveling and gravity.	Shoveling and wheelbarrows.	Gravity and shoveling.	Shoveling	Shoveling	Gravity	Shoveling and scrapers.	Gravity	Gravity and shoveling.
Grizzly openings (inches).	12		0	11		10	10	9	
Proportion of total material broken sorted out in stopes.	Some	Some	40 percent	2 percent		0	0	0	Some.
Method of handling filling in stopes.	Rills, gravity, horizontal, cars, and scrapers.	Hydraulic (sand), shoveling, and scrapers.	Gravity	Cars	Cars	Gravity	Shoveling and scrapers.	Gravity	Gravity.
Sources of filling	Surface development and old fills.	Mill tailing and development.	Sorting and development.	Surface and development.	Surface	Development and surface.	Surface (¾) and development (¼).	Surface and development.	Sorting and development.
Kind of supports in stopes.	Stulls and cribs		Casual pillars	Stulls and cribs.	Cribs	Stulls and cribs.	Stulls and cribs.	Stulls	Stulls.
Spacing of supports	Irregular			Irregular	17-foot centers	Casual	Casual	Casual	Casual.

[1] 100 feet on dip.

[2] Alternate levels used for haulage.

NACOZARI

The Pilares mine of the Moctezuma Copper Co., a subsidiary of the Phelps Dodge Corporation, is at Pilares, in the Nacozari district, Sonora, Mexico. All ore mined is treated in a concentrator at Nacozari, and the concentrates are shipped to Douglas, Ariz., for smelting. The mine is worked through adits and two vertical shafts.[73]

The ore bodies occur around the circumference and within the boundaries of a pipelike mass of brecciated material having nearly vertical walls. The oval is roughly 2,000 feet long and 1,000 feet wide. The ore bodies around the wall are tabular; those within the core are irregular in shape. Usually the walls are weak, and the ore is hard and mixed with waste. The horizontal cut-and-fill method has been found best suited to these conditions. Where the walls are good, the ore body is extensive enough, and the ore is relatively clean the inclined cut-and-fill method is used. Square-setting supplemented by top slicing is used to mine pillars.

Most of the horizontal cut-and-fill stopes are silled on the level. An 8-foot cut is first mined to the limits of the ore or stope section; the back is then cut to a height of 14 feet above the floor. While this work is in progress, fill raises are driven at 30- to 50-foot intervals along the stope. A timbered gangway is then built, and the sill floor (except the gangway) is filled with waste. Cribbed chutes and manways are built up through the fill on 30-foot centers. Starting at the weakest end of the stope 8-foot horizontal cuts are drilled with mounted machines. The broken ore is shoveled by hand into the chutes back of the drillers. Where possible, temporary chutes are placed under the fill raises, and the waste is drawn into cement buggies for distribution in the stope. In large stopes scrapers are used successfully for distributing the fill.

Only the higher-grade stopes are floored, as the loss of fines in the lower-grade ore and the disadvantage in shoveling on the filling do not warrant the additional expense. Umbrella stulls and bulkheads are used for supporting weak backs. The timber is salvaged for reuse.

Where conditions permit, stopes are silled 20 feet above the level; this eliminates the gangway timbering. Chute raises are driven from the drift below at 25-foot intervals to the stope. Figure 72 shows the general plan of a stope lay-out.

MATAHAMBRE

At the Matahambre mine, on the north coast of Cuba, 70 percent of the ore is stoped by horizontal cut-and-fill and 30 percent by square-setting.

Preparatory to mining by the cut-and-fill method a drift is run on the ore for the length of the ore body. Chute raises are started on 50-foot intervals and a manway is taken up alongside every third chute.[74] After the chute raises have been carried up about 20 feet the ore is silled 14 feet above the floor of the drift. If the ore is wide, crosscuts are driven from the drift below to allow more raises to be brought up to the stope for ore chutes. Figure 73 shows the method of mining.

[73] Leland, Everard, Mining Methods and Costs at the Pilares Mine, Pilares de Nacozari, Sonora, Mexico: Inf. Circ. 6307, Bureau of Mines, 1930, 34 pp.
[74] Richert, G. L., Mining Methods at Minas de Matahambre, Matahambre, Pinar del Rio, Cuba: Inf. Circ. 6145, Bureau of Mines, 1929, 18 pp.

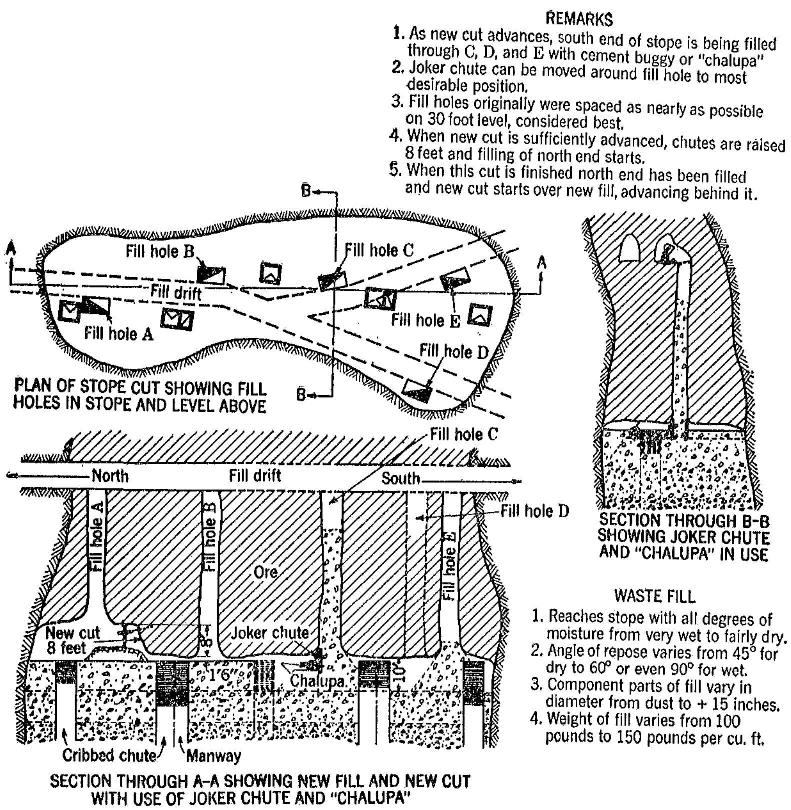


FIGURE 72.—Typical cut-and-fill stope, Pilares mine.

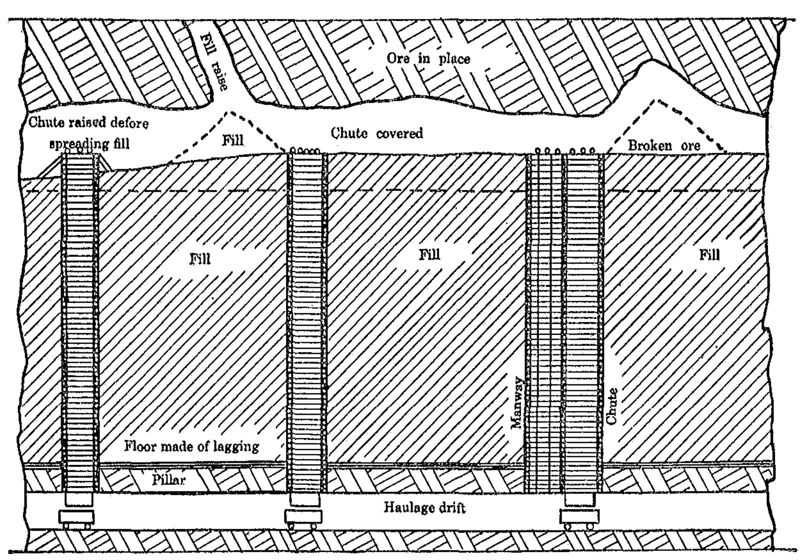


FIGURE 73.—Longitudinal section through cut-and-fill stope, Matahambre mine.

Filling comprises development waste and deslimed tailing from the concentrator in about equal proportions. When waste is used it is spread by wheelbarrows or scrapers.

CHAMPION

The Champion mine of the Copper Range Co. is at Painesdale, Mich., on the Baltic lode. The Champion lode, of which 8,000 feet is on Champion property, is here very straight and has a dip of 70° or slightly less. Its thickness ranges from 10 to 80 feet, and the average mining width in 1931 was 17 feet.[75] The mine is worked through three 70° inclined shafts. Haulage levels are driven 100 feet apart in the lode.

As depth was attained and drift maintenance costs became excessive the method of mining was changed from an advancing horizontal cut-and-fill to a retreating inclined cut-and-fill system. Sublevels used with the retreating system provide three stoping faces to each level. (See fig. 74.) Horizontal cut-and-fill stoping is still being used on the upper levels, but the main production is obtained by the inclined cut-and-fill method.

In the retreating system the levels are run to the boundary of the section of the lode to be mined. Two subdrifts between levels are driven the full width of the copper-bearing section of the lode from raises about 200 feet apart. The broken rock is pulled into the raises by scrapers. A raise is put up at the boundary to the level above, and stoping starts at this raise on the upper sublevel, followed at intervals of 45 feet or more by stoping on the lower sublevel and the main level. The faces are carried at an angle of 38°, so as to parallel the angle of repose of the waste fill. (See fig. 74.)

Rounds 10 feet deep are drilled parallel to the angle of repose of the fill and blasted against the fill. The ore is sorted from the waste at the toe of the slope, shoveled into a small car, and trammed to the nearest raise, or it is scraped to the raise directly from the face. About 40 percent of the material broken is rejected and used for fill. Each cut is 3½ to 4 feet thick; a relatively small burden is placed on the holes to facilitate sorting. The pillar left at the top of the slice is drilled but not blasted until all the ore on the slope has been picked out and all necessary exploration has been done on the footwall and hanging-wall sides. Loose blocks of ground are supported by stulls.

UNITED VERDE

The United Verde underground mine at Jerome, Ariz., in 1929 mined 64 percent of the 3,000 tons produced daily by cut-and-fill methods.[76] A drift is first run in the schist at the sulphide contact; from this opening the ore body is diamond-drilled and the stope lay-out planned. Stopes are usually 30 to 160 feet wide and 60 to 200 feet long, with 30- to 40-foot pillars between. (See fig. 75.) If the ore body is wide a second extraction drift is run in the sulphide. The next step is to drive a 6- by 11-foot raise, which later serves as a waste raise, to the

[75] Mendelsohn, Albert, Mining Methods and Costs at the Champion Copper Mine, Painesdale, Mich.: Inf. Circ. 6515, Bureau of Mines, 1931, 16 pp. Mendelsohn, Albert, and Jackson, Chas. F., the Sublevel Inclined Cut-and-fill Stoping System: Trans. Am. Inst. Min. and Met. Eng., vol. 102, 1932, pp. 43-58.

[76] Quayle, T. W., Mining Methods and Practices at the United Verde Copper Mine, Jerome, Ariz.: Inf. Circ. 6440, Bureau of Mines, 1931, 31 pp.

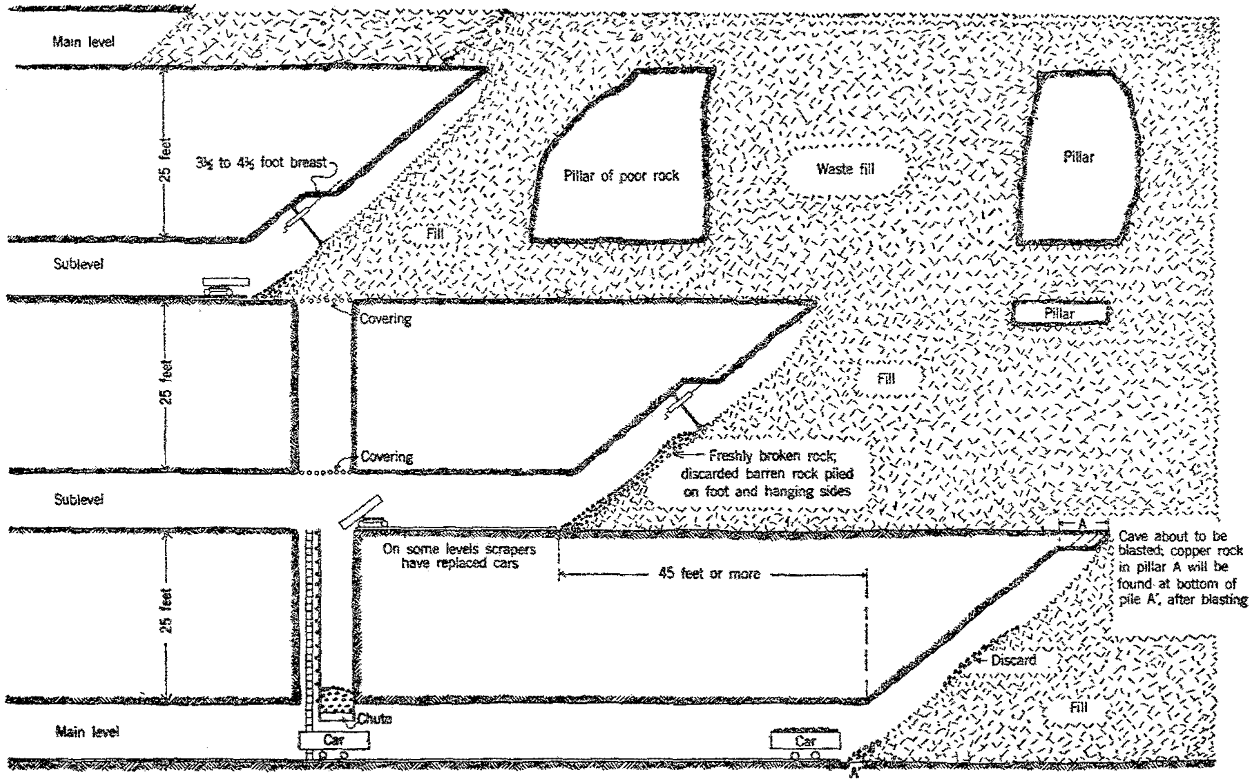


FIGURE 74.—Inclined cut-and-fill stoping using sublevels at Champion mine.

level above. This raise is usually located as near the center of the stope as possible. If the stope is more than 100 feet long a second raise is driven near one end of the stope. Chute raises are erected on 16½-foot centers in sulphide ore and 22-foot centers in porphyry or schist ore. The stopes are then silled, beginning at the waste raise 21 to 25 feet above the rail. Stoping consists of breaking 7-foot horizontal slices, removing the ore, and filling, as shown in figure 76.

The ore is shoveled into the chutes; about 30 percent of it requires secondary blasting to pass through the 11-inch grizzlies. Scrapers have been tried but have proved uneconomical on account of the large percentage of oversize.

A temporary shoveling floor of 2-inch planks is laid on the fill before a cut is blasted. The filling is drawn from a chute into an end-dump car, which is pushed by hand on a sectional track. Cribs or bulkheads are used to support weak parts of the back and are recovered for reuse after each cut.

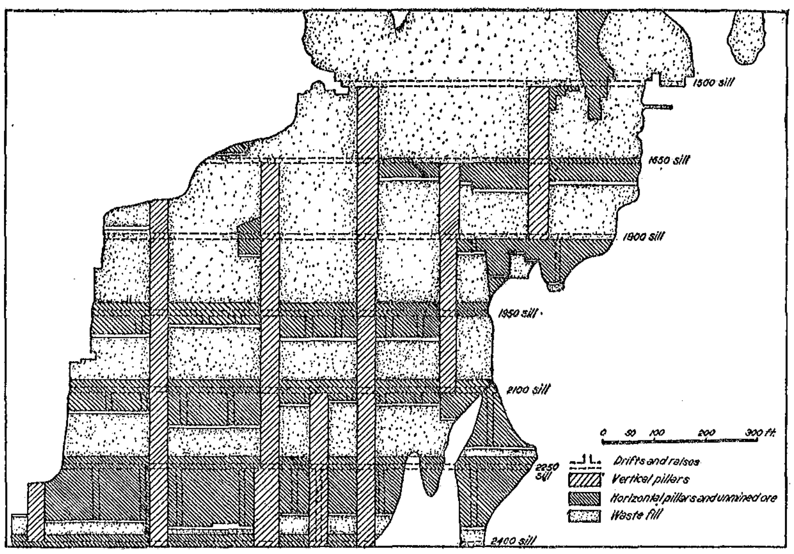


FIGURE 75.—Diagrammatic section, 1,500- to 2,400-foot levels, showing pillar arrangement, United Verde mine.

FROOD

A horizontal cut-and-fill method (fig. 77) similar to that used at the United Verde mine was adopted for the lower levels of the Frood mine of the International Nickel Co. of Canada, Ltd., in the Sudbury district, Ontario. Stopes 45 feet wide are laid out across the strike of the ore body with 35-foot pillars between. Mutz [77] states:

> The horizontal cut-and-fill method was adopted as most suitable to the conditions of the ore body on the lower levels. This area was known to be of a grade that required a method permitting the highest extraction. Drilling had disclosed that considerable waste rock was present in certain zones—a condition that points to the possibility of underground sorting. Likewise, the drill holes had proved the presence of barren horses of rock within the ore body and had also indicated that the footwall and hanging

[77] Mutz, Herman J., Mining the Frood Ore Body at Depth: Eng. and Min. Jour., vol. 130, Nov. 10, 1930, pp. 445-452.

wall were very irregular. It was felt that close filling would be essential to leave the ore body above the 2,000 level intact and in condition to mine later.

Heavy ground in some parts of this mine, however, has necessitated a change to the square-set system.

The ore body is outlined by diamond drilling ahead of level development. Crosscuts are run from a footwall drift to the ore body at

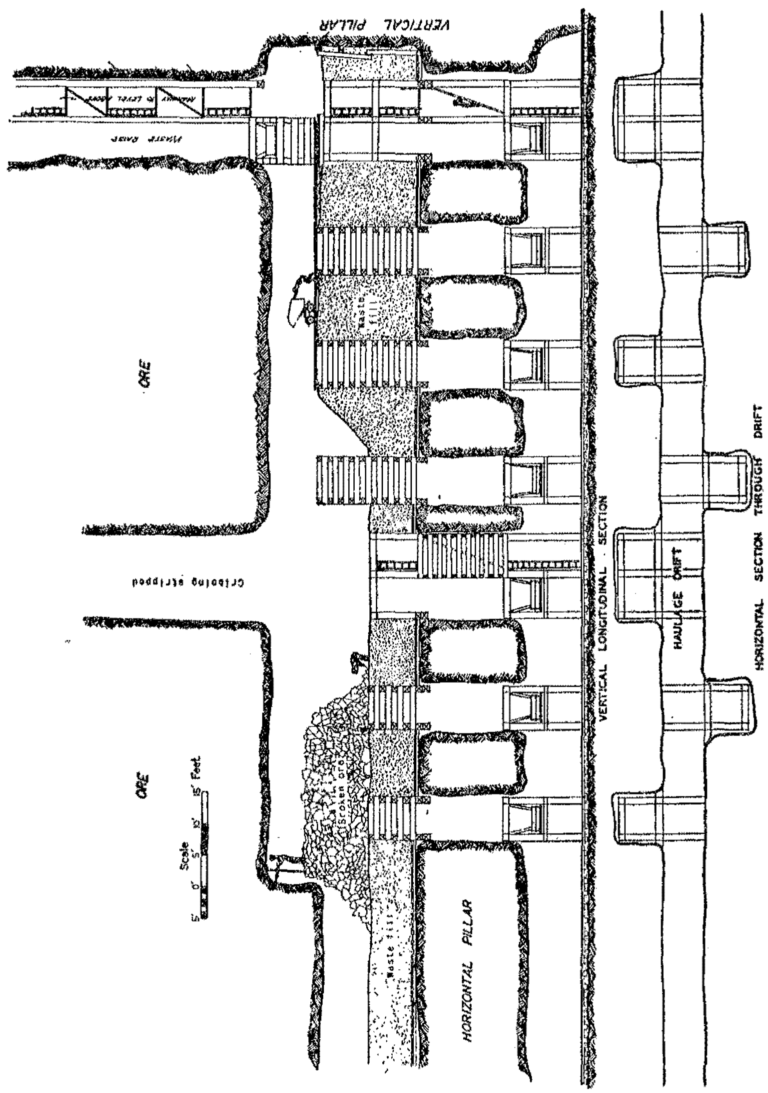


FIGURE 76.—Typical cut-and-fill stope, United Verde mine, Jerome, Ariz.

400-foot intervals. The first extraction drift in the cut-and-fill system of mining is run 10 feet within the ore on the footwall side. Parallel drifts are run at 44- to 60-foot intervals, the spacing depending on the width and strike of the ore. Where the ore is less than 70 feet wide only one extraction drift is run; where it is 200 feet wide four are used. Chutes are located on opposite sides of the extraction drifts at intervals of 22 or 27½ feet. A fill raise is driven near the center of the ore in each stope. Where the ore is more than 160 feet wide a second fill raise is driven near the footwall. At both the Frood and Creighton mines a small part of the filling comes from development work and the rest from waste sorted from the ore at the surface. The waste reaches the top of a stope through an independent system of fill raises. After the sill floor is completed stoping begins; the height of each cut is 7 feet. The back is usually supported by one or two lines of cribs of 8- by 10-inch by 7-foot timber down the middle of the stopes.

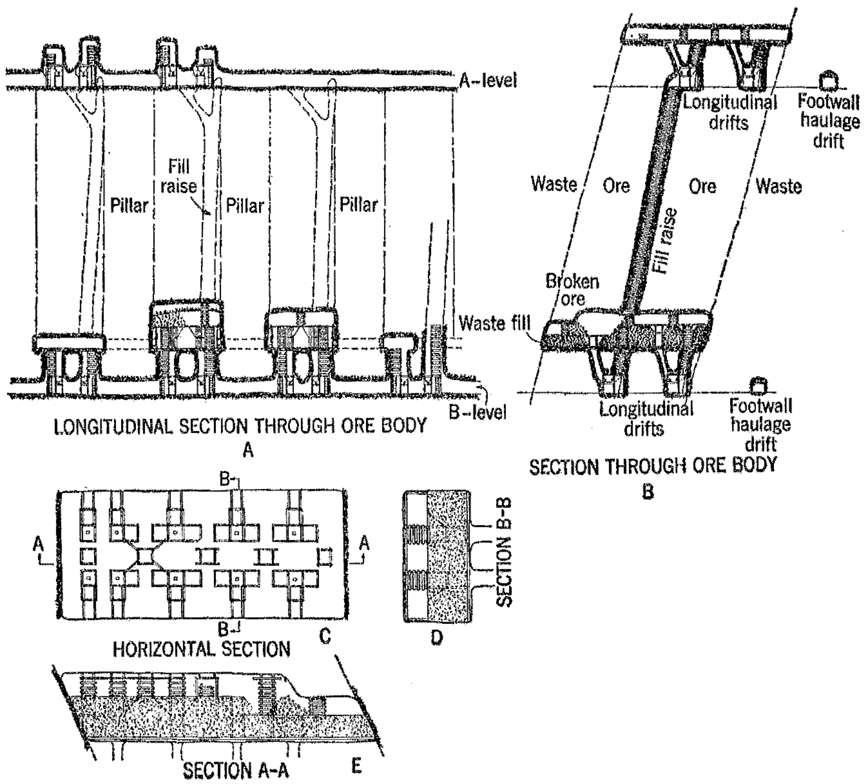


FIGURE 77.—Cut-and-fill stopes, Frood mine.

CAMPBELL

The Campbell mine of the Phelps Dodge Corporation was formerly a part of the holdings of the Calumet and Arizona Mining Co. in the Warren district, Bisbee, Ariz. It is worked through a vertical shaft as a separate unit, although it is connected to the older workings. Haulage levels are 200 feet apart, with a working level midway between two levels.

The outlines of the ore body were established and the method of mining was planned before stoping began. The ore body was divided into stope and pillar sections, with two contiguous stoping sections, each 45 to 50 feet wide, separated by 45-foot pillars from the pairs of stopes on either side.[78] Pillars were alined vertically from level to level throughout the ore body. In wide parts of the deposit above the 1,600-foot level a section is divided by a row of sets into two stopes. No pillars had been mined up to March 1932.

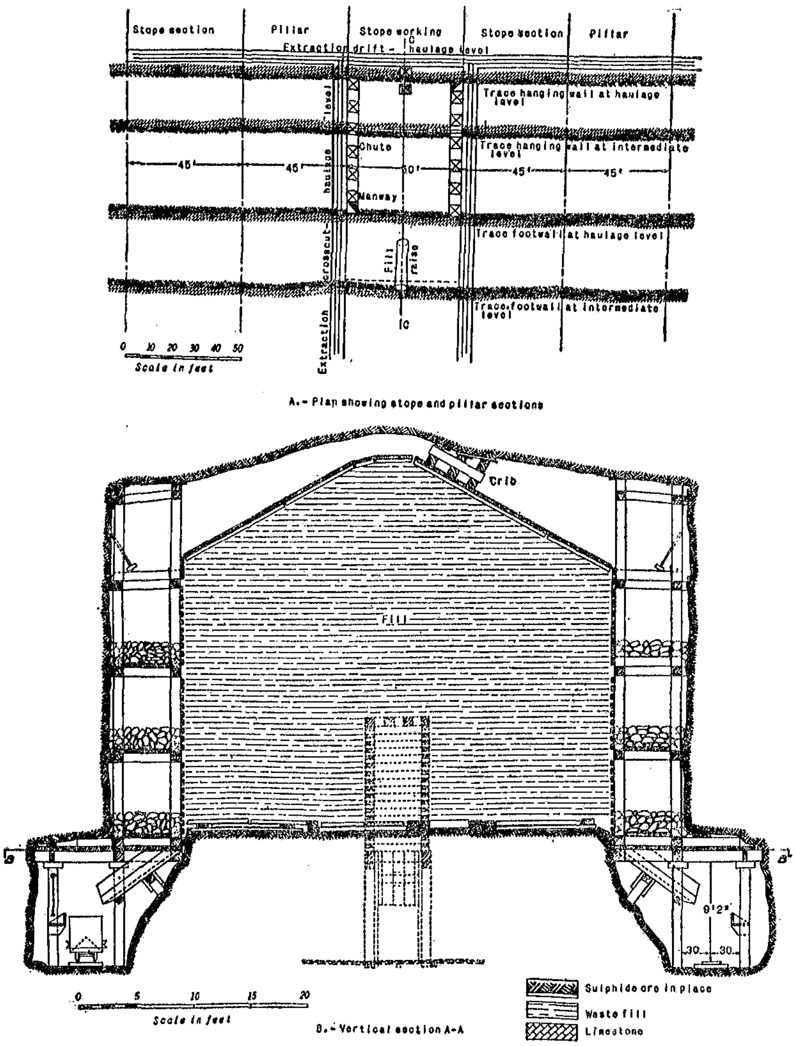


FIGURE 78.—Double-lead, inclined, flat-bottomed, cut-and-fill stope with fill raise in footwall, Campbell mine.

Stoping starts on a haulage level and is carried the entire 200 feet if the ore is continuous. Fill raises are run in the ore, usually on the footwall. Extraction crosscuts are run on the pillar lines. (See fig. 78.)

The stopes are silled at the level of the top of the crosscuts and floored with scrap timber. The back is sloped upward from hanging

[78] Lavender, H. M., Mining Methods at the Campbell Mine of the Calumet and Arizona Mining Co., Warren, Ariz.: Inf. Circ. 6289, Bureau of Mines, 1930, 18 pp.

wall to footwall at an angle of 37°, the angle of repose of the waste fill; then the broken ore is removed and the stope filled with waste rock through a raise that has previously been run to the level above. Stoping proceeds in regular cycles by mining 12-foot slices, cleaning out the broken ore, and filling. Stulls are used to give temporary support to loose ground in the back. The ore is taken out through rows of square-sets carried up each side of the stope.

Many variations of this method are practiced. A row of square-sets may be used on one side only; the other side of the stope if in ore is lined with single sets and lagging. A cribbed raise may be carried up through the fill to serve as an additional extraction chute. If the ground is strong enough a semishrinkage method may be used, that is, several slices may be taken before filling, which reduces flooring and filling costs. Stope sills may be flat, as shown in figure 78, peaked like a roof, or gently inclined from one side to the other.

COLORADA

The Colorada mine of the Cananea Consolidated Copper Co. now produces most of the output in the Cananea district of Mexico. During 1929 about 54 percent of the ore from this mine was produced from cut-and-fill stopes. Inclined cut-and-fill stoping was used in the relatively narrow ore above the 700-foot level.[79] A horizontal cut-and-fill method is used in the main ore body. Virtually all of the inclined cut-and-fill stopes were completed by 1932. The typical horizontal cut-and-fill stope is 30 feet wide by 50 to 160 feet long and has a row of raises along the center line at 40-foot intervals. The stope lay-out and method of stoping are shown in figure 79. Pillars 40 to 50 feet wide are left between the stope sections; the pillars are mined by top slicing.

On the 900 level the stopes were silled 11 feet above the rail; on the 1,000 level, to protect the haulage drifts, the sill floor is 22 feet above the floor of the drifts (two top-slice floors). The stope is silled 11 feet high, starting from the raises, then filled. In good ground it has been found possible to mine two slices before filling. Stulls and cribbing are used occasionally for temporary support of loose ground. Formerly the sides of the stope adjacent to ore pillars were carefully timbered. Since a decomposed tuffaceous rock has been used for fill this has been found unnecessary, because the new material packs so tightly as not to run when the adjacent pillars are top-sliced. Scrapers and wheelbarrows are used to pull the ore to chutes and to spread the fill. One major difficulty, that of breaking the heavy, blocky ground by primary blasting sufficiently to be handled, has been largely overcome by what is called space blasting; that is, spreading the charge in drill holes by inserting spacers between the sticks of gelatin dynamite. Figure 83 (p. 242) shows the grizzlies at the ore pockets, also a motor and train at the 800 level.

MAGMA

A combination rill-stope and pillar system is used at the Magma mine, Superior, Ariz., in places where the vein is more than 15 or 20

[79] Catron, William, Mining Methods, Practices, and Costs of the Cananea Consolidated Copper Co., Sonora, Mexico: Inf. Circ. 6247, Bureau of Mines, 1930, 41 pp.

feet wide.[80] The rill stopes are 16 feet wide (fig. 80) and are run across the ore body. The pillars are 14 feet wide. A haulage drift is first run in the footwall, with crosscuts every 250 feet to the vein. An

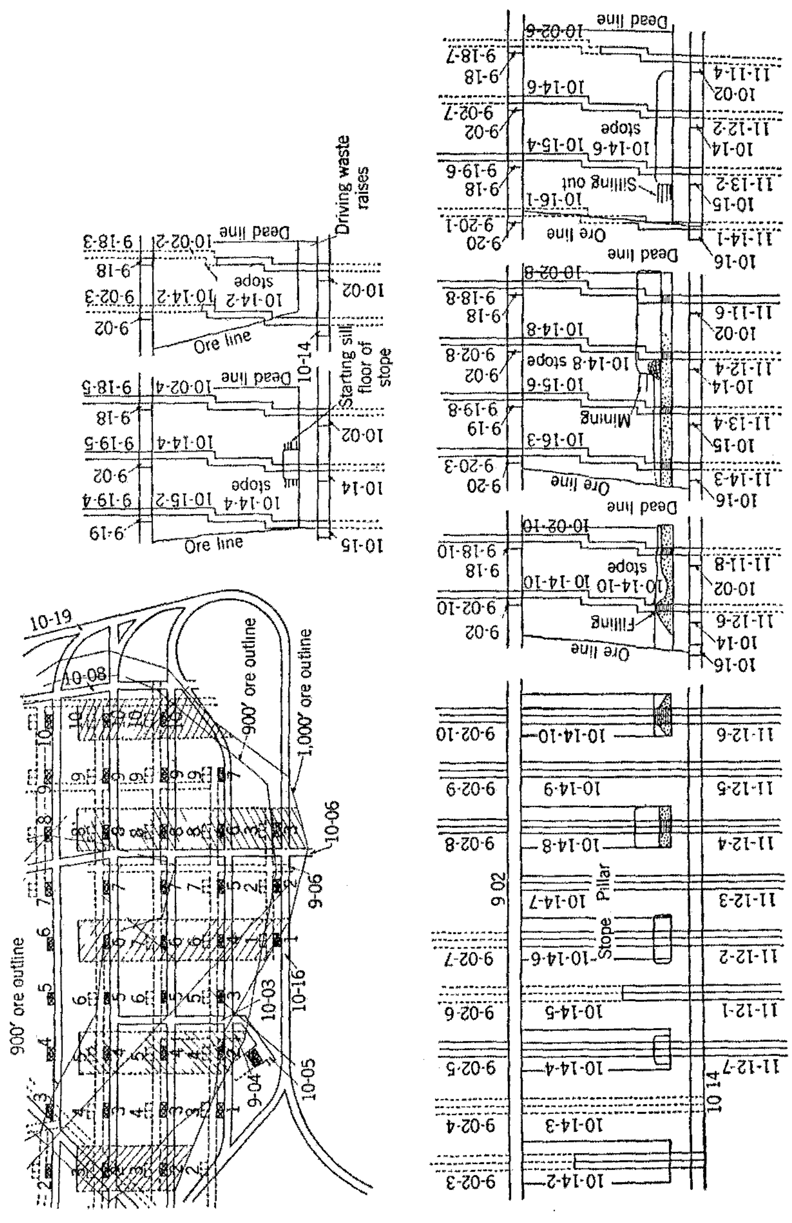


FIGURE 79.—Horizontal cut-and-fill stopes, Colorada mine.

extraction drift is then run in the vein on the footwall. Crosscuts from this drift are run on 30-foot centers to the hanging wall, widened

[80] Snow, Fred W., Mining Methods and Costs at the Magma Mine, Superior, Ariz.: Inf. Circ. 6168, Bureau of Mines, 1929, 32 pp.

out to 16 feet, and shot down to a height of 10 or 12 feet, thus forming the sill floor of the rills. The length of the rills ranges from 15 to 60 feet. A double 2-inch plank floor is laid on 10- by 10-inch timbers, 16 feet long, placed on 5-foot centers across the vein. A row of square-sets is placed across the vein at one pillar line and held open to give access to a hanging-wall drift. Three square-sets are carried up on the footwall to form a three-compartment raise as mining progresses. On the 2,800-foot level 21-foot stopes with 14-foot pillars were used successfully. A saving was made in the number of waste raises necessary.

After all sill-floor timbering is in place a cribbed raise is started in the hanging wall of the vein about 12 feet from the bottom of the sill. The broken waste from the raise is dropped into the stope for filling. When the natural slope of the waste brings it about 4 feet from the back, a floor is laid on the waste and a 7-foot cut of ore is mined.

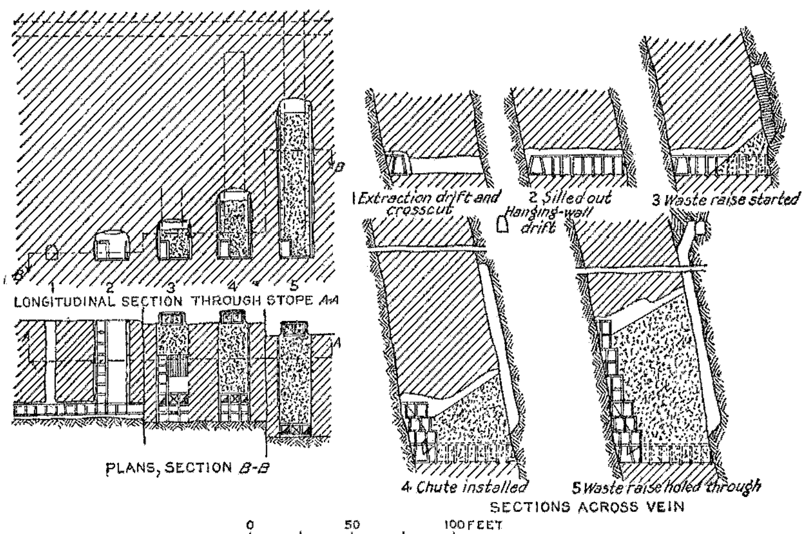


FIGURE 80.—Rill stopes, Magma mine, Superior, Ariz.

The back of the stope is inclined upward to the hanging wall, and its slope roughly parallels the angle of rest of the waste fill, which is about 35°. Mechanical shovels or scrapers are used to load the ore into cars on the sill floor until the slope is high enough to permit the ore to be run by gravity into the extraction raise.

The broken ore is cleaned out after each cut, and gob lines comprising single posts and 2-inch lagging are placed along each side. The ends of the posts are framed in half, and a single strand of old hoist cable is used to tie together corresponding posts on opposite sides of the stope; the cable passes around the joints of the posts. The floor is taken up and used in the gob line.

The waste raise advances only as fill is needed in the stope, usually reaching the level above when the stope is about 35 feet high on the hanging-wall side. Subsequently waste from other sources is dropped into the stope as required. The waste raise is also used for lowering timber and supplies into the stope. An attempt is made to keep the raise open below the level of the stope fill only under exceptional conditions (as, for example, when required for ventilation). When the waste raise approaches the level above, it is inclined away from the vein so as to connect with the near side of the hanging-wall drift. This drift is run to leave a 13-foot pillar between it and the vein. The top of the chute compartment of the waste raise is covered with a 9-inch grizzly made of 30-pound rails.

When the stope approaches the level, square-setting is used to extract the ore directly under old filled stopes. The pillars are mined as described under square-setting, page 195. Where the ore is less than 15 to 20 feet wide, cut-and-fill stopes 105 feet long are run on the strike of the vein with 15-foot pillars between sections. Raises are run in the ore to the level above. Stoping begins on both sides of a raise. Until the rills are established, the ore is shot down on timbers over the drift and loaded into cars through temporary chutes. Boards are laid on the fill before shooting the ore. Some cribbing is needed to hold the back until the space is filled; this is salvaged and reused. A line of sets is carried up each side of a pillar. One set is used as a manway and the other as an ore chute. In veins less than 6 feet wide, two sets are put in parallel to the strike. After two adjoining sections are completed the intervening pillar is mined as before by the Mitchell top-slice method.

ANACONDA

About 15 percent of the ore produced at the Butte (Mont.) mines of the Anaconda Copper Co. comes from cut-and-fill stopes; the square-set method described on page 187 accounts for the other 85 percent. Cut-and-fill stopes are employed in narrow veins, usually not more than 15 feet wide, where the wall rock is relatively strong. Inclined cuts are used in nearly all stopes. The stope is developed in the same manner as for inclined square-setting. (See fig. 56.) The drift is cut out the full width of the ore and timbered. Three-compartment raises on 112-foot centers are run to the level above and timbered with regulation square-sets. The middle compartment is used for a manway and the outside compartments for dropping fill into the stopes. After the drift is timbered the tops of the drift sets are lagged over, and an 8-foot cut is taken the full length of the stope. The ore is dropped through the lagging into cars in the haulage drift. This first floor is timbered with square-sets; temporary chutes are built in every other set along the track.

The stope is then developed for rilling by taking progressive 8-foot cuts on a 40° incline, beginning at both ends of the stopes at the fill raises. The ore is allowed to pile up in the stope to permit the miners to reach the back. After the slope is established the ore is drawn from the stope and a gob floor laid in the row of sets above the drift floor as in square-setting. Filling is then run into the stope from the raises at either end until it lies at its angle of repose, after which a floor is laid on top of the fill.[81] The toe of the slope of the fill is 10 feet from the ore chute.

Three lines of square-sets, comprising two ore chutes with a manway between, are erected in the center of the stope. In addition, two rows of sets are placed on either side of the chutes to provide a space for

[81] Harrer, C. M., Utilizing Gravity in Mining and Sorting: Eng. and Min. Jour., vol. 133, 1932, pp. 146–148.

the disposal of waste sorted from the ore. Formerly sets were not so placed, and sorted waste was piled at one side or dropped into a separate chute alongside the ore chute whence it was drawn into cars for disposal elsewhere.

The area on top of the two sets on either side of the chutes provides a sorting platform. Although this method necessitates shoveling all of the ore, the saving made due to more efficient sorting more than compensates for the added expense.

Successive 8-foot cuts of ore are taken, and the space is filled to within two or three sets of the level above. The top of the stope is mined by square-setting. Including raises, about half of the stope is mined by this method. To permit continuous production a cut of ore is usually taken on one side of a stope while the other side is filled. When horizontal cuts are taken ore chutes 25 to 40 feet apart are built up through the fill. Fill is drawn into cars from the fill raises and distributed.

GLORY HOLE

The choice between a glory hole and an open pit is usually decided by the scale of operations and the size of the deposit. The cost of equipping a mine for glory-hole work is much less than that for open-pit work, and for a small or moderate tonnage the over-all mining cost would be correspondingly less. However, for large-scale work over a long period lower costs are obtainable with open-cut mining.

The method of stoping by glory holes (sometimes called underground milling or mill holes) has been used at many copper mines for extracting the upper parts of vein deposits that come to the surface. As depth is attained other methods are used. At the mine of Britannia Mining & Smelting Co., Ltd., British Columbia, the method has been used successfully for mining some of the ore body that extended to the surface. It is the principal method used at the relatively new Copper Mountain mine and has been much used at the Hidden Creek mine, both of which belong to the Granby Consolidated Mining, Smelting & Power Co., Ltd., British Columbia.

In the United States the method has been applied to copper mining principally at the Sacramento Hill mine of the Copper Queen branch of the Phelps Dodge Corporation, at Bisbee, Ariz. The method was tried in the lower pit of the United Verde open-cut mine to mine the ore after the overburden had been removed by power shovels but was unsuccessful, mainly on account of the difficulty of mining separately the different products of the mine.[82]

COPPER MOUNTAIN

The Copper Mountain or Allenby mine is near Copper Mountain, British Columbia. The ore consists of disseminated copper sulphides in monzonite.[83] One ore body, which is 2,700 feet long and 50 to 75 feet wide, contains 3 percent of copper, but the average copper content of the ore in the mine is 1.5 percent. The mine is operated through three adits and an interior shaft. The ore is mined mainly by glory-hole methods, but shrinkage stoping is used in some places in the mine. The ore is concentrated in a plant having a capacity of 3,000 tons a day.

[82] Alenius, E. M. J., Methods and Costs of Stripping and Mining at the United Verde Open-pit Mine, Jerome, Ariz.: Inf. Circ. 6248, Bureau of Mines, 1930, 32 pp.
[83] Mines Handbook, 1931, p. 2117.

The production during 1929 was about 920,000 tons of ore, which yielded 22,540,000 pounds of copper; in 1930 it was 700,000 tons of ore, which yielded 15,490,000 pounds of copper.

HIDDEN CREEK

About 30 percent of the ore produced at the Hidden Creek mine at Anyox, British Columbia, has come from glory holes. In 1932, however, only 5 to 10 percent was derived from this source.[84] The method is used to mine the direct-smelting ore where the spirals described under "Open stopes" (p. 181) have been brought near the surface; it has also been used to mine concentrating ore in two other ore bodies. A high tonnage per man-shift and very low costs are obtained.

The method is used throughout the year, despite the deep snows of winter. The ore is broken in benches 8 to 10 feet high wherever it is possible to keep the slope of the sides of the glory hole at an angle of about 45°. On flatter slopes lower benches are necessary. The holes are drilled with machines mounted on tripods, except that heavy jackhammers are used where there is not room enough to set up a tripod. Long drill holes are sometimes chambered. The ore is drawn through 18-inch grizzlies, where all large pieces are block-holed.

BRITANNIA

At the mine of the Britannia Mining & Smelting Co. Ltd., Britannia Beach, British Columbia, glory-hole mining was practiced in the Bluff ore body where the deposit was large enough to warrant its use. Later glory holes were opened up in the upper Fairview where the closely spaced parallel veins with some mineralization between made this method preferable to more expensive selective mining.[85] Working benches were 3 feet wide and varied in length. The ore was drawn through grizzly chambers into transfer raises of the underground transportation system.

Later, when the problem became one of handling still lower-grade blocks of virgin ground, as well as ore left around old glory holes and underground stopes connected with the glory hole, benching became impractical and coyote blasting was adopted. In the East Bluff 25,000 to 50,000 tons of burden are broken by each blast; in the Fairview as much as 150,000 tons has been broken at once. Although it is necessary to break some waste to get all the ore and although the ore is diluted owing to normal caving from the hanging wall, the lower breaking cost offsets the cost of handling the extra tonnage. The actual recovery of ore is high. The method formerly used is shown in figure 81.[86]

SACRAMENTO HILL

When the depth of operations in the Sacramento Hill open-cut mine at Bisbee, Ariz., made transportation of the ore from the pit unduly expensive, the lower extension of the ore body was mined by a glory-hole method. The ore obtained by this method in 1929 amounted to 574,000 tons and in 1930 to 304,000 tons. The maximum produc-

[84] Lindsay, W. R., and Healy, R. L., Mining Methods, Hidden Creek Mine; Trans. Canadian Inst. Min. and Met. Eng., vol. 32, 1920, pp. 187–193.
[85] Brennan, C. V., Mining Operations at the Property of the Britannia Mining & Smelting Co. Ltd., Britannia Beach, British Columbia: Inf. Circ. 6815, Bureau of Mines, 1935, p. 19.
[86] Moore, James I., Jr., Operations at the Britannia Mines: Eng. and Min. Jour., vol. 122, no. 24, Dec. 11, 1926, pp. 924–930.

tion was about 2,500 tons a day. The bottom of the old pit was oval, about 450 feet long, and 400 feet wide. Development work for mining this ore by glory holing consisted of opening a haulage level 2,000 feet long on the 500-foot level of the Sacramento shaft, 239 feet below the bottom of the pit. A loop was driven under the pit to serve six glory holes. (See fig. 82.) Later five more glory holes were developed. A grizzly level was put in 135 feet below the bottom of the pit and connected with each glory-hole raise. (See fig. 82.) The bottom of each raise constituted a pocket 24 feet long, 8 feet wide, and 45 feet high. Pockets and raises were unlined. Grizzly stations or bulldozing chambers were 8 feet wide, 7 feet high, and 15 feet long. The grizzly bars comprised 100-pound rails 17 inches apart.

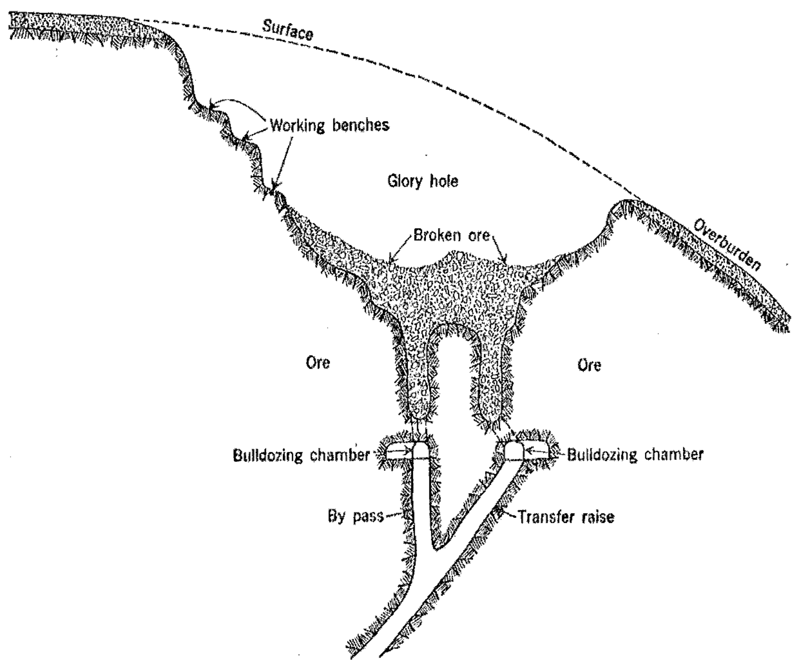


FIGURE 81.—Vertical section of glory hole, Britannia mine.

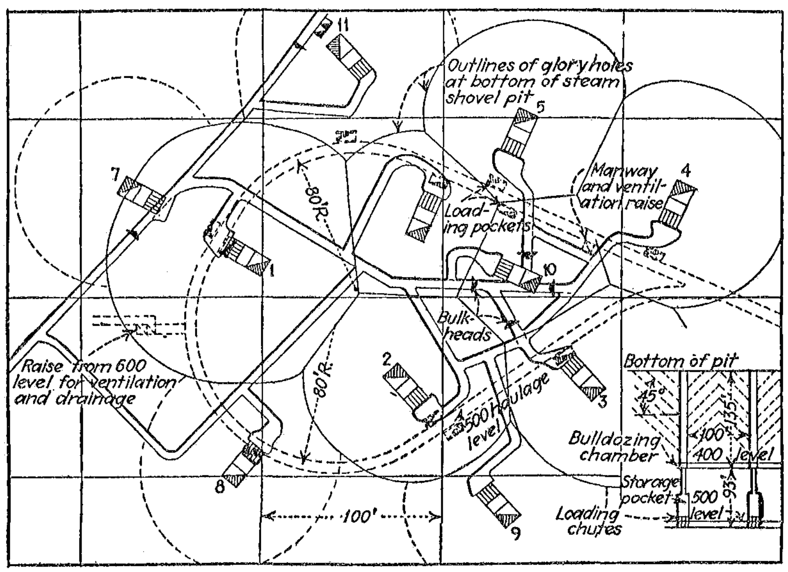
Next, 11 feet from the edge of the grizzly, three-compartment, L-shaped raises were erected to a height of 90 feet and within 35 feet of the surface. The raise could not be holed through because the steam shovels were still working in the pit while the glory-hole mine was being developed. After a raise was completed the timber was shot out. The stripped raises were over 10 feet in diameter. To hole through to the surface several churn-drill holes were drilled from the pit over each raise and shot simultaneously.

The haulage level was connected with the grizzly level by several manway raises, and one manway and ventilation raise extended to the surface.

Production was started at the glory holes by drilling and blasting 6-inch churn-drill holes 60 feet deep and 18 to 23 feet apart around the raises far enough back to insure a firm foundation; two gasolinepowered drills were used. As the craters were established other holes were drilled 18 to 25 feet deep on the slopes with air drills mounted on tripods, usually beginning at a pocket made by a blast of a churn-drill hole.

The boulders resulting from shooting the churn-drill holes were very large; those that could be reached were plugged and blasted from the surface. After the glory-hole craters were established a large proportion of the ore was broken by means of air-drill holes; this practice cut down the proportion of large fragments. The ore reaching the grizzly, however, was very coarse, and over the 2-year period breaking costs at the grizzly were more than at the surface.

Six crews, each comprising a driller and a helper, did the drilling. One two-man crew blasted the churn-drill holes and another loaded and fired the air-drill holes. The steel and explosive were distributed



A, PLAN **B, DIAGRAMMATIC VERTICAL SECTION THROUGH TWO GLORY HOLES**

FIGURE 82.—Glory-hole method of mining, Sacramento Hill mine.

by a packer with a burro. A pipe fitter also worked in the pit. About 100 tons were mined per man-shift.

Two men worked at each grizzly and pulled the ore through with bars; heavy reinforcing around the grizzly set afforded protection from this blasting. The ore was pulled from the pockets at the haulage level into 4-ton gable-bottom cars and hauled in trains of 15 cars by a trolley locomotive to a 200-ton shaft pocket.

Figure 84 shows a churn drill in one of the glory holes. Figure 85 is a photograph of men working at a bulldozing chamber under the Republica glory hole at Cananea.

TOP SLICING

Top slicing consists of mining an ore body from the top downward by horizontal or inclined slices and allowing the capping to follow down as each slice is taken out. It is distinguished from a caving method in that the ore itself is not caved. As each slice is taken out the resulting space is timbered and a floor laid. The timber is next blasted, which allows the capping to settle down. A timber mat is soon built up which prevents the waste above from mixing with the ore. The floor of one slice is the top of the one next below.

Top slicing usually is confined to fairly wide ore bodies over which the wall or cap rock will cave readily. If the overlying rock does not follow down readily as slicing proceeds, it must be made to cave or the space must be filled. Usually in copper mines the top part of an ore body to be mined by top slicing is stoped by square-setting and filled to provide a cushion of waste under which the slices are taken.

The method varies principally in the manner of cutting the slices (horizontal or inclined). With an inclined slice shoveling is eliminated in the stopes, but as the workings are on a slope other operations are more difficult. With advances in the technique of scraping, the disadvantage of working on a sloping floor probably outweighs the saving made by not having to handle the ore in the stopes. Moreover, with scrapers ore chutes may be placed farther apart, with a resultant saving in stope development. In top slicing, as in caving, a relatively large amount of preparatory work is necessary before stoping begins. Under usual conditions the daily output for each stope is relatively small, which makes a number of working places necessary for a large output.

Top slicing has been used extensively in copper mines, but of late years comparatively little copper has been produced by this method. The method, patterned after the practice in the iron mines of the Lake Superior region, was first introduced in the Southwest at Cananea in 1907. It displaced square-setting at Cananea in many ore bodies and was the principal method used until the Colorada mine was developed in 1928.

The first ore mined by the Miami Copper Co. at Miami, Ariz., in 1911 was produced by top slicing. Before the method was supplanted by shrinkage and caving of pillars, which in turn was displaced by block caving, over 4,500,000 tons of ore were mined.

During the last few years top slicing has been used mainly for extracting pillars between cut-and-fill stopes at the Colorada mine at Cananea (where it is again the principal method used), for mining a few ore bodies at Bisbee, and for taking out caved pillars over cut-and-fill stopes at the United Verde mine.

CANANEA

The pillars between cut-and-fill stopes in the Colorada ore body are 40 to 50 feet wide and 60 to 100 feet long (see p. 234); they are mined by top slicing. Ultimately about 60 percent of the main ore body will be removed by this system.[87] The top slice stopes are partly developed by the work done preparatory to mining the cut-and-fill stopes. Drifts have been run along the center lines of the pillars

[87] Catron, William, Mining Methods, Practices, and Costs of the Cananea Consolidated Copper Co., Sonora, Mexico: Inf. Circ. 6247, Bureau of Mines, 1930, 41 pp.

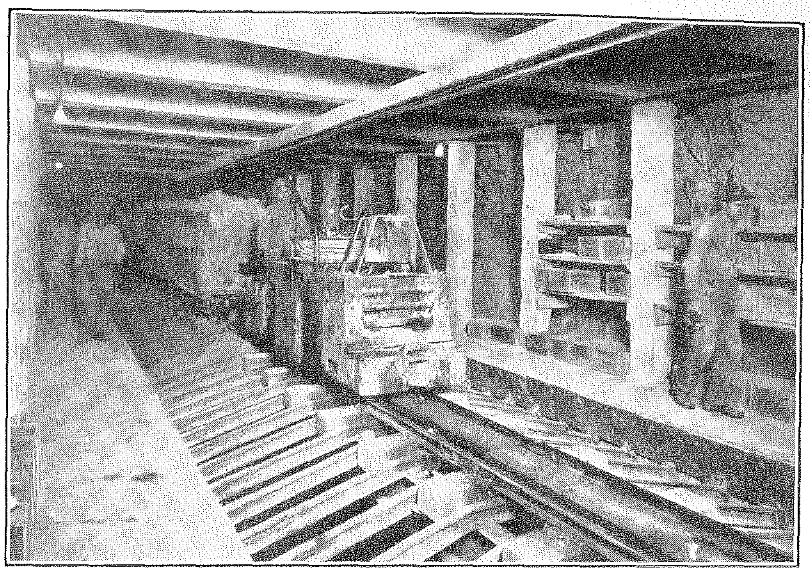


FIGURE 83.—Motor and train at 800-foot level, Colorada mine.

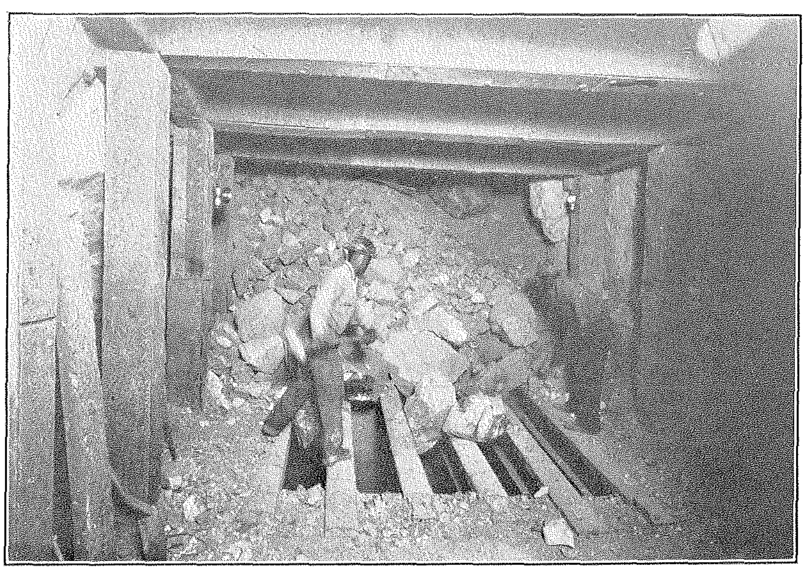


FIGURE 84.—Bulldozing chamber under Republica glory hole, Cananea mine.

FIGURE 85.—Glory holes, Sacramento Hill mine.

and raises put up to the top of the ore, generally at 40-foot intervals. In addition, some raises have inclined branches or fingers extending toward the edges of the pillars, which have been used for drawing ore from the cut-and-fill stopes on the level above. In starting a top slice, as described by Catron, a heading was driven lengthwise through the pillar on the top floor; it intersected the raises or the fingers in the pillar and extended past the end of the block into country rock, where it connected with a permanent manway raise. The heading was driven as small as possible, about 3 or 4 by 6 feet, leaving 5 feet of solid ore between it and the filled or caved stope above. Stoping began at the end of the heading away from the manway. A cut 5 feet wide and 11 feet high (the standard height of top-slice floors) was carried across the heading to the old fill on each side. The sills of the old floor above were caught up by 8-inch round posts with long head blocks. Another 5-foot cut was then drilled and blasted. Scrapers were used to move the ore into the chutes. Two to four cuts 10 to 20 feet long, depending upon the weight that developed, were taken out before the timbers were shot down, and a floor was laid which comprised 2-inch planks 10 feet long nailed to double sills of the same material. The sills were laid parallel to the direction of advance of the stope for convenience in picking them up when taking out the next floor below. A light gob fence of 2-inch plank nailed to the row of posts nearest the face was built around each area before it was caved. All posts except those in the gob fences were drilled with air augers and blasted with a stick of powder each. These shots were timed so as to bring the floor down as smoothly as possible and more or less intact. The same procedure was repeated on each floor. Where two or more floors were worked at one time the lower had to be kept at least 40 to 50 feet behind the upper to avoid interference.

The method proved satisfactory, and all ore was recovered with practically no dilution; however, the system has been improved. According to C. W. Weed, general manager, Cananea Consolidated Copper Co., a modification of this method which has been adopted involves driving a subdrift one or two floors below the active slice. From this sublevel short finger raises are put up to the mining floor. The broken ore falls or is shoveled through these raises onto the floor of the sublevel, where it is pulled by a scraper to a single, central transfer raise from the haulage level. This not only eliminates two or three main raises in each pillar (where they were not already driven) but permits better spacing of ore chutes in the slices, thus making a saving in ore-handling costs. The method is shown in figure 86. In June 1932, 45.5 percent of the ore was mined by top slicing, 33.5 percent by cut-and-fill stoping, and 19.3 percent by shrinkage stoping; 1.7 percent was obtained from development work.

COPPER QUEEN

A modification of top slicing and square-setting, which has been developed and used successfully at the Copper Queen mine, at Bisbee, Ariz., is locally termed the "Copper Queen slot method." The method of laying out the stopes is the same for either top slicing or square-setting. One advantage of the method is that top slicing can be changed to a square-set or cut-and-fill method whenever selective mining becomes necessary, as when metals are low in price. In top slicing, the floors are timbered with square-sets 11 feet high, instead of 8 feet as in square-setting.

The ore bodies are developed by running a drift through them lengthwise, with crosscuts from the drift on 50-foot centers. Raises are also put up on 50-foot centers to the top of the ore body or to the level above.[88]

After the preliminary work is done slots on 20-foot centers are mined up through the ore body to the level above or the top of the ore. A slot is one set wide and two sets long on the first floor (floor next above the drift). The length is increased one set on each side of the center for each additional floor until the maximum length is attained. Although the sections are rarely over 50 feet long, occasionally slots are 14 sets (70 feet) long at the top to reach bulges in the ore body.

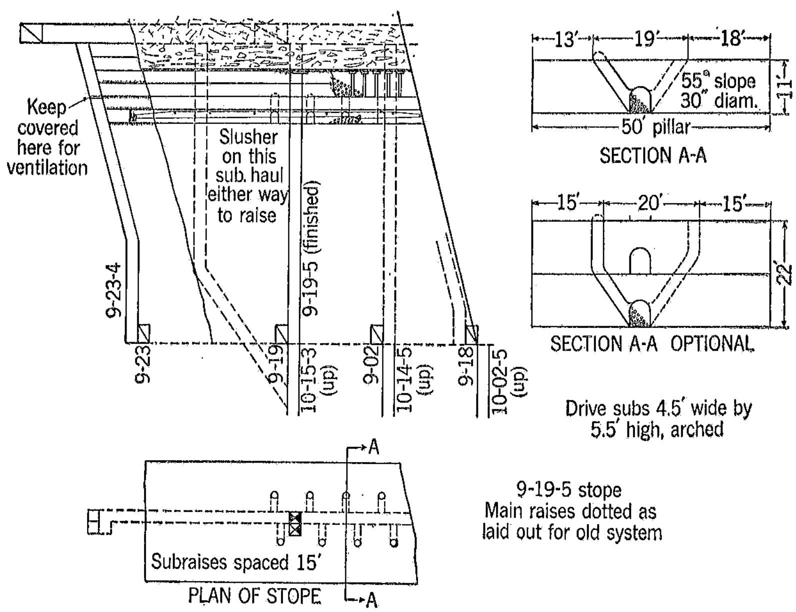


FIGURE 86.—Modified (Catron) top-slicing method, Colorada mine.

The manner of placing the slots depends mainly on the shape of the ore body. It is planned that all ore except relatively small offshoots be within 15 feet of a slot. In a large ore body several parallel slots are run; in a smaller deposit only one main slot may be needed. Slots at right angles to each other may be used in ore bodies of nearly circular outline or in a large ore body for mining bulges. Two square-sets at one side of the middle of the main slot are lagged and carried up along with the slot. One compartment is used for a manway and the other for getting supplies into the slot or later into the stope.

When the stope is top-sliced a floor or section of a floor at the top of the ore is mined out and timbered with square-sets. Sills are placed, and a timber mat is laid. Then the floor or panel is shot down.

[88] Gardner, E. D., and Vanderburg, Wm. O., Square-set System of Mining: Inf. Circ. 6691, Bureau of Mines, 1933, 73 pp.

Each floor is connected to the service raises, which are kept open between levels until the stope is finished.

When a stope is mined downward the ore left near the bottom near the ends of the slots is taken out on each floor as reached. Short vertical or inclined untimbered raises, locally called bean holes, are run from the drifts or crosscuts to receive this ore when it is more than 15 feet from a slot. If an offshoot is not large enough to justify a separate raise, the ore is brought in wheelbarrows to the nearest slot. When a stope is mined upward this lower ore is taken out through the short raises after the rest of the stope is filled. Figure 87 shows a stope mined by square-setting. The same method is used for top slicing except that the waste is allowed to follow each floor down as it is taken out.

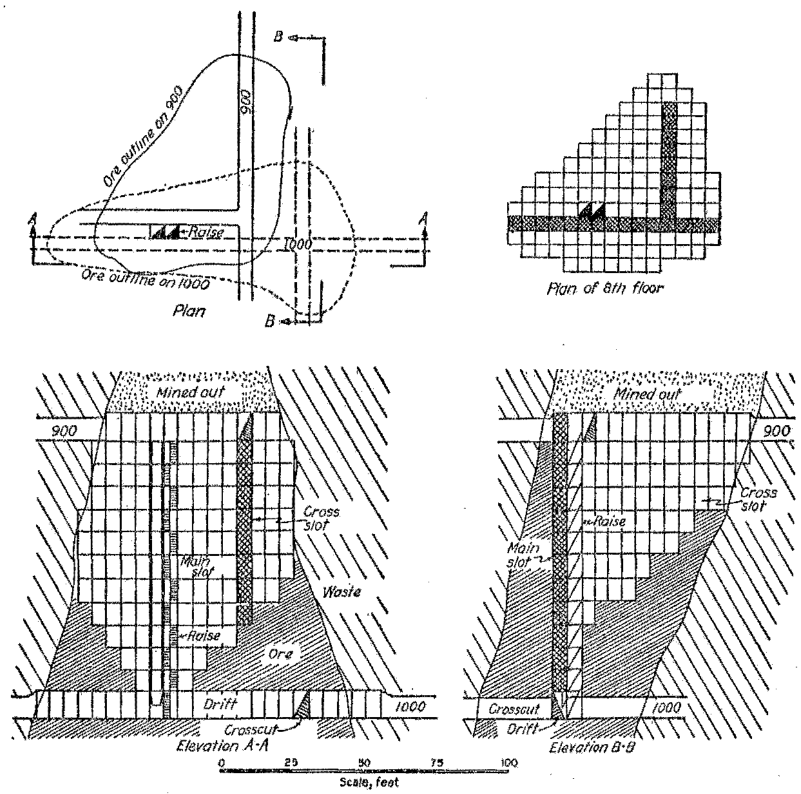


FIGURE 87.—"Slot method" of stoping, Copper Queen mine.

Stope development for this system of mining (that is, drifting, crosscutting, and original raises to the level above) costs 30 cents a ton; the general development charge is 60 cents a ton of ore. Usually about 14 tons of ore are obtained for each foot of development work. Slightly more timber is used with top slicing than with filled sets. The average cost of timber for either the top-slice or the square-set method is 40 cents per ton.

MIAMI

Top slicing as used at Miami has been described by Deane.[89] The ore was blocked out into mining units 200 to 250 feet square. Haulage drifts were on 50-foot centers and 150 feet apart vertically. Two intermediate sublevels 50 feet apart vertically were used to facilitate building chutes and to ventilate the mine. Raises for receiving the ore were put up along the haulage drifts on 50-foot centers.

Stoping began at the limits of the block from the ends of two drifts run at right angles to each other from a raise in the center of the block. Slices were 10 feet thick; as soon as one was advanced a few feet, a second and then a third were begun. The method had the advantage of a large number of working faces in a concentrated area.

OLD DOMINION

About 50 tons per day were being stoped by top slicing at the Old Dominion mine in 1930.[90] The method was used where the grade of the ore was so high that clean mining was necessary and where it was permissible to cave the overlying ground. It was a combination of flat and inclined top slicing and was used successfully in ore bodies 10 to 80 feet wide.

A flat section 20 feet wide was run across the ore body from footwall to hanging wall. From the flat slice inclined slices were run with the strike of the vein. A complete stoping section extended 70 feet along the strike of the vein; 11- and 9-foot stulls were used respectively in the flat and inclined sections. A stope was worked through two two-compartment raises 10 feet apart on the footwall in the flat slice. Narrow slots were run from each raise to the hanging wall in advance of each slice to bring the ore into the chute compartments of the raises. The slope of the incline was 33° and that of the bottom of the slots 36°.

Unless the ground was very heavy the section was blasted down as a unit. The timbering at apexes between stopes had to be joined so that waste could not come through at these points. Figure 88 shows the details of the method.

SUBLEVEL CAVING

Sublevel caving is a modification of top slicing and a forerunner of undercut block caving. Slices are cut and timbered as in top slicing, but instead of the ore being taken out to the mat above a layer is left between the slice and back. After the cut is completed the ore above is allowed to cave, starting at the end of the slice and retreating toward the entrances. Top slicing and sublevel caving usually are applicable to similar conditions; the former may be used where the capping and mat will come down and immediately fill the slice when blasted down and the latter where the capping and mat tend to arch over and temporarily hang up (a dangerous condition for top slicing).

Sublevel caving has been used most in the iron mines of the Gogebic range in Michigan. The method has been used in the copper mines

[89] Deane, E. G., The Block Method of Top Slicing of the Miami Copper Co.: Trans. Am. Inst. Min. Eng., vol. 55, 1917, pp. 240, 244.

[90] Shoemaker, A. H., Mining Methods at the Old Dominion Mine, Globe, Ariz.: Inf. Circ. 6237, Bureau of Mines, 1930, 21 pp.

at Cananea and Morenci and in 1931 was being employed at the Ruth mine of the Nevada Consolidated Copper Co. at Ruth, Nev.[91]

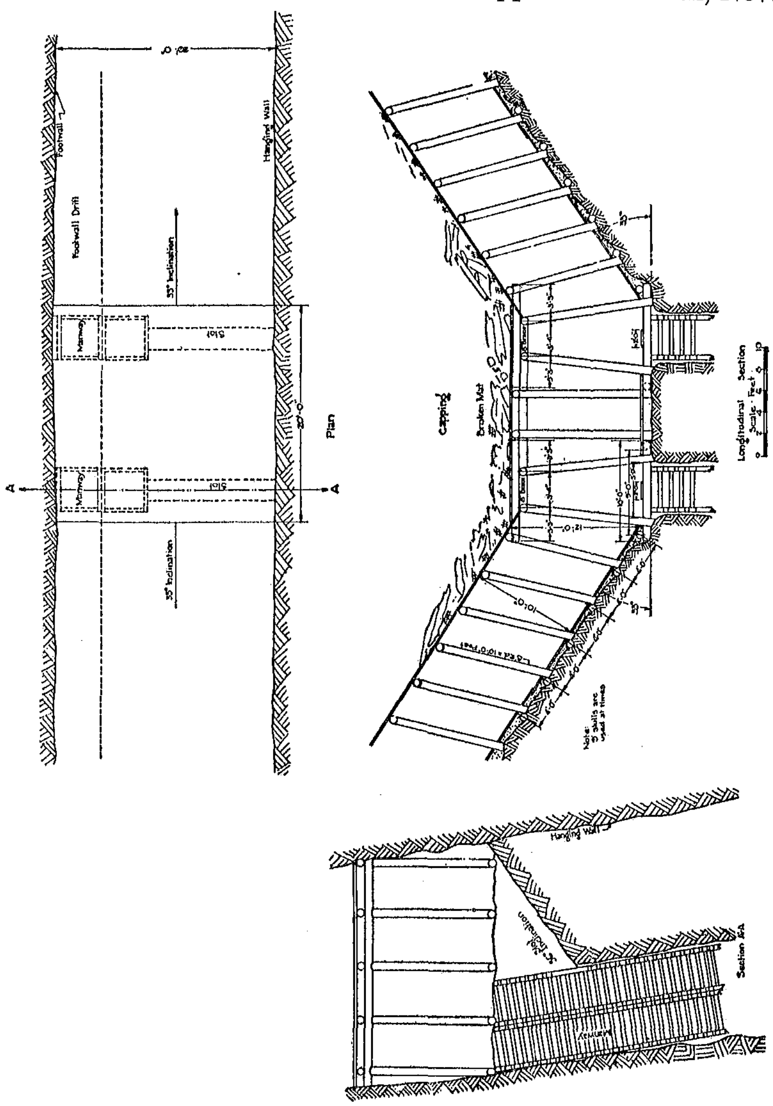


FIGURE 88.—Inclined top slicing, Old Dominion mine.

RUTH

The method is used at Ruth for mining a high-grade ore body most of which lies between levels 6 and 7. The ore body is lenticular and has a length of 950 feet, an average thickness of 60 feet, and an average width of 200 feet. In former years it was mined by square-setting and filling. Sections three sets wide and the width of the ore body were carried upward from a timbered drift under the ore body. It was difficult to keep the stope sections open three sets wide and two

[91] Gardner, E. D., and Vanderburg, W. O., Square-set System of Mining: Inf. Circ. 6691, Bureau of Mines, 1933, p. 45.

sets high owing to the heavy ground. The waste required for filling could not always be obtained conveniently; another objection to the system was the large amount of timbering required. To overcome these disadvantages the square-set method was abandoned in favor of a sublevel caving method, which has proved more satisfactory.

In preparation for mining by sublevel caving a drift is run the length of the ore body, as in the square-set method. Three-compartment raises are driven on 17-foot centers from the drift to the top of the ore body. These raises are at right angles to the haulage drift, and the center compartment, used as a manway and for supplies, is directly over it. Crosscuts extend from the top of the raises to the ore limits at right angles to the drift.

The 11⅓-foot pillar of ore between consecutive crosscuts at the top of the ore body is mined by starting at the ends of the drifts and retreating toward the chutes. Square-sets of 8- by 8-inch timber are placed as the pillar is mined. Scrapers operating on a floor of 2- by 12-inch plank laid on the sills are used in driving the crosscuts and mining the pillars. Cross scraping is employed to drag the ore from the pillars into the crosscut runways. Old timbers and sawmill trimmings form a mat beneath the capping, thus preventing dilution when the ore below is extracted. Finally, the square-set posts in each slice are broken with small charges of dynamite placed in auger holes.

After the first slice is extracted crosscuts are run about 16 feet below the mat, and about 8 feet of ore is left over the square-sets, as shown in the cross section. (See fig. 89.) Mining progresses as in the first slice; in addition the 8 feet of ore over the back of the sets is taken down at the same time. No timber is used in mining the top ore; timber slides are used to guide the ore into the drift runways.

In this method of mining the area remaining open at any one time is small; the amount of timber used is only about half of that required with square-setting or top slicing; mining is rapid, as several places can be worked on the same floor without interference; waste fill is unnecessary; and hand shoveling is practically eliminated. The mining procedure with the sublevel caving system is more systematic, as the general scheme does not have to be varied as in square-setting to suit local conditions arising from heavy ground and the need of waste for filling. If the ground becomes heavy the sets can be blasted down as soon as the ore is mined, and the mat and capping can be allowed to settle over the unmined ore.

LEACHING IN PLACE

Leaching in place is not strictly a mining method but usually is considered to be a mining operation. It consists of percolating leaching solutions through unmined ore. Usually the ore to which this method is applied has been fractured by caving or settling and is of too low a grade to be removed and treated economically. The leaching solutions usually comprise mine-drainage water, but acidified creek water may be used. They are collected in workings below the ore, and the copper is precipitated on scrap iron in tanks.

The drainage water at most copper mines contains soluble copper salts. At Butte, Bisbee, Jerome, and other districts the copper is precipitated from the water either at the surface or underground and recovered. At the Utah Copper drainage water from the mine and

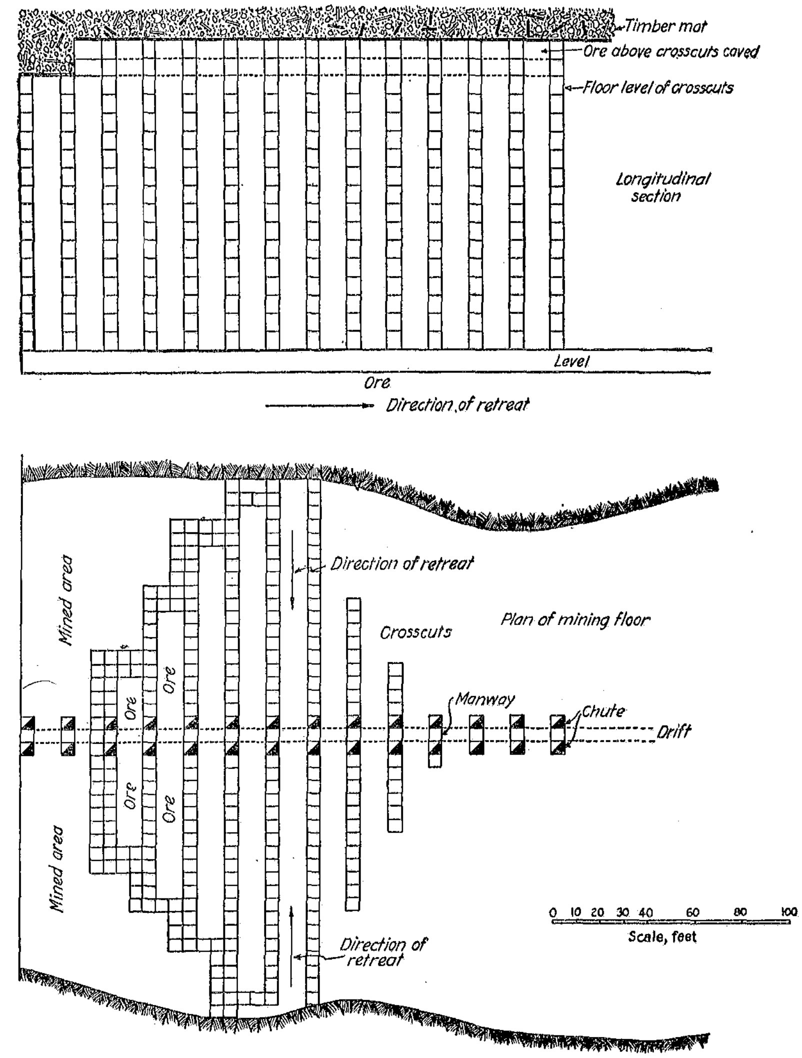


FIGURE 89.—Method of sublevel caving, Ruth mine.

run-off water through waste dumps in gulches are treated in a plant at the mouth of the canyon for the recovery of the contained copper.

OHIO COPPER

Leaching copper from ore in place has been carried on more systematically and on a larger scale at the mine of the Ohio Copper Co., at Bingham, than elsewhere on the continent. Acidified creek water was run onto ground that previously had been fractured by caving operations. The water percolated through the rock and was collected in an adit at the bottom of the mine, where the copper in solution was precipitated upon scrap iron.[92]

Leaching operations began in 1922. During 1926 the output was 4,964,000 pounds of copper; in 1929, 2,215,000 pounds; in 1930, 2,048,000 pounds; in 1931, 659,000 pounds; and in 1934, 375,000 pounds. There was no production in 1932 or 1933. The cost was 14.0 cents per pound in 1929 and 13.3 cents in 1930.

COPPER QUEEN

Ore in old stopes and old fills is leached at the Copper Queen mine, Bisbee, Ariz. In 1929 about 1,000,000 pounds and in 1930 about 600,000 pounds of copper were obtained by leaching underground at the Holbrook plant of the Copper Queen mine. Part of this copper, however, was precipitated from the regular mine-drainage water. A substantial quantity of copper was also obtained during these years by leaching in place in the Junction mine, now a part of the Copper Queen group.

CANANEA

At Cananea, Mexico, underground leaching began in 1923. By 1929 about 2,000,000 pounds of copper were produced annually in this manner at a cost of slightly over 4 cents a pound. The material leached is old stope filling and low-grade ore in abandoned shrinkage stopes and in the crushed and caved ground surrounding and overlying worked-out top-slice stopes and other caved areas. The leaching water is pumped from the leaching plant, together with fresh make-up water, to tanks on the hillside and thence to desired points on the surface. A form of prospecting consists of running streams of water onto certain surface areas then noting where the water comes through into the underground workings and determining the copper content. In some places drifts are driven through old fills to guide and redistribute the water. The water is collected on an adit level whence it flows by gravity to the precipitation plant. In the first half of 1929 about 51,000,000 gallons of water were used to leach areas at the Oversight mine. Return water amounted to 39,000,000 gallons carrying 2.7 grams of copper to the liter. In addition, the copper-bearing water was pumped from two nearby mines.

The precipitation plant comprises a series of long, narrow, concrete vats filled with baled tin cans and scrap iron. Air for agitation is introduced through rubber hose lying in the bottom of the vats. The precipitate is sluiced out periodically through gates in the ends of the vats. Leaching and precipitation required about 38 man-shifts per 24 hours; copper production was about 140 pounds per man-shift; and iron consumption was 1.1 pounds per pound of copper.

UTAH COPPER

Copper-bearing water at the Utah Copper mine is conveyed to the precipitation plant in an 18-inch redwood pipe. Water from the

[92] Oldright, G. L., Leaching of Caved Areas in the Ohio Copper Mine, Bingham, Utah: Min. and Met., October 1923, vol. 4.
Wormster, F. E., Progress in Leaching the Ohio Copper Mine: Eng. and Min. Jour., vol. 118, Sept. 26, 1924, p. 124.
Anderson, A. E., and Cameron, F. K., Recovery of Copper by Leaching, Ohio Copper Co. of Utah: Trans. Am. Inst. Min. and Met. Eng., vol. 73, p. 31, 1926.

various gulches is collected in branch lines. The precipitation plant at Copperton is housed in a building 49 feet wide by 976 feet long.[93] The plant consists of four launders 4 feet wide by 4 feet deep with false bottoms 2½ feet from the top to permit the precipitate to fall through. They are divided into 20-foot sections. The copper mined from the precipitate boxes is flushed into a settling tank 960 feet long by 7 feet high by 24 feet wide. The tank is divided into 40-foot compartments. The detinned scrap used for precipitating the copper is handled mechanically by a crane or an overhead track. The resulting 90-percent copper is handled by the trainload. Recovery of copper from the water in 1929 was 95 percent.

BRITANNIA

Rain water from a 40-acre catchment basin percolates through a large caved zone of low-grade sulphides in the upper Fairview ore body [94] and leaches out considerable copper. The water is sampled periodically at various points where it enters the mine workings and is routed according to its copper content either to waste or to a flume on the 2,200-foot level leading to a precipitation plant at the portal. The flow varies with the seasons (the annual rainfall is 100 inches) and ranges from 6,000 to 40,000 gallons per hour. The copper content of the water ranges from 0.6 to 2.0 grams per liter and tends to increase with greater flow.

The precipitation plant consists of 37 cedar tanks each with two 5- by 5-foot compartments. The tanks have sloping floors, above which are wooden gratings. Shredded scrap iron and tin cans are used to precipitate the copper. After the pregnant solution passes through a settling tank it is agitated in the tanks with low-pressure air introduced through perforated lead pipes. Ninety-five percent of the copper is precipitated in 55 minutes of exposure. The copper sludge is periodically flushed from the tanks, dried, sacked, and shipped to the smelter; it averages 67 percent of copper. An average of 1,200,000 pounds is produced annually at a cost of 2.14 cents per pound.

COPPER-MINING COSTS

Conditions vary so at different copper mines that relative mining costs can seldom be given on a strictly comparable basis. Moreover, very few companies figure unit costs in the same manner.

FACTORS AFFECTING COSTS

Among the many factors that influence mining costs, the most important are the mining method used, scale of operations, natural underground conditions, labor, and management. Other factors are prices of supplies and power, modernity of operations, climate, topography, taxes, and Government regulations.

MINING METHOD USED

Physical conditions largely determine the choice of a method. The principal mining methods used in copper mines, arranged in order of

[93] Martin, Gail, A Modern Scrap-iron Precipitation Plant; Eng. and Min. Jour., vol. 128, September 1929, pp. 467–469.

[94] Brennan, C. V., Mining Operations at the Property of the Britannia Beach Mining & Smelting Co., Ltd., Britannia Beach, British Columbia: Inf. Circ. 6815, Bureau of Mines, 1935, p. 26.

increasing average costs per ton, are open-cut mining, undercut block caving, open stoping, shrinkage stoping, cut-and-fill stoping, and square-setting. The cost of mining by glory holes is usually a little less than by open stopes in similar ground. Top-slicing costs are probably between those for cut-and-fill and square-setting. Sublevel-caving costs are less than top-slicing costs under comparable conditions.

SCALE OF OPERATIONS

Fixed charges and overhead are, of course, distributed to the tonnage mined. As the tonnage increases these are relatively smaller; thus the over-all cost per ton is lower. On the other hand, tonnage demand occasionally has an adverse influence on mining practices; ore may have to be extracted as opened up, which may not be by the most economical method.

The unit costs of many phases of mining are less if the work is performed on a large scale. For example, powerful and efficient hoisting equipment with resultant lower costs can be installed where a large daily tonnage is to be handled but would not be justified at a small mine. Moreover, new shafts can be sunk in advantageous positions for mining large known ore bodies but would not be economical for mining small ones with a small daily tonnage.

NATURAL CONDITIONS UNDERGROUND

Natural conditions largely determine costs at different mines that use the same method. Where ore bodies are small and scattered an excessive amount of development work must be performed per ton of ore extracted. At some mines development costs may be even higher than stoping costs. Moreover, at such mines ore may have to be transferred several times to get it to the surface.

In wet mines worked through shafts, such as the Old Dominion mine, pumping may be a serious item. On the other hand, in arid regions water for mining may have to be brought from a distance at great expense.

In mines with high rock temperatures large volumes of air must be circulated through the mine. In some mines this forms a considerable part of the mining costs. Moreover, in extremely hot and humid mines labor efficiency is low.

LABOR

In normal times most of the labor employed in North American copper mines can be classed as skilled. Many drill runners, timbermen, and other workmen are very proficient. During periods of expansion, however, many unskilled men are employed. Some companies have regular training courses for such new employees. When labor is plentiful fewer unskilled workmen need be employed, and those on the pay roll do more work than when jobs are easy to obtain. When labor is scarce a contract or bonus system of paying the labor is used at most copper mines. A bonus system appears to give, as a rule, lower unit labor costs. Some companies that do not pay bonuses, however, have relatively low labor costs, but they usually pay higher base wages than bonus-paying companies in the same district.

The attitude of workmen toward their employers affects their efficiency; however, relations between the mining companies and their employees were harmonious during 1929 and 1930. No major strike has occurred at North American copper mines from the end of the World War (November 1918) to the time of writing (July 1933).

The wages paid labor account for over half the cost per ton of mining copper ore. In normal times and in districts where living conditions are comparable, labor efficiencies correspond roughly to wage scales; unit labor costs vary little in the different districts. The most efficient workmen naturally gravitate to localities where they can make the most money. In districts where wages and living standards are relatively low, direct costs are lower for some classes of work, but usually this saving is offset to a considerable extent by higher supervisory costs.

In a few copper camps where living conditions are extremely favorable good workmen are content with lower wages than are paid at distant mines where conditions are less to their liking; under such circumstances labor costs are relatively lower.

Table 31 shows the wage scales at Butte and in Arizona and Michigan from 1922 to 1935.

TABLE 31.—*Miners' wages in Arizona, Butte, and Michigan copper mines, 1922–35, per 8-hour shift*

Dates [1]	Butte	Arizona	Michigan	Dates [1]	Butte	Arizona	Michigan
January 1922	$4.25	$4.50	$3.00	May 1930	5.25	5.45	4.75
October 1922	4.75	4.95	3.65	June 1930	4.75	4.95	4.75
March 1923	5.25	5.45	4.25	July 1930	4.75	4.95	4.25
November 1923	4.75	4.95	4.25	November 1930	4.75	4.95	3.85
October 1928	5.25	5.45	4.25	June 1931	4.75	4.95	3.65
February 1929	5.50	5.69	4.25	October 1931	4.75	4.50	3.00
March 1929	5.75	5.94	4.75	December 1931	4.25	4.50	3.00
April 1929	6.00	6.19	4.75	May 1932	4.25	4.05	--------
May 1929	5.75	5.94	4.75	September 1935	4.75	4.95	--------
June 1929	5.50	5.69	4.75				

[1] Virtually all changes were effective on the first of the month indicated.

Wages in the Tennessee copper mines formerly ranged from 50 cents to $1 less than in the Michigan copper district; early in 1932, however, miners' wages at Ducktown were $4.18 compared with $3.85 in Michigan. The copper mines in Plumas County, Calif., generally pay somewhat less than those in Arizona. For example, miners' wages in Plumas County were $5 in October 1929 and $5.69 in Arizona. The wage for miners in the Ely district of Nevada was increased from $4.95 to $5.15 in October 1935.

Canadian copper mines apparently pay about the same as Butte mines, or slightly less, although no recent data are at hand. Miners' wages were $4.72 at Sudbury and $4.75 at Anyox in 1924, when the Butte scale was $4.75. In 1932 miners were paid $4 at the Sherritt-Gordon, and the average of all mine labor at Flin Flon in March 1933 was $3.85 per shift. Wages at the copper mines in Sonora, Mexico, expressed in Mexican currency (pesos), are usually about the same as in the neighboring Arizona mines, expressed in dollars.

The rate of pay for muckers and trammers generally is 50 cents less than that for miners. The rates for timbermen and miners are the same at some mines; at others the former is 25 to 75 cents more. The rate for motormen is usually between that for miners and muckers. The rate for surface labor at underground mines ranges from muckers' pay down to $1.25 less. Skilled surface labor at underground mines receives the same pay as miners or up to $1.25 per shift more The hoistmen receive the highest pay among mine workmen; it ranges from the miners' rate to $1.50 more. At Butte the base rate is the same for miners, timbermen, muckers, trammers, nippers, etc.; pumpmen, mechanics and a few others, however, received a rate above the base. This is the only important copper camp that pays the same rate for all ordinary classifications underground. The average wage in open-cut mines is lower than that in underground mines for work of the same class.

The following table shows the wage scale (8-hour shifts) at the New Cornelia mines for 1930 and the end of 1935:

	1930	1935
Electric-shovel runners	------	$7. 70
Steam-shovel runners	$8. 15	7. 00
Cranemen	5. 70	------
Powdermen	4. 66	------
Machine-drill runners	4. 48	4. 12
Firemen	4. 37	4. 12
Jackhammer runners	3. 14	------
Drillers' and powdermen's helpers	3. 14	3. 00
Pitmen	2. 74	------
Trackmen and muckers	2. 56	2. 40

The wage scale at the United Verde open pit at the end of 1935 was as follows:

Shovel operators	$7. 70	Truck drivers (heavy)	$4. 95
Oilers	4. 40	Truck dumpers	3. 30
Locomotive engineers	5. 50	Bank miners	4. 95
Switchmen	4. 40	Surface miners	4. 40
Firemen	4. 13	Nippers	3. 85
Churn drillers	5. 50	Pluggermen	3. 85
Churn-drill helpers	3. 03	Miscellaneous helpers	3. 85
Pipemen	5. 23	Common labor	2. 64
Mechanics	5. 23		

The effect of the high cost of labor and supplies in 1928 and 1929 is shown by the costs per ton at a western copper mine for 1925 to 1930 as follows: 1925, $5.17; 1926, $5.29; 1927, $5.90; 1928, $6.00; 1929, $6.55; 1930, $4.92. Mining conditions remained the same, and no changes were made in methods or practice throughout the period.

Table 32 gives the man-hours per ton of ore mined and per pound of copper produced in 1929 at United States copper mines having an output valued at $100,000 or more. The output per man-hour has increased considerably since 1929, especially at the open-cut mines. The number of men employed and man-days of labor at copper mines, mills, and smelters in the United States, 1929–34, are shown in table 33.

TABLE 32.—*Man-hours per ton of ore mined and per pound of copper produced at all copper mines in the United States with output valued at over $100,000 in 1929* [1]

Mining method	Number of mines	Man-hours worked	Ore mined, tons	Percent of total	Yield, pounds per ton	Man-hours: Per ton of ore mined	Man-hours: Per pound of copper
Open-cut	7	9, 139, 610	31, 563, 645	48.0	20.6	0.29	0.014
Block caving	7	11, 679, 624	18, 576, 196	28.4	18.6	.63	.034
Open stope	11	7, 797, 971	5, 881, 949	9.0	26.7	1.33	.050
Square-set	26	20, 919, 781	4, 216, 427	6.4	92.0	4.95	.054
Shrinkage	12	5, 584, 400	2, 296, 380	3.5	33.6	2.45	.073
Cut-and-fill	6	6, 628, 064	1, 576, 735	2.5	67.5	4.18	.062
Sublevel caving [2]	2	4, 115, 504	1, 349, 943	2.2	59.0	3.05	.051
Total	71	65, 864, 954	65, 461, 275	100.0	27.4	1.00	.036

[1] Compiled by Charles F. Jackson from Bureau of Mines records at Washington.
[2] Includes Old Dominion Morenci slide method.

TABLE 33.—*Employment at copper mines, mills, and smelters in the United States, 1929–34* [1]

Year	Number of employees: Underground	Open pit	Surface	Mills and smelters	Total
1929	25, 239	3, 276	8, 632	14, 118	51, 265
1930	18, 569	1, 985	7, 138	12, 748	40, 440
1931	12, 602	1, 807	5, 278	9, 521	29, 208
1932	5, 441	1, 171	2, 943	6, 236	15, 791
1933	4, 132	970	1, 874	5, 572	12, 548
1934	4, 605	1, 537	1, 942	6, 924	15, 008
1935	6, 203	1, 630	2, 355	9, 871	20, 059

Year	Number of man-days: Underground	Open pit	Surface	Mills and smelters	Total
1929	8, 016, 491	1, 143, 664	2, 823, 557	4, 987, 131	16, 970, 843
1930	5, 427, 527	667, 786	2, 154, 924	4, 347, 380	12, 597, 617
1931	3, 206, 678	563, 636	1, 305, 548	2, 867, 798	7, 943, 660
1932	1, 273, 440	259, 122	758, 107	1, 572, 254	3, 862, 923
1933	998, 476	214, 108	477, 204	1, 589, 500	3, 279, 288
1934	1, 016, 390	330, 801	493, 607	2, 065, 076	3, 905, 874
1935	1, 700, 447	464, 388	622, 248	3, 118, 250	5, 905, 333

[1] Compiled by W. W. Adams from Bureau of Mines records at Washington.

MANAGEMENT

The copper industry has been a model among large enterprises in North America in that there has been a free interchange of technical knowledge. Details regarding improved practices at one mine are quickly available at others, and cost figures are commonly compared. Under such conditions poor management is not likely to continue very long. Although nearly all the mining companies are well managed, differences in the ability of the directorial personnel exist and are reflected in costs. The management is responsible for general policies, relations with labor, accident prevention, choice of mining method, choice of treatment methods, and equipment. It has also the responsibility of seeing that the most economical mining practices are followed. Mistaken policies and unwise expenditures affect costs adversely.

Management is responsible for harmonious relations among the different departments of the mine. At small properties with few supervising officials this may not be a problem, but in large organizations the duties of each department should be defined clearly to prevent internal jealousies and friction. A straight line of authority is desirable; every man should have only one boss and know who his boss is.

Figure 90 shows the organization chart of the Phelps Dodge Corporation.[95]

The organization chart of the Ray mines of the Nevada Consolidated Corporation is shown in figure 91 and that of the Britannia Mining & Smelting Co., Ltd., in figure 92.

PRICE OF SUPPLIES AND POWER

That the price of supplies and power affects mining costs is self-evident. As an example, timber in the Southwest costs about double what it does in northern timbered areas. At the Copper Queen mine the cost of lumber in square-set and top-slice stopes is 40 cents a ton, which means a difference of 20 cents a ton in favor of the northern districts. Other factors, however, mask the differences in price of these items in most copper camps.

Power costs do not vary greatly; the cost of generating power at modern plants compares favorably with that of power from hydroelectric sources. The direct cost at the Miami steam plant in 1930 was 0.5 cent per kilowatt-hour compared with 0.7 to 0.75 cent at Inspiration, Ray, and Magma for power purchased from hydroelectric plants.

MODERNITY OF OPERATIONS

Newly opened mines have an advantage over old mines in that modern equipment can be installed, levels can be laid out to the best advantage, and full advantage can be taken of improved practices. At many old mines it is difficult to change from established practices. For example, at many western mines 18-inch gage track is used; this was suitable when operations were begun but may not fit present needs. However, the investment in the old haulage equipment may appear too great to warrant discarding it, hence operation continues under this handicap. Furthermore, a new enterprise is free of uneconomical inherited working practices, which once established are hard to change.

Lower costs may be expected at mines where the size and shape of the ore bodies and physical characteristics of the ore and wall rocks are known prior to laying out stopes than where they are not. If the conditions are known in advance they can be met in the most economical way.

CLIMATE

Climate influences the cost of mining. For instance, the necessity of removing snow in northern open-cut mines increases operating costs. On the other hand, the hot summers in the Southwest reduce

[95] Beckett, P. G., general manager, Phelps Dodge Corporation, A Flow Sheet of Organization: Eng. and Min. Jour., vol. 126, Oct. 27, 1928, p. 644.

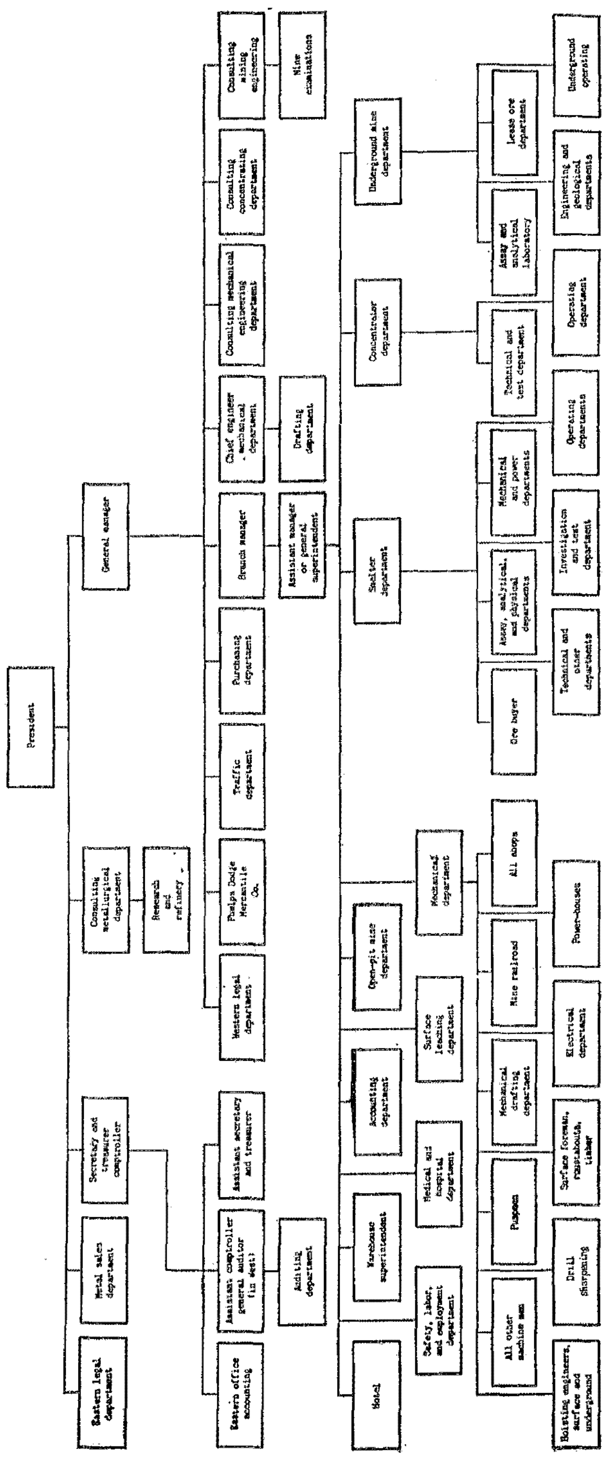


FIGURE 90.—Organization chart, Phelps Dodge Corporation.

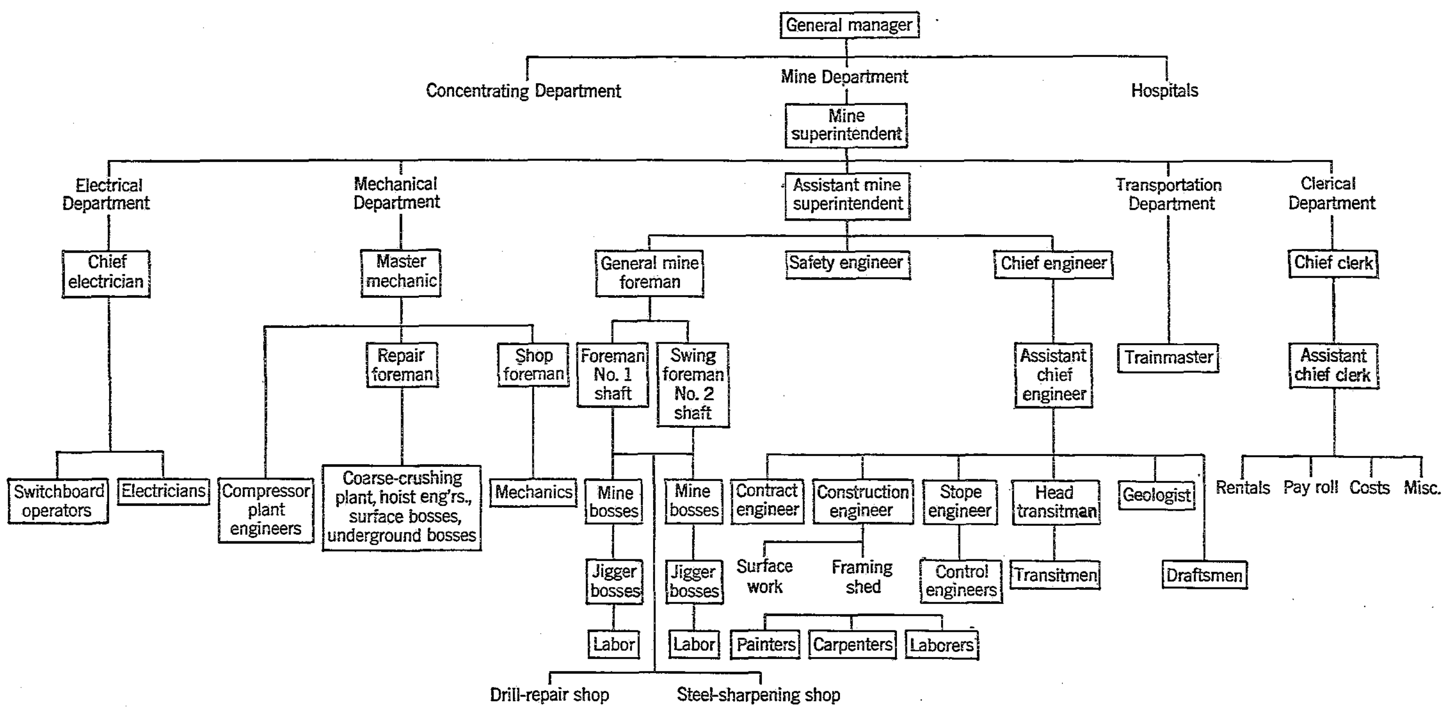


FIGURE 91.—Organization chart, Ray mines, Nevada Consolidated Copper Co.

the workmen's efficiency and may greatly increase the labor turn-over. Keeping water pipes from freezing and chipping ice from hoisting shafts during the winter in cold regions adds to over-all costs. Moreover, deep snow may make it necessary to freight all supplies into the mine in the summer, thereby tying up capital.

TOPOGRAPHY

If the topography is such that the ore can be trammed through a fairly short adit, lower costs are possible than if the ore must be hoisted. Owing to surface conditions in some places, a shaft cannot be located at the most advantageous spot as regards underground workings. This may increase tramming costs. In surface workings the topography affects haulage costs.

TAXES AND GOVERNMENT REGULATIONS

Federal and local taxation is an important item of cost in the copper industry. Local taxes are usually included in the calculated cost per ton of ore and per pound of metal produced; the practice, however, is not uniform. Most companies report costs exclusive of Federal income taxes.

The current cost of property taxes per ton of ore or per pound of copper produced of course depends upon the scale of operation. Tax rates are not the same in the copper-producing countries of North America or uniform in the different States or Provinces of the countries. Copper companies in localities where the rates are low have an advantage over those in districts where taxes are high.

Workmen's compensation laws and Government safety codes are in effect in nearly all copper-mining districts in North America. Such laws or regulations when first put in force probably affected costs adversely, but after the necessary adjustments were made mining costs have been permanently increased in only a few mines.

Few restrictions are placed by law on copper mining in the United States or Canada. In Mexico, however, industry must conform to a labor code passed to protect the workmen. This code has increased copper-mining costs only slightly, as the copper companies understand the condition and have shown a disposition to obey the law, and the excellent relations between the companies and the various Government agencies has permitted equitable adjustments to be made on controversial points.

MINING COSTS AT REPRESENTATIVE MINES

OPEN-CUT

Direct costs of mining at four open-cut mines are shown in table 34. The mining cost at the Copper Flat mine of the Nevada Consolidated Corporation was $0.242 per ton in 1929; $0.282 in 1930, $0.266 in 1931, and $0.224 in 1932. Variations in cost are due principally to differences in conditions at the various mines, as previously described. Where the rock is hard to break, drilling and blasting costs are higher and the capacity of the power shovels is less owing to the coarseness of the broken material. Haulage costs are governed mainly by the topography of the deposit. Good management helps to reduce costs by supplying the most economical equipment to perform each task.

TABLE 34.—*Direct mining costs at open-cut copper mines*

	Utah Copper [1]		New Cornelia [2]	Chino [3]	United Verde [4]
Year	1928	1934	1930	1929	1928
Ore mined ... tons	16, 558, 500	4, 086, 800	2, 376, 764	2, 621, 340	731, 488
Costs per ton:					
Drilling	$0. 027	$0. 026	$0. 070	$0. 02106	$0. 139
Blasting	(5)	(5)	(5)	. 02748	(5)
Shovel operation	. 019	. 020	. 049	. 03604	. 077
Transportation	. 053	. 040	. 066	. 10794	. 086
Miscellaneous	. 018	. 022	. 025	. 01432	
Total mining	. 117	. 108	. 210	. 20684	. 302
Explosives	0. 015	0. 013	0. 025	0. 02126	[6] . 031
Power	. 003	. 005	. 004	[7] . 00517	[7] . 005
Fuel	. 007		. 030		
Labor	[8] . 038	. 026	. 104		[6] . 139
Supervision	. 006	. 009	. 021		
Supplies			. 081		[6] . 092
Ore mined per man-shift ... tons	[9] 56. 0	[9] 57. 6	42. 2		[6] 32. 6
Explosives ... pounds per ton	[9] . 118	[9] . 114	. 17		[6] 0. 195
Power ... kilowatt-hours per ton	[10] . 46	[10] 1. 29	[11] 0. 56	[7] 0. 04	

[1] Ore mining only. (See Bureau of Mines Inf. Circ. 6412.)
[2] Figures cover all material removed. (See Bureau of Mines Inf. Circ. 6666.)
[3] Figures for ore mining with electric shovels only. (See Bureau of Mines Inf. Circ. 6412.)
[4] Open-pit mine; ore mining only, except as noted. (See Bureau of Mines Inf. Circ. 6246.)
[5] Included with drilling.
[6] Includes stripping.
[7] Shovel operation only.
[8] Operating labor only.
[9] Ore and waste.
[10] For loading, haulage, and air compression.
[11] For air compression and lighting.

UNDERCUT BLOCK CAVING

Table 35 shows direct costs at five representative mines at which the ore is mined by the undercut-block-caving method. The total direct mining cost at Morenci in 1931 was $0.40; in 1932 when no development work was being done the cost was about $0.25 per ton. A daily average of 6,000 tons was being mined 6 days per week. The mining cost at Inspiration in 1932 was about $0.50 per ton. The underground mining cost at Ruth in 1929 was $0.950; in 1930, $0.936; in 1931, $0.748; and in 1932, $0.641 per ton. The mining cost of the Morenci slide method of block caving at the Old Dominion mine is shown in table 36.

There are so many variables at each mine that direct comparisons of total costs are impracticable. The principal factor governing costs is the selection of practices best suited to the conditions in each ore body.

TABLE 35.—*Mining costs at five copper mines using block-caving method*

	Morenci	Ray	Inspiration	Miami		Ruth [1]
Period	1928	1928	1928	1925–29	1930	1924
Ore mined____tons	1, 483, 984	3, 243, 159	4, 897, 646	[2] 4,139,074	6, 125, 000	850, 000
Copper in ore____pounds per ton	35	21. 3	18. 3	[3] 11. 4	[3] 10. 5	----
Ore per man in stopes____tons	62. 76	115. 61	----	----	----	----
Ore per man underground____do	11. 84	12. 13	13. 1	27	27	----
Ore per man on pay roll____do	10. 45	10. 83	----	----	24	----
Explosive per ton____pounds	. 19	0. 139	----	0. 223	----	----
Timber per ton____board feet	. 25	1. 88	----	1. 045	----	----
Power per ton____kilowatt-hours	----	----	----	1. 9	----	----
Mining costs per ton:						
Development	$0. 142	$0. 104	$0. 211	$0. 100	$0. 100	$0. 300
Stoping	. 129	. 243	. 208	. 136	. 174	. 340
Haulage	----	. 091	[4] . 144	. 053	. 043	. 086
Hoisting	----	. 020	----	. 033	. 024	. 052
General underground	----	. 057	. 012	. 032	. 025	. 393
Engineering and sampling	----	----	----	. 014	. 010	----
Mine surface	[5] . 045	----	. 008	. 020	. 014	----
Mine accident	----	----	----	. 011	. 006	----
General	----	. 102	----	----	----	} . 098
Supervision	. 040	----	----	----	----	}
	[6] 0. 356	[7] 0. 677	[7] 0. 583	[8] 0. 399	[8] 0. 396	1. 249
Per pound of copper	. 010	. 033	. 032	. 035	. 036	----

[1] Parsons, A. B., The Rejuvenation of Nevada Consolidated: Eng. and Min. Jour., vol. 120, Nov. 7, 1925, p. 729.
[2] Annual average, Oct. 1, 1925 to Sept. 30, 1929; total ore, 16,556,296 tons.
[3] Net.
[4] Includes hoisting.
[5] Compressed air, steel, water, and drill expense.
[6] Extraction cost only.
[7] Partial underground cost.
[8] Total mining cost.

TABLE 36.—*Cost of mining by Morenci slide method, Old Dominion mine*

Period____9 months	1931
Ore mined____tons	300, 000
Cost per ton of ore:	
Exploration and level development	$0. 171
Stope preparation and stope development	. 160
Extracting ore (ore into chutes)	. 439
Underground repairs	. 047
Tramming and haulage (ore and waste)	. 309
Hoisting (ore and waste)	. 256
Pumping and drainage	. 179
Sanitation	. 018
Ventilation	. 119
Assaying and sampling	. 038
Mine-department expense	. 210
Diamond drilling	. 100
	2. 046

OPEN STOPES

Table 37 shows the cost of mining by open-stope methods in the Michigan district, at Ducktown, Tenn., and at the Sherritt-Gordon mine, together with amounts of labor and supplies. The breaking cost at the Osceola mine in January and February 1931 was $0.456 per ton. Costs at other mines using open stopes are not available.

TABLE 37.—*Cost of mining by open-stope methods*

	Michigan copper mines			Calumet & Hecla Conglomerate [1]	Tennessee Copper Co.[2]	Ducktown Chemical Co. [3]	Sherritt-Gordon
Year	1927	1927	1927	1930	1928	1928	1931–32
Ore mined ... tons	[4] 681,600	[5] 791,961	[6] 1,151,557	872,834	473,292	108,519	372,225
Costs per ton of ore:							
Development	$0.217	$0.197	$0.379	-----	$0.229	$0.355	$0.327
Stoping	[7] 2.259	[7] .870	[7] .920	-----	.332	.890	.746
Haulage	(.647)	(.372)	(.303)	-----	.386	.369	[8] .162
Hoisting	.579	.148	.176	-----	-----	-----	.069
Pumping	.399	.008	.048	-----	-----	-----	-----
Surface	.082	.071	.081	-----	.043	.060	-----
General	-----	-----	-----	-----	.145	.154	-----
	3.536	1.294	1.554	-----	1.135	1.828	1.304
Labor	2.218	0.826	0.934	-----	0.654	1.082	-----
Supervision	-----	-----	-----	-----	.090	.087	-----
Supplies	1.136	.403	.527	-----	.356	.443	-----
Power	.182	.065	.093	-----	.035	.216	-----
	3.536	1.294	1.554	-----	1.135	1.828	-----
Ore mined per 8-hour manshift (tons):							
Stopes	-----	-----	-----	6.33	13.20	3.81	-----
Total mining	2.61	6.62	6.39	3.06	6.08	3.58	-----
Timber per ton, total mining:							
Board feet	8.380	0.341	5.768	-----	0.610	-----	-----
Linear feet	-----	-----	-----	6.19	-----	-----	-----
Explosives (pounds per ton):							
Stopes	0.711	0.991	0.978	-----	-----	0.420	-----
Total mining	3.411	4.431	2.744	0.75	0.755	0.782	-----
Power ... kilowatt-hours per ton	22.970	0.551	2.670	[9] 4.942	7.92	10.50	-----
Coal ... pounds per ton	100.98	42.34	63.37	-----	-----	81.10	-----

1 Bureau of Mines Information Circ. 6526.
2 Bureau of Mines Information Circ. 6149.
3 Bureau of Mines Information Circ. 6397.
4 Large, square open stopes supported by stulls (Bureau of Mines Bull. 306).
5 Large open stopes with pillar support (Bureau of Mines Bull. 306).
6 Long, narrow open stopes supported by narrow pillars (Bureau of Mines Bull. 306).
7 Includes tramming, compressor, and drill expense.
8 Includes mucking.
9 Air compression only.

GLORY HOLE

The cost of glory-hole mining at the Copper Queen mine from June 1929 to March 1931 is given in table 38 and the cost of mining by a combination glory-hole and shrinkage method at Britannia in table 50. No other data are available on costs of copper mining by this method.

TABLE 38.—*Cost per ton of glory-hole mining at the Copper Queen mine, June 1929 to March 1931* [1]

	Glory hole	Grizzlies	Total
Stoping:			
Labor	$0.033	$0.056	$0.089
Supplies	.008	.013	.021
Drills and tools	.012	.016	.028
Timber		.001	.001
Explosives	.034	.018	.052
	.087	.104	.191
Exploration and development			.017
Underground repairs			.014
Tramming			.082
Hoisting			.116
Drainage			.029
Sanitation			.005
Ventilation			.003
Sampling and assaying			.015
Mine-department expense			.045
			.517
Proportion of general departments			.027
Transportation to concentrator			.064
Total			.608

[1] Data supplied by J. P. Hodgson, manager, Copper Queen Branch, Phelps Dodge Corporation. Ore from glory holes, 1,357,441 dry tons; through grizzlies, 1,415,617 dry tons.

Tons per man-shift:	
Glory holes	120.11
Grizzlies	98.39
Total	54.09

TOP SLICING

Table 39 shows mining costs at the Colorada mine at Cananea in March 1931; at this time more ore was being mined by top slicing than by any other method. The table also shows comparative costs of top slicing, shrinkage, and horizontal cut-and-fill stoping.[96] Comparative costs of top slicing, square-setting, and cut-and-fill stoping at the Copper Queen and Pilares mines are shown in tables 41 and 42.

TABLE 39.—*Costs at Colorada mine, Cananea, 1931*

[Ore produced in 1931, 384,397 tons]

UNDERGROUND COSTS PER TON OF ORE PRODUCED [1]

	Labor	Supervision	Compressed air, drills, and steel	Power	Explosives	Timber	Other expenses	Total
Development (drifts and raises in ore and waste)	$0.0445	$0.0021	$0.0172		$0.0209	$0.0087	$0.0157	$0.1091
Mining	.2757	.1244	.1093		.1406	.2219	.0654	.9373
Transportation	.1552			$0.0216			.0281	.2049
General underground	.1449			.0164			.0142	.1755
Surface expense	.1084			.0013			.0406	.1503
	.7287	.1265	.1265	.0393	.1615	.2306	.1640	1.5771

[1] These costs exclude mine departmental expenses, such as superintendence, engineering, sampling, watchmen, and time office.

[96] Information furnished by C. E. Weed, general manager, Cananea Consolidated Copper Co.

STOPING COSTS PER TON OF ORE MINED

	Shrinkage	Top-slice	Horizontal cut-and-fill	Combined cost
Labor	$0.1829	$0.3883	$0.2183	$0.2900
Explosives	.1122	.1190	.0951	.1095
Timber	.0036	.2790	.1068	.1662
Other supplies	.0032	.0066	.0049	.0054
Drills, etc	.0279	.0280	.0281	.0280
Air	.0608	.0605	.0570	.0597
Breaking, hauling, and spreading waste	.0734	.1103	.1867	.1291
	.4640	.9917	.6969	.7879
Ore mined:				
Total	74,307	174,874	128,830	378,011
Per man-shift	12.13	5.86	7.82	7.21

SQUARE-SETTING

Table 40 gives cost data at representative copper mines where square-setting is used. Comparative data on square-setting and other methods at the Copper Queen and Pilares mines are given in tables 41 and 42, respectively. Table 43 shows the average cost of square-setting and top slicing at the Copper Queen mine in 1930, exclusive of supervision and general charges. An average of 13,000 tons was mined per month. The ore mined per man with straight square-set mining was 7.0 tons; with top slicing, 6.8 tons. Table 44 shows mining costs at the United Verde Extension mine in 1929, 1930, and 1931. The output per man-shift from stoping operations was gradually reduced from 7 tons in 1924 to 3 in 1932; 3½ tons per man-shift for the total mine force was produced in 1932.

The total mining cost at the Denn mine, 1927 to 1931, was $6.01 a ton.

In 1932 the contract price for breaking and timbering in square-set stopes at Butte was 4½ cents and that for shoveling 3 cents per cubic foot.

TABLE 40.—*Cost data at representative mines using the square-set method*

	Anaconda	Old Dominion	United Verde	Copper Queen	United Verde Extension
Year	--------	1928	1929	[1] 1930	1928
Ore mined ... tons	--------	350,661	--------	725,409	275,212
Ore per man-shift (tons):					
Stopes	5.05	[2] 5.70	[2] 2.45	6.8	4.8
Total mining	1.82	[3] 1.89	--------	--------	2.0
Timber ... board feet per ton	22.05	[4] 4.8	[2] 12.52	14.5	16.8
Explosives (pound per ton):					
Stopes	--------	[5] 0.28	[2] 0.60	0.67	0.39
Total mining	1.07	--------	1.05	--------	--------
Ore per foot of development ... tons	[6] 18.0 [7] 14.2	--------	28.7	14	--------
Cost per ton:					
Stopes	--------	--------	--------	$2.17	$1.83
Total mining	--------	--------	--------	--------	$4.29

[1] Last 6 months.
[2] Square-set stopes only.
[3] All methods including development.
[4] Stoping only.
[5] All methods excluding development.
[6] 1927.
[7] 1926.

TABLE 41.—*Relative costs of labor and explosives in representative stopes in the Limestone Division, Copper Queen mine, during the last 6 months of 1930*

Method used	Ore mined				Explosives (pounds)	Cost			
	Total		Per man-shift			Labor	Explosives	Total	Per ton
	Tons	Cubic feet	Tons	Cubic feet					
Square-set:									
Hard ground	15, 245	133, 200	7. 7	67. 0	8, 975	$14, 203. 73	$2, 087. 47	$16, 291. 20	$1. 07
Medium ground	17, 508	191, 847	7. 8	85. 6	9, 656	16, 498. 91	2, 126. 54	18, 625. 45	1. 06
Soft ground	3, 189	40, 050	5. 5	69. 7	7, 780	3, 560. 76	461. 64	4, 022. 40	1. 26
Top-slice:									
Hard ground	13, 087	133, 418	5. 9	60. 4	8, 814	14, 658. 84	2, 028. 23	16, 687. 07	1. 27
Medium ground	8, 887	92, 483	8. 2	86. 0	3, 999	7, 270. 45	997. 43	8, 267. 88	. 93
Cut-and-fill: Hard ground	2, 181	20, 120	6. 2	57. 2	1, 460	2, 272. 58	400. 49	2, 673. 07	1. 23

TABLE 42.—*Comparative data on production, costs, and supplies used with various systems at the Pilares mine*

System	Percent of total mined by each method	Stope cost per ton	Timber per ton (board feet)	Powder per ton (pound)	Ore mined per man-shift (tons)
Horizontal cut-and-fill	59. 4	$1. 07	3. 6	0. 415	4. 30
Rill cut-and-fill	17. 6	. 91	4. 2	. 345	5. 02
Square-set	17. 7	1. 62	13. 6	. 325	3. 36
Shrinkage	1. 6	. 64	. 15	. 485	6. 88
Top-slice	3. 7	1. 55	11. 5	. 286	3. 95

TABLE 43.—*Cost of square-setting and top-slicing at Copper Queen mine, 1930*

Labor in stopes	$1. 07
Explosives, surface	. 10
Explosives, handling to stopes	. 03
Timber, surface	. 30
Timber, handling to stopes	. 10
Compressed air, drills, steel, etc	. 37
Stope preparation	. 30
General development	. 60
	2. 87

TABLE 44.—*Mining costs per ton at the United Verde Extension mine, 1929–31*

Year	1929	1930	1931
Tons mined (wet weight)	379, 538	318, 055	192, 720
Prospecting and development	$0. 404	$0. 514	$0. 531
Stoping	1. 956	1. 742	1. 276
Repairs (supplies and labor)	. 311	. 341	. 243
Ventilation	. 046	. 045	. 042
Haulage	. 436	. 383	. 282
Hoisting	. 125	. 119	. 084
Pumping and drainage	. 021	. 027	. 029
Miscellaneous underground	. 096	. 092	. 073
Drills and steel	. 128	. 094	. 060
Compressed air	. 063	. 054	. 051
Waste pit	. 029	. 020	. 020
Office and general	. 567	. 632	. 723
Total	4. 182	4. 063	3. 414

The underground costs per ton of mining 133,195 tons by square-setting at the Victoria mine of the Britannia Mining & Smelting Co. for a 12-month period (1932–33) are shown in table 45.

The manner of attacking the ore is second in importance to the character of the ore and walls in square-setting. This method requires better-trained men and more efficient supervision than some methods of mining.

TABLE 45.—*Costs per ton of square-setting at the Victoria mine, Britannia Mining & Smelting Co., 1932–33*

	Labor	Supervision	Compressed-air drills and steel	Power cost	Explosives	Timber	Filling	Total
Development	$0.062	$0.005	$0.009	$0.013	$0.022	$0.020		$0.131
Mining	.462	.030	.054	.071	.126	.113	$0.275	1.131
Tramming	.121	.013						.134
General underground expense [1]	.142	.009		.024		.037		.212
Surface expense (directly applicable to underground operation) [2]	.144	.016						.160
Total	.931	.073	.063	.108	.148	.170	.275	1.768

[1] Includes track, drainage, level maintenance, hoisting, and sampling.
[2] Includes geology, surveying, assaying, office and warehouse, and workmen's compensation.

CUT-AND-FILL STOPING

Table 46 shows the mining costs and table 47 the stoping costs at representative mines using a cut-and-fill method. Mining costs in units of labor, power, and supplies are shown in table 48. Comparative costs of three methods of direct stoping in 1928 in average stopes at the Magma mine are shown in table 49. The mining cost at Magma in 1931 was $3.36 per ton. The variation in costs is due to differences in natural conditions and in applying the method.

TABLE 46.—*Mining costs at mines using cut-and-fill method*

	Colorada	Pilares	Magma	Michigan	Champion	Matahambre
Year	1929 [1]	1929	1928	1927	1930 [2]	1928
Ore, tons	262,278	835,036	263,094	400,000	103,597	364,746
Methods used [3], percent	40 R, 14 H (46 S, Sh)	59 H, 18 R, 18 S	R, P	H, R	H, R	70 H, 30 S
Costs per ton:						
Labor	$1.102	$1.374	$3.343	[4] $1.877	$1.649	$1.433
Supervision	.171	.219	.311		.107	.155
Explosives	.279	.231	.209	[5] .621	.172	.200
Timber	.257	.319	.532		.109	.269
Power	.070	.055	.197	.191	.192	.087
Compressed air, drills, and steel	.235	.232	.326		.197	.161
Other supplies	.328	.170	.579		.102	.264
	2.441	2.600	5.497	2.689	2.528	2.569
Development	.561	.399	.736	.598	.490	.711
Stoping	1.203	1.370	2.849	[6] 1.765	1.163	1.267
Haulage	[7] .166	[7] .370	[7] .616		[7] .552	[7] .388
Hoisting				.158		
General underground	.278	.250	1.166		.078	.184
Pumping				.168	.133	
Surface	.233	.211	.130		.112	.019
	2.441	2.600	5.497	2.689	2.528	2.569

[1] First 6 months.
[2] Last 5 months.
[3] H, Horizontal cut-and-fill; R, rill cut-and-fill; S, square-set; Sh, shrinkage; G, top-slice; P, pillar.
[4] Includes supervision.
[5] All supplies.
[6] Includes haulage.
[7] Includes hoisting.

TABLE 47.—*Stoping costs by the cut-and-fill method*

Mine	Year	Method [1]	Ore (tons)	Cost: Labor and supervision	Explosives	Timber	Compressed air, drills, and steel	Other supplies	Total
United Verde	1929	H	[2] 600,000	[3] $0.85					
Do	1928	R	[2] 30,000	[3] .71					
Iron Cap	1923	R	85,211	1.20	$0.24	$0.22	$0.41	$0.82	$2.90
Campbell	1926–29	[4] R	51,970	.620	.135	.155	.215		1.125
Do	1928–29	[5] R	20,894	.70	.10	.13	.20		1.13
Do	1928–29	[6] R	280,527	.765	.12	.175	.22		1.28
Colorada	1931	H	128,830	.218	.095	.107	.085	[7] .192	.697
Do	1929 [8]	H	34,060	.500	.133	.254	.185	[7] .170	1.242
Do	1929	R	17,314	.494	.140	.284	.185	[7] .170	1.273
Pilares	1929	H	[2] 500,000						1.07
Do	1929	R	[2] 150,000						.91
Eighty-Five	1928–30	[9] R	7,034	1.26	.28	.20	.60	.03	2.37
Do	1929–30	[10] R	4,056	1.62	.17	.27	.38	.02	2.46
Michigan	1927	[11] H	400,000	1.340	[12] .405			[13] .009	1.754
Champion	1930 [14]	R	193,597	.851	.103	.061	.128	.020	1.163

[1] H, Horizontal cut-and-fill; R, rill cut-and-fill.
[2] Estimated.
[3] Min. Cong. Jour., vol. 16, April 1930, p. 320.
[4] Stope 68.
[5] Stope 90.
[6] Stopes in entire Campbell area, including minor amount from other types of stopes.
[7] Chiefly breaking, hauling, and spreading waste fill.
[8] First 6 months.
[9] Stope 10.
[10] Stope 26.
[11] Including small amount of rill stoping.
[12] All supplies.
[13] Electric power.
[14] Last 5 months.

TABLE 48.—*Mining costs in units of labor, power, and supplies at mines using cut-and-fill method*

	United Verde [1]	Magma	Colorada	Pilares	Michigan	Matahambre	Champion
Year	1929 [2]	1928	1929 [3]	1929	1927	1928	1930 [4]
Ore ... tons	[5] 900,000	263,094	262,278	835,036	400,000	364,746	193,597
Methods used [6] ... percent	61 H, 3 R, 27 S, 8 T	R, P	40 R, 14 H, 46 S, Sh	59 H, 18 R, 18 S	H, R	70 H, 30 S	R
Labor (man-hours per ton):							
Breaking	0.235	[7] 2.208	0.876			1.07	0.73
Timbering	.466		.799			[8] .78	.30
Mucking	.623		.990			1.43	.92
Haulage and hoisting	.238	.639	.494			.64	.86
Supervision	.126	.479	.297			.11	.12
General	1.037	.934	.163			.28	.11
	2.725	4.26	3.619			4.31	3.04
Tons per man-shift underground	2.92	1.88	2.21	2.11	2.68	1.86	2.64
Labor ... percent of total cost		56	52			60	66
Explosives ... pounds per ton	.84	.90	1.37	.38		1.02	1.3
Timber ... board feet per ton	7.84	19.00	7.19	5.71	3.22	[9] 1.15	4.3
Power (kilowatt-hour per ton):							
Compressed air	10.15	14.16	6.0			8.00	12.4
Haulage			1.0				
Hoisting	2.35	8.92	3.4			3.00	4.00
Pumping	.26	5.54	.5			1.79	4.7
Ventilation	3.07	4.91	.2			1.22	[10] 1.0
Other	.61	1.08	.2				
		34.61	11.3	6.2	[11] 8.18	14.01	22.1

[1] United Verde costs apply to cut-and-fill mining only but have been refigured to include a share of development.
[2] First 10 months.
[3] First 6 months.
[4] Last 5 months.
[5] Estimated.
[6] H. Horizontal cut-and-fill; R, rill cut-and-fill; S, square-set; T, top-slice; Sh, shrinkage; P, pillar.
[7] Includes timbering and mucking.
[8] Includes timbering.
[9] Plus considerable round timber.
[10] Includes other supplies.
[11] Plus 70 pounds of coal for steam power.

TABLE 49.—*Comparative direct costs of different methods in average representative stopes, Magma, 1928*

	Square-set	Inclined cut-and-fill	Mitchell slice
Tons	5,246	18,556	43,035
Labor:			
Mining	$0.46	$0.32	$0.26
Mucking	.05	.01	.05
Timbering	.53	.40	.38
Bonus	.08	.17	.22
Total labor	1.12	.90	.91
Supplies:			
Waste fill	.13	.03	.23
Explosives	.06	.08	.04
Timber, incidentals	.48	.34	.28
Total supplies	.67	.45	.55
Total stoping	1.79	1.35	1.46

SHRINKAGE STOPING

Table 50 gives costs per ton at representative copper mines using shrinkage stoping. Costs in units of labor and supplies are shown in table 51. Owing to the widely different manner of applying the method at the Beatson mine, the costs at this property are not directly comparable with those at the other mines listed.

Table 52 shows the cost of breaking pillars at the Engels mine in 1928.

The cost of producing 518,509 dry tons of ore at the Walker mine in 1930 was as follows:

Breaking ore, including shaft sinking and current development	$1.267	
Producing ore, including current development and waste	.553	
Mining cost		$1.820
Mill operating		.637
Tram (per ton of concentrate $0.784)		.050
Total		2.507

The cost of producing copper, after crediting gold and silver recovered, was 10.542 cents per pound.

TABLE 50.—*Direct mining costs at mines using shrinkage stoping*

	Michigan copper mines		Engels [1]	Beatson [2]	Eighty-Five [3]	Verde Central [4]	Colorada [5]	Britannia [6]	
								East Bluff [7]	Fairview [8]
Period	[9] 1927	[10] 1928	1928	1922–26	1928–29	1929–30	1931	([11])	([11])
Ore mined ... tons	465,100	...	303,301	2,239,821	7,332	139,203	74,307	398,664	1,268,234
Cost per ton:									
Development	$0.439	$0.218	$0.479	$0.290	$1.00	...	...	$0.080	$0.008
Stoping	[12] 1.022	.482	.760	.366	1.75	$1.661	$0.464	.235	.161
Haulage	(.393)	[13] .556	[13] .438	[13] .137	.82	[13] .580	...	.060	[14] .131
Hoisting	.201	...	...	...	.33	...	...	.000	.000
Pumping	.076	...	...	...	.66	...	...	.000	.000
Surface	.096	...	([15])	...	.31	.033	...	[16] .070	[16] .035
General	([17])	.240	([15])	...	.62	.111	...	.069	.035
	1.834	1.496	1.677	.793	5.49	[18] 2.385	...	.514	.371
Labor	[19] 1.065	.838	1.028	.347	2.47	1.268	[19] .183	[20] .260	.277
Supervision	...	...	.070	...	...	.126	...	.050	.018
Supplies	.585	.558	.390	.446	.92	.915	[21] .220	.169	.025
Power	.196	.100	1.89	...	1.10	.076	.061	.035	.050
	1.846	1.496	1.677	.793	[18] 4.49	[18] 2.385	[22] .464	.514	.370

[1] Bureau of Mines Inf. Circ. 6260.
[2] Trans. Am. Inst. Min. and Met. Eng., vol. 68.
[3] Bureau of Mines Inf. Circ. 6413.
[4] Bureau of Mines Inf. Circ. 6464.
[5] Information from C. E. Weed, general manager, Cananea Consolidated Copper Co.
[6] Bureau of Mines Inf. Circ. 6815.
[7] "Britannia" shrinkage.
[8] Combined "Britannia" shrinkage and glory hole.
[9] Method "D" (Bureau of Mines Bull. 306).
[10] Bureau of Mines Inf. Circ. 6293.
[11] A 12-month period 1932–33.
[12] Includes haulage.
[13] Includes hoisting.
[14] Includes $0.054 cost of railroad haulage to mill.
[15] Not included.
[16] Includes workmen's compensation.
[17] Included in surface.
[18] Excludes development.
[19] Includes supervision.
[20] Includes $0.043 grizzly chamber expense.
[21] Includes breaking and handling waste fill.
[22] Stoping only.

TABLE 51.—*Costs in units of labor, power, and supplies at mines using shrinkage stoping*

	Michigan copper mines [1]		Engels [1]	Beatson [1]	Eighty-Five [1]	Verde Central [1]	Britannia (East Bluff)
Ore mined per 8-hour man-shift (tons):							
Stopes	...	22.22	8.0	...	...	[2] 7.6	...
Total mining	5.21	6.46	5.21	...	1.45	[3] 4.28	[4] 36.18
Timber per ton (total mining):							
Board feet	5.90	2.56	1.231	...	[5] $0.07	[3] 3.08	.52
Linear foot	...	.805	.074	...	...	...	...
Explosives (pounds per ton):							
Stopes	.950	...	.593	0.215	[5] $0.31	[3] 1.16	.61
Total mining	2.900	.983	1.379	.347	...	...	...
Power (per ton) ... kilowatt-hour	[6] 2.73	...	9.64	...	[5] $1.10	[3] 16.44	6.59

[1] See references in table 50.
[2] Breaking and timbering only.
[3] Excluding development and prospecting.
[4] Does not include development of 0.46 ton per man-shift.
[5] Cost.
[6] Plus 83.95 pounds of coal per ton.

TABLE 52.—*Cost of breaking pillars at Engels mine*

Kind of drilling	Tons in pillar	Number of holes	Length of holes	Total [1] cost of drilling	Total cost of loading and explosives	Total cost per ton of broken ore
Air drill	7,475	1,205	12,144	$2,911.00	$1,439.40	$0.5819
Diamond drill	40,178	17	2,208	8,815.00	2,052.14	.2704
Do	11,205	6	688	2,446.00	289.62	.2441

[1] Total cost of drilling includes estimated cost of sharpening and replacing worn steel, upkeep of machine drills, and cost of labor for drilling.

Costs at the Isle Royale, 1929–32, were as follows:

Year	Ore hoisted, tons	Ore milled, tons	Mining cost per ton
1929	669,049	515,025	$2.67
1930	668,700	510,262	2.44
1931	450,682	353,075	2.24
1932	74,189	58,311	2.63

The cost of shrinkage stoping is affected by virtually the same factors as that of cut-and-fill stoping.

GENERAL COSTS PER TON OF ORE AND PER POUND OF COPPER

Tables 53 to 56 show the mining, milling, general, and total costs per ton of ore and per pound of copper at selected mines. The data are taken chiefly from company annual reports; some production figures and costs per ton have been taken from Mineral Resources and Bureau of Mines information circulars on mining and milling methods and costs. The costs per pound for mining, milling, and general expenses are calculated for all mines except the Miami mine. The costs per ton of mining in these tables include prepaid development and other charges not included in the direct costs shown in some previous tables.

TABLE 53.—*Mining, milling, and general cost per ton of ore and per pound of copper at selected porphyry copper mines.*

Mine	Year	Ore treated (tons)	Copper produced (pounds)	Mining cost		Milling cost		General cost		Total cost	
				Per ton	Per pound	Per ton	Per pound	Per ton	Per pound	Per ton	Per pound
Utah Copper	1929	17,724,000	[1] 296,626,000	$0.412	$0.0247	$0.366	$0.0219	------	------	[2] $0.868	[3] $0.0665
Do	1930	9,552,000	[1] 161,139,000	.504	.0299	.461	.0273	------	------	[2] 1.069	[3] .0847
Do	1931	8,148,000	[1] 142,695,000	.396	.0226	.393	.0224	------	------	[2] .90	[3] .0699
Do	1932	3,169,000	[1] 61,962,000	.458	.0234	.590	.0302	------	------	[2] 1.658	[3] .0848
Do	1933	3,521,000	[1] 69,462,000	.411	.0208	.503	.0255	------	------	[2] 1.272	[3] .0645
Do	1934	4,087,000	[1] 78,787,000	------	------	------	------	------	------	[3] 1.848	[3] .0959
Chino	1929	3,973,000	[4] 85,000,000	[5] .314	------	.59	------	------	------	------	[6,7] .0953
Do	1930	2,514,000	[4] 56,000,000	[5] .334	------	.545	------	------	------	------	[6,8] .1085
Do	1931	2,620,000	[4] 51,000,000	[9] .267	------	.360	------	------	------	------	[6,10] .0868
Do	1932	1,165,000	[4] 22,000,000	[11] .270	------	.437	------	------	------	------	[6,12] .0964
Copper Flat	1929	3,970,000	[4] 60,000,000	[15] .242	------	.421	------	------	------	------	(14)
Do	1930	2,165,000	[4] 48,000,000	[13] .282	------	.458	------	------	------	------	(14)
Do	1931	1,820,000	[4] 38,000,000	[15] .266	------	.443	------	------	------	------	(14)
Do	1932	1,009,000	[4] 20,000,000	[16] .224	------	.481	------	------	------	------	(14)
Ruth	1929	1,144,000	[4] 48,000,000	[13] .95	------	.421	------	------	------	------	(14)
Do	1930	644,000	[4] 24,000,000	[13] .936	------	.458	------	------	------	------	(14)
Do	1931	408,000	[4] 18,000,000	[15] .748	------	.442	------	------	------	------	(14)
Do	1932	243,324	[4] 8,000,000	[16] .641	------	.481	------	------	------	------	(14)
Ray	1929	3,609,000	[4] 66,000,000	[17] .828	------	.394	------	------	------	------	(14)
Do	1930	2,327,000	[4] 36,000,000	[17] .680	------	.382	------	------	------	------	(14)
Do	1931	1,287,000	[4] 24,000,000	[18] .753	------	.400	------	------	------	------	(14)
Do	1932	582,000	[4] 10,000,000	[19] .742	------	.428	------	------	------	------	(14)
Inspiration	1929	5,774,000	107,307,000	------	------	------	------	------	------	------	[20] .1113
Do	1930	3,054,000	65,607,000	------	------	------	------	------	------	------	[20] .0967
Do	1931	2,621,000	61,368,000	------	------	------	------	------	------	------	[20] .0843
Do	1932	464,000	[21] 17,024,000	------	------	------	------	------	------	------	------
Miami	1929	5,018,000	58,841,000	.423	.036	.359	.031	$0.152	$0.013	[22] .934	[23] .1174
Do	1930	6,125,000	67,125,000	.396	.036	.297	.027	.139	.013	[22] .832	[23] .1088
Do	1931	4,439,000	50,573,000	.357	.031	.277	.024	.183	.016	[22] .818	[23] .0993
Do	1932	1,418,000	15,681,000	.340	.031	.251	.023	.143	.013	[22] .734	[23] .1441
Consolidated Coppermines	1930	1,115,000	32,612,000	1.535	.052	------	------	------	------	------	[24] .1107
Do	1931	617,000	15,076,000	------	.053	------	------	------	------	------	[24] .1097

[1] Includes copper from precipitating plant.
[2] Excludes depreciation and Federal taxes.
[3] Excludes Federal taxes, but includes depreciation and all general expenses, also gold, silver credits, and miscellaneous earnings.
[4] Estimated.
[5] Excludes depreciation and taxes, also prepaid stripping charge of $0.45.
[6] Average for Chino, Copper Flat, Ruth, and Ray; excludes Federal taxes; includes depreciation and general expenses; also includes credit for gold and silver and miscellaneous earnings.
[7] $0.0888 before Federal taxes and depreciation.
[8] $0.0959 before depreciation.
[9] Excludes depreciation, taxes, and $0.35 prepaid stripping charge.
[10] $0.0813 before depreciation.
[11] Excludes depreciation and taxes and $0.30 prepaid stripping charge.
[12] $0.0914 before depreciation and depletion.
[13] Excludes depreciation and taxes and $0.40 prepaid stripping and development charge.
[14] See Chino figure.
[15] Excludes depreciation and taxes and $0.30 prepaid stripping and development charge.
[16] Excludes depreciation and taxes and $0.25 prepaid stripping and development charge.
[17] Excludes depreciation and taxes and $0.15 prepaid development.
[18] Excludes depreciation and taxes and $0.065 prepaid development.
[19] Excludes depreciation and taxes and $0.05 prepaid development charge.
[20] Includes depreciation; excludes depletion and Federal taxes.
[21] Includes 4,997,000 pounds of copper from cleanings incident to closing mine.
[22] To concentrate on board cars at Miami; excludes depreciation.
[23] Excludes depreciation and depletion. Cost in concentrate on board cars at Miami: 1929, $0.0797; 1930, $0.0759; 1931, $0.0718.
[24] Includes depreciation and Federal taxes; also credit for gold and silver (about $0.01).

TABLE 54.—*Mining, milling, and general cost per ton of ore and per pound of copper at selected United States lode copper mines*

Mine	Year	Ore treated (tons)	Copper produced (pounds)	Mining costs		Milling costs		General cost		Total cost	
				Per ton	Per pound	Per ton	Per pound	Per ton	Per pound	Per ton	Per pound
United Verde Extension	1929		[1] 64,112,000	(2)	$0.025						[3] $0.0842
Do	1930		[1] 45,567,000		.028						[3] .0833
Do	1931	183,000	24,591,000		.027						[4] .0714
Do	1932	242,000	35,753,000								[5] .0545
Do	1933	242,000	33,212,000								[5] .0628
Do	1934	196,000	26,161,000								[5] .0657
Shattuck-Denn	1929	129,000	12,736,000								[6] .1260
Do [7]	1930	[8] 86,000	9,948,000								[6] .1206
Walker	1929	458,000	13,958,000	[9] $1.782	.059	[10] $0.776	$0.025			[11] $2.558	[12] .1080
Do	1930	519,000	14,655,000	[9] 1.820	.064	[10] .687	.024			[11] 2.507	[12] .1054
Engels	1929	395,000	11,000,000	[13] 2.565	.092	[14] .660	.024	[15] $0.516	$0.019	3.742	[16] .1799
Do [17]	1930	177,000	4,073,000	[18] 1.891	.082	[14] .767	.033	1.050	.046	3.708	[19] .2007
Verde Central	1929	93,000	4,335,000								[20] .1693
Mother Lode	1929	57,200	[21] 12,243,000	[22] 4.15							[23] .0673
Do	1930	27,500	[21] 9,647,000	[22] 12.53							[23] .1152
Calumet & Arizona [24]	1929		130,487,000								[25] .0984
Do	1930		88,840,000								[25] .1127
Cananea	1929	896,000	58,827,000								[26] .0660
Do	1930		42,425,000								[26] .1041
Do	1931		41,873,000								[26] .0513
Magma	1929	270,000	[27] 38,235,000			[28] .934	[28] .007				[28] .1029
Do	1930	252,000	[29] 31,884,000	5.46	.043						[30] .0877
Do	1931	231,862	28,840,000								[30] .0787
Do	1932	149,000	21,706,000								[31] .0833
Do	1933	145,000	19,713,000								[31] .0780
Do	1934	264,000	32,023,000								[31] .0573

[1] Includes about 5,000,000 pounds of custom ore.
[2] Mining cost in 1928 for 275,000 tons was $4.286 per ton and about $0.026 per pound.
[3] Excludes depletion; includes depreciation and taxes, also credit for gold, silver, and miscellaneous income.
[4] Excludes item of $0.036 for Federal taxes and losses sustained.
[5] Before depletion and Federal taxes.
[6] Includes depreciation and interest, also credit for gold and silver of $0.020 in 1929 and $0.016 in 1930.
[7] Stoping ceased Nov. 1, 1930.
[8] Estimated.
[9] Stoping cost $1.245 in 1929 and $1.267 in 1930.
[10] Includes aerial tramway operation of $0.906 per ton of concentrate, equivalent to $0.065 per ton of ore milled.
[11] Operating cost.
[12] Excludes depreciation, depletion, and interest; includes credit (about $0.03) for gold and silver.
[13] Includes $1.436 development and exploration and $0.080 electric haulage of ore to mill.
[14] Includes loading of concentrates.
[15] To concentrate on board cars; includes depreciation, general and fixed expense, taxes, and insurance.
[16] Includes freight and smelting (about $0.054), also credit for gold and silver (about $0.008).
[17] Mine shut down July 1930.
[18] Includes $0.652 for development and exploration and $0.086 for electric haulage of ore to mill.
[19] Includes freight and smelting (about $0.046), also credit for gold and silver (about $0.006).
[20] Excludes $0.0129 charged to depreciation and amortization, also excludes $0.002 credit for gold and silver.
[21] Shipped to smelter.
[22] Mining and milling.
[23] Excludes depreciation and depletion; includes general expense and taxes, also $0.004 credit for silver.
[24] Includes New Cornelia.
[25] Includes depreciation and taxes, also credit for gold and silver (about $0.018).
[26] Includes depreciation and taxes, also credit for gold and silver (about $0.007).
[27] Includes 1,719,000 pounds from purchased ore.
[28] First 6 months.
[29] Includes 326,000 pounds from purchased ore.
[30] Includes depreciation, general expenses, and taxes, also credit for gold and silver ($0.025 in 1929, $0.018 in 1930, and $0.0156 in 1931).
[31] Includes depreciation but does not include depletion and Federal tax.

TABLE 55.—*Mining, milling, and general cost per ton of ore and per pound of copper at selected Michigan copper mines*

Mine	Year	Ore treated (tons)	Copper produced (pounds)	Mining costs		Milling costs		General cost		Total costs	
				Per ton	Per pound	Per ton	Per pound	Per ton	Per pound	Per ton	Per pound
Calumet & Hecla	1929	[1] 3,129,000	[1] 90,319,000	------	------	------	------	------	------	------	[2] $0.1143
Do	1930	[1] 2,934,000	[1] 87,898,000	------	------	------	------	------	------	------	[2] .1056
Do	1931	[1] 2,066,000	[1] 72,367,000	------	------	------	------	------	------	------	[2] .0977
Do	1932	577,000	32,354,000	------	------	------	------	------	------	------	[2] .1064
Do	1933	493,000	[3] 33,197,000	------	------	------	------	------	------	------	.0675
Do	1934	459,000	[4] 32,847,000	------	------	------	------	------	------	------	.0708
Mohawk	1929	620,000	20,043,000	------	------	------	------	------	------	$1.859	.0732
Do	1930	472,000	13,308,000	------	------	------	------	------	------	1.949	.0872
Do	1931	444,000	13,100,000	------	------	------	------	------	------	1.650	.0704
Do	1932	217,000	8,450,000	------	------	------	------	------	------	1.434	.0516
Quincy	1929	208,000	4,459,000	------	------	------	------	------	------	------	.1997
Isle Royale	1929	515,000	10,864,000	------	------	------	------	------	------	2.67	[2] .1401
Do	1930	510,000	10,659,000	------	------	------	------	------	------	2.44	[2] .1297
Do	1931	353,000	7,731,000	------	------	------	------	------	------	2.24	[2] .1148
Do	1932	58,311	1,403,000	------	------	------	------	------	------	2.63	[2] .1093
Champion	1929	447,000	20,661,000	------	------	------	------	------	------	------	[2] .1176
Do	1930	----------	20,000,000	[5] $2.528	[5] $0.055	------	------	------	------	------	[2] .1160
Do	1931	[6] 405,000	[6] 17,721,000	------	------	------	------	------	------	------	[2] .0975
Do	1932	291,000	12,189,000	------	------	------	------	------	------	------	[2] .0865
Do	1933	204,000	12,167,000	------	------	------	------	------	------	------	[2] .0746
Do	1934	241,000	13,930,000	------	------	------	------	------	------	------	[2] .0855
All Michigan mines [7]	1929	5,154,000	152,900,000	------	.082	------	[8] $0.025	------	[9] $0.026	------	[10] .1369
Do	1930	5,100,000	151,500,000	------	.073	------	[8] .023	------	[9] .025	------	[10] .1251

[1] Excluding reclamation plants.
[2] Excludes depreciation and depletion.
[3] Also 1,929,000 pounds of oxide at an average selling price of 5.23 cents per pound.
[4] Also 1,790,000 pounds of oxide at an average selling price of 4.77 cents per pound.
[5] Underground cost for 194,000 tons hoisted last 5 months of 1930.
[6] Includes Baltic mine.
[7] State mine appraiser's report as quoted in Mineral Resources; excludes reclamation plants.
[8] Includes transportation to mill, concentration, smelting, and refining.
[9] Includes depreciation and local taxes.
[10] Includes selling expense, new construction, and income taxes.

TABLE 56.—*Mining, milling, and general cost per ton of ore and per pound of copper at selected Canadian copper mines*

Mine	Year	Ore treated (tons)	Copper produced (pounds)	Mining costs per ton	Total cost	
					Per ton	Per pound
Granby (all units)	1929	2,490,000	60,855,000			[1] $0.1061
Do	1930	[2] 2,200,000	46,800,000			[1] .0980
Do	1931	1,578,000	36,512,000			[1] .0682
Do	1932	1,740,000	38,649,000			[1] .0561
Do	1933	1,533,000	34,460,000			[1] .0674
Do	1934	1,892,000	37,092,000			[1] .0773
Noranda	1929	422,000	51,625,000			[3] .1001
Do	1930	849,000	76,142,000			[3] .0716
Do	1931	1,012,000	62,859,000			[3] .0435
Do	1932	1,218,000	63,013,000			[4] .1294
Do	1933	1,542,000	65,009,000			[4] .1196
Do	1934	1,777,000	70,176,000			[4] .1136
Flin Flon [5]	1931	1,101,000	[6] 31,385,000		[7] $3.13	[8] .0887
Do	1932	1,447,000	42,425,000		[9] 2.55	[10] .1353
Do	1933	1,608,000	41,373,000		[9] 2.72	[10] .1584
Do	1934	1,477,000	37,725,000		[9] 3.14	[10] .1862
Sherritt-Gordon [11]	1931	214,000	14,718,000	$1.368		[12] .0805
Do	1932	158,000	[2] 10,867,000	1.214		[13] .0646

[1] Excludes depreciation, depletion, and income taxes; includes credit for gold and silver (about $0.007).
[2] Estimated.
[3] Includes general expenses, taxes, depreciation, and credit for gold, silver, and other income as follows: 1929, $0.041; 1930, $0.041; 1931, $0.090; excludes depletion.
[4] Includes all costs before depletion but does not include credits. Credits for gold are as follows: 1932, $0.1120; 1933, $0.1242; 1934, $0.1222.
[5] Started August 1930.
[6] Also 35,056,000 pounds of zinc.
[7] Includes all operating, administrative, and overhead costs for mining, milling, smelting, and electrolytic zinc refining.
[8] Includes depreciation, interest, and all expenses, also $0.093 (estimated) credit for zinc, gold, and silver.
[9] Includes all operating, administrative, and overhead costs for mining, milling, smelting, refining, and selling.
[10] Net before depletion; does not include credit for gold, silver, and zinc which is estimated to be about $0.09.
[11] 9 months; started April 1931.
[12] Includes depreciation, prepaid development, also $0.008 credit for gold and silver.
[13] Net before depreciation, taxes, and depletion; does not include gold credit.

Depreciation and interest usually are not calculated as operating costs but are added as separate items to the cost per pound of producing copper; the same is often true of income taxes. Depletion is another proper charge, amounting at some mines to 6 cents a pound, but is rarely shown in income statements and is excluded from the costs shown in tables 53 to 56. It is customary to credit the cost per pound with the gold and silver recovered in the ore; miscellaneous revenues also are credited. No uniform method of accounting is followed by the mining companies, and the costs shown in these tables are not comparable except as qualified by the footnotes.

Table 57 shows the weighted average costs of producing copper in the United States in 1929 and 1930.

TABLE 57.—*Weighted average costs of American copper production, 1929 and 1930, in cents per pound* [1]

	1929 [2]	*1930* [3]
Operating	9.92	9.51
Taxes and interest	1.21	.84
Depreciation	1.07	1.05
Miscellaneous expense	.13	.01
Sundry metals	1.52	1.20
Miscellaneous income	1.17	.48
Net costs, after depreciation	9.65	9.73
Copper output (millions of pounds)	2,091	1,370

[1] Strauss, S. D., Copper Production and Costs in 1929 and 1930: Met. and Min. Markets, June 25, 1931, p. 11.
[2] Includes Phelps Dodge, Kennecott, and Cerro de Pasco which were not in 1930 list.
[3] Includes Consolidated Coppermines, Greene Cananea, and Walker Mining, which were not in 1929 list.

Table 58 contains cost data, including taxes, interest, depreciation, and depletion, for 19 American copper companies in 1929. The data were compiled by the Mineral Research Corporation. The weighted average costs per pound of copper for these companies for 1928 and 1929 are shown in table 59.

TABLE 58.—*Cost per pound of producing copper by 19 American copper-mining companies in 1929* [1]

[All figures in cents per pound of copper produced unless otherwise stated]

Company	Production, millions of pounds, 1929	Cumulative debits				Cumulative credits				
		Operating costs	Plus taxes and interest	Plus depreciation	Plus depletion	Sundry metal income	Plus miscellaneous income	Minus miscellaneous expense	Cost after depreciation but before depletion	Cost after depreciation and depletion
Andes	[2] 130.5	7.42	9.06	10.09	--------	[3] 0.30	0.65	--------	9.44	--------
Calumet & Arizona	130.5	9.89	11.38	12.15	--------	1.80	2.79	2.31	9.84	--------
Calumet & Hecla	123.8	8.71	[4] 10.34	11.71	13.35	--------	.44	.25	11.46	13.10
Copper Range	24.2	[5] 16.48	--------	--------	--------	--------	[6] 5.11	[7] 2.02	[8] 14.46	--------
Cerro de Pasco	99.9	[9] 19.36	([9])	[10] 21.57	[10] 24.89	11.88	13.56	--------	8.01	11.33
Chile	[11] 212.6	4.74	[12] 7.75	8.88	--------	--------	.98	--------	7.90	--------
Granby	60.8	12.32	13.27	15.53	16.82	.73	1.42	--------	14.11	15.40
Howe Sound	43.0	27.20	28.34	30.65	--------	10.81	20.96	--------	9.69	--------
Inspiration	107.3	[13] 10.37	10.64	11.49	--------	--------	.03	--------	11.46	--------
Kennecott	501.1	[14] 12.22	13.46	14.34	--------	.61	[15] 6.70	6.56	7.78	--------
Magma	38.2	10.39	12.04	13.14	--------	2.48	[16] 2.85	--------	10.29	--------
Miami	58.8	11.06	11.75	12.47	--------	--------	[17] .49	--------	11.98	--------
Mohawk	20.0	7.04	7.83	8.23	9.04	--------	1.14	--------	7.09	7.90
Mother Lode	12.2	6.39	7.22	--------	--------	.41	.42	--------	[8] 6.80	--------
Nevada Consolidated	266.3	10.10	10.67	[18] 11.40	--------	.51	1.24	--------	10.16	--------
Noranda	51.6	9.27	10.66	13.13	--------	2.94	4.00	[19] 3.17	9.96	--------
Phelps Dodge	[20] 151.2	[21] 14.75	15.37	16.53	--------	[3] .51	[22] 4.13	--------	12.40	--------
United Verde Extension	64.1	8.47	9.70	9.81	14.26	.84	1.39	--------	8.42	12.87
Utah	298.6	7.65	8.47	8.96	--------	.97	[23] 1.10	--------	7.86	--------

[1] Tower, G. W., 3d, Cost of Producing Copper in the Americas: Eng. and Min. Jour., vol. 130, July 10, 1930, pp. 25–26.
[2] Represents sales; actual production was 162,663,775 pounds.
[3] Estimated credit, precious metal in own ores, based on percentage of sales to total production.
[4] Includes selling and administration expense.
[5] Includes $675,000 railroad-operating charge and other operating expense and taxes.
[6] Includes $899,000 railroad revenue and other operating revenue.
[7] Includes ½ Champion profits, or $633,000, to St. Mary's Mineral Land Co. and $114,000 interest on railroad bonds.
[8] Before depreciation and depletion; for Copper Range this amounted to 2.8 cents per pound in 1928.
[9] All operating expense and includes taxes.
[10] Charge allots 40 percent to depreciation and 60 percent to depletion assumed for total charge of $5,524,627.
[11] Represents sales; production was 299,575,752 pounds.
[12] Taxes and miscellaneous charges; interest and discount on bonds.
[13] Includes administration and Federal taxes.
[14] Includes cost of metal products.
[15] Includes revenue from railroads, steamships, and wharves, dividends and interest, and estimated return from metal products, less ½ Nevada Consolidated dividend.
[16] Includes profit from railroad after all charges.
[17] Does not include gain in sale of securities.
[18] Including $227,000 plant and equipment retirements.
[19] Includes $428,000 development and mining prior periods.
[20] Represents sales; production, including purchase ores, was 222,003,363 pounds.
[21] Less $5,420,000 merchandise sales.
[22] Includes precious metals in custom ores and estimated coal sales.
[23] Less ½ Nevada Consolidated dividend.

TABLE 59.—*Weighted average cost per pound of copper for 19 American copper companies 1928 and 1929* [1]

Year	Output, millions of pounds	Debit			Credit			Cost after depreciation but before depletion
		Operating costs	Taxes and interest	Depreciation	Sundry metal income	Miscellaneous income	Miscellaneous expense	
1929	2,091	9.93	1.21	1.07	1.52	1.17	0.13	9.65
1928	2,150	9.32	1.19	1.02	1.54	1.02	.11	9.08

[1] Tower, G. W., 3d, Cost of Producing Copper in the Americas: Eng. and Min. Jour., vol. 130, July 10, 1930, pp. 25–26.

The average price of producing copper, 1922–32, by 18 United States mining companies, 2 United States smelting companies, and 9 foreign mining companies is shown in tables 60 to 62.[97] The costs in table 60 are based on total dividends, those in table 61 on total dividends plus interest on bonded indebtedness, and those in table 62 on net earnings. All revenues for other metals in the ores are credited to the cost of producing copper.

These and subsequent tables were prepared by the Minerals Division, United States Bureau of Foreign and Domestic Commerce. Production data were taken from the American Bureau of Metal Statistics and Bureau of Mines publications. Financial data were abstracted from Moody's Investors Service and the Mines Handbook, by Rand and Sturgis.

TABLE 60.—*Average cost of producing copper, 1922–32, based on total dividends*

[Minus signs represent deficit]

	Production, thousands of pounds	Total dividends		Average selling price per pound, cents [1]	Average cost per pound, cents	Ratio of copper sales to total sales, percent
		Total, dollars	Average per pound, cents			
Mining companies, United States:						
Anaconda Copper	2,186,072	95,227,607	4.356	13.448	9.092	59.3
Calumet & Arizona [2]	524,913	29,473,646	5.615	14.711	9.096	85.6
Calumet & Hecla [3]	854,742	29,079,778	3.402	13.521	10.119	100.0
Champion Copper [4]	169,997	6,320,003	3.718	14.185	10.467	100.0
Copper Range	89,304	4,045,513	4.530	11.528	6.998	100.0
Inspiration Consolidated	840,991	14,185,604	1.687	12.990	11.303	100.0
Isle Royal Copper	95,875	1,462,500	1.525	13.673	12.148	100.0
Magma Copper	274,061	8,928,313	3.258	13.650	10.392	86.3
Miami Copper	594,408	12,887,736	2.168	13.375	11.207	100.0
Mohawk	172,020	5,527,388	3.213	13.356	10.143	100.0
Mother Lode Coalition	215,016	14,000,000	6.511	13.567	7.056	96.6
Nevada Consolidated [5]	1,994,264	47,035,766	2.359	13.687	11.328	97.3
New Cornelia [6]	431,367	16,380,000	3.797	13.743	9.946	96.4
Old Dominion [7]	210,685	−3,504,749	−1.664	13.644	15.308	94.7
Phelps Dodge	1,879,513	31,478,000	1.675	13.228	11.553	56.4
Utah Copper	2,100,147	112,901,654	5.350	13.713	8.363	93.4
United Verde [8]	903,552	38,400,000	4.250	14.285	10.035	86.9
United Verde Extension	460,696	24,816,560	5.387	13.347	7.960	93.6
Total	14,007,623	488,645,319	3.488	13.578	10.090	
Total United States mine production	15,836,749					
Percent of total	88.5					

[1] Average selling price for each company is weighted according to company yearly output, based on average New York selling price for the year; also the average for all companies combined is weighted accordingly.
[2] Calumet & Arizona, 1922–30, inclusive.
[3] Calumet & Hecla, 1923–32, inclusive.
[4] Champion Copper, 1922–30, inclusive.
[5] Nevada Consolidated, includes Ray and Chino mines prior to 1926.
[6] New Cornelia, 1922–28, inclusive.
[7] Old Dominion, 1922–31, inclusive.
[8] United Verde Copper, 1922–31, inclusive.

[97] Keiser, H. D., Costs and the Code, An Interpretation of the Future; Eng. and Min. Jour., vol. 135, April 1934, p. 174.

TABLE 60.—*Average cost of producing copper, 1922–32, based on total dividends*—Continued

	Production, thousands of pounds	Total dividends		Average selling price per pound, cents	Average cost per pound, cents	Ratio of copper sales to total sales, percent
		Total, dollars	Average per pound, cents			
Smelting and refining companies:						
American Metal	3, 921, 094	20, 460, 758	0. 522	13. 283	12. 761	49. 4
American Smelting & Refining	9, 514, 684	81, 448, 195	. 856	13. 684	12. 828	46. 0
Total	13, 435, 778	101, 908, 953	0. 758	13. 567	12. 809	
Total United States smelter production	16, 913, 451					
Percent of total	70. 4					
Foreign mining companies:						
Andes Copper [9]	522, 092	10, 828, 498	2. 074	14. 139	12. 065	98. 0
Cerro de Pasco	903, 666	42, 614, 301	4. 716	13. 240	8. 524	54. 5
Chile Copper	2, 207, 995	106, 178, 596	4. 809	13. 559	8. 750	100. 0
Granby Consolidated	490, 416	8, 015, 584	1. 634	13. 149	11. 515	93. 7
Greene Cananea	388, 909	7, 125, 000	1. 832	13. 102	11. 270	92. 3
Hudson Bay [10]	75, 599	−597, 896	−. 791	6. 841	7. 632	45. 0
Kennecott (Braden and Alaska)	2, 140, 593	93, 007, 870	4. 345	13. 119	8. 774	99. 3
Noranda Mines, Ltd.[11]	285, 669	8, 111, 901	2. 840	11. 376	8. 536	65. 5
Katanga	2, 092, 936	40, 308, 792	1. 926	[12] 12. 132	10. 206	100. 0
Total foreign	9, 107, 875	315, 592, 546	3. 465	12. 963	9. 498	
Total shown, United States and foreign	23, 115, 498					
Total world mine production	34, 845, 907					
Percent of world mine production	66. 3					

[9] Andes Copper, 1927–32, inclusive.
[10] Hudson Bay, 1930–32, inclusive.
[11] Noranda, 1928–32, inclusive.
[12] London price.

TABLE 61.—*Average cost of producing copper, 1922–32, based on total dividends plus interest on bonded indebtedness*

[Minus signs represent deficit]

	Production, thousands of pounds	Total dividends plus interest on bonded debt		Average selling price per pound, cents [1]	Average cost per pound, cents	Ratio of copper sales to total sales, percent
		Total, dollars	Average per pound, cents			
Mining companies, United States:						
Anaconda Copper	2, 186, 072	158, 966, 477	7. 272	13. 448	6. 176	59. 3
Calumet & Arizona [2]	524, 913	29, 473, 646	5. 615	14. 711	9. 096	85. 6
Calumet & Hecla [3]	854, 742	29, 079, 778	3. 402	13. 521	10. 119	100. 0
Champion Copper [4]	169, 997	6, 320, 003	3. 718	14. 185	10. 467	100. 0
Copper Range [4]	89, 304	5, 178, 834	5. 799	11. 528	5. 729	100. 0
Inspiration Consolidated	840, 991	16, 511, 524	1. 963	12. 990	11. 037	100. 0
Isle Royal Copper	95, 875	1, 462, 500	1. 525	13. 673	12. 148	100. 0
Magma Copper	274, 061	9, 673, 851	3. 530	13, 650	10. 120	86. 3
Miami Copper	594, 408	12, 887, 736	2. 168	13. 375	11. 207	100. 0

[1] Average selling price for each company is weighted according to company yearly output based on average New York selling price for the year; also the average for all companies combined is weighted accordingly.
[2] Calumet & Arizona, 1922–30, inclusive.
[3] Calumet & Hecla, 1923–32, inclusive.
[4] The Copper Range Co. up until 1931 owned a half interest in the Champion Copper Co., at which time it took over the latter in entirety. In the annual statements of the Copper Range, 50 percent of the Champion production is shown each year, and to obtain the actual production of Copper Range, 1922–30, inclusive, 50 percent of Champion production was deducted, as Champion is shown separately, 1922–30, inclusive, after which it is consolidated with the Copper Range.

TABLE 61.—*Average cost of producing copper, 1922–32, based on total dividends plus interest on bonded indebtedness*—Continued

[Minus signs represent deficit]

	Production, thousands of pounds	Total dividends plus interest on bonded debt		Average selling price per pound, cents	Average cost per pound, cents	Ratio of copper sales to total sales, percent
		Total, dollars	Average per pound, cents			
Mining companies, United States—Continued.						
Mohawk	172, 020	5, 527, 388	3. 213	13. 356	10. 145	100. 0
Mother Lode Coalition	215, 016	14, 000, 000	6. 511	13. 567	7. 056	96. 6
Nevada Consolidated [5]	1, 994, 264	48, 364, 047	2. 425	13. 687	11. 262	97. 3
New Cornelia [6]	431, 367	16, 380, 000	3. 797	13. 743	9. 946	96. 4
Old Dominion [7]	210, 685	−3, 504, 749	−1. 664	13. 644	15. 308	94. 7
Phelps Dodge	1, 879, 513	31, 478, 000	1. 675	13. 228	11. 553	56. 4
Utah Copper	2, 110, 147	112, 901, 654	5. 350	13. 713	8. 363	93. 4
United Verde [8]	903, 552	38, 400, 000	4. 250	14. 285	10. 035	86. 9
United Verde Extension	460, 696	24, 816, 560	5. 387	13. 347	7. 960	93. 6
Total	14, 007, 623	557, 917, 249	3. 983	13. 578	9. 595	
Total United States mines production	15, 836, 749					
Percent of total	88. 5					
Smelting and refining companies:						
American Metal	3, 921, 094	23, 546, 092	0. 600	13. 283	12. 683	49. 4
American Smelting & Refining	9, 514, 684	105, 327, 152	1. 107	13. 684	12. 577	46. 0
Total	13, 435, 778	128, 873, 244	0. 959	13. 567	12. 608	
Total United States smelter production	16, 913, 451					
Percent of United States total	79. 4					
Foreign mining companies:						
Andes Copper [9]	522, 092	16, 413, 098	3. 144	14. 139	10. 995	98. 0
Cerro de Pasco	903, 666	43, 631, 661	4. 828	13. 240	8. 412	54. 5
Chile Copper	2, 207, 995	131, 716, 396	5. 965	13. 559	7. 594	100. 0
Granby Consolidated, Ltd	490, 416	9, 526, 726	1. 942	13. 149	11. 207	93. 7
Greene Cananea	388, 909	7, 125, 000	1. 832	13. 102	11. 270	92. 3
Hudson Bay [10]	75, 599	676, 221	. 894	6. 841	5. 947	45. 0
Kennecott (Braden and Alaska)	2, 140, 593	93, 007, 870	4. 345	13. 119	8. 774	99. 3
Noranda Mines, Ltd.[11]	285, 669	8, 111, 901	2. 840	11. 376	8. 536	65. 0
Katanga	2, 092, 936	40, 308, 792	1. 926	[12] 12. 132	10. 206	100. 0
Total	9, 107, 875	350, 517, 665	3. 849	12. 963	9. 114	
Total shown, United States and foreign	23, 115, 498					
Total world production	34, 845, 907					
Percent of world production	66. 3					

[5] Nevada Consolidated, includes Ray and Chino mines prior to 1926.
[6] New Cornelia, 1922–28, inclusive.
[7] Old Dominion, 1922–31, inclusive.
[8] United Verde Copper, 1922–31, inclusive.
[9] Andes Copper, 1927–32, inclusive.
[10] Hudson Bay, 1930–32, inclusive.
[11] Noranda, 1928–32, inclusive.
[12] London price.

TABLE 62.—*Average cost of producing copper, 1922–32, based on net earnings*

[Minus signs represent deficit]

	Production, thousands of pounds	Net earnings		Average selling price per pound, cents [2]	Average cost per pound, cents	Ratio of copper sales to total sales, percent
		Total, dollars [1]	Average per pounds, cents			
Mining companies, United States:						
Anaconda Copper	2, 186, 072	105, 195, 515	4. 812	13. 448	8. 636	50. 3
Calumet & Arizona [3]	524, 913	24, 840, 791	4. 732	14. 711	9. 979	85. 6
Calumet & Hecla [4]	854, 742	1, 730, 426	. 202	13. 521	13. 319	100. 0
Champion Copper [5]	169, 997	1, 231, 375	. 724	14. 185	13. 461	100. 0
Copper Range	89, 304	−1, 826, 258	−2. 045	11. 528	13. 573	100. 0
Inspiration Consolidated	840, 991	14, 095, 448	1. 676	12. 990	11. 314	100. 0
Isle Royal Copper	95, 875	−1, 721, 824	−1. 796	13. 673	15. 469	100. 0
Magma Copper	274, 061	7, 947, 117	2. 900	13. 650	10. 750	86. 3
Miami Copper	594, 408	6, 682, 632	1. 124	13. 375	12. 251	100. 0
Mohawk	172, 020	4, 146, 077	2. 410	13. 356	9. 946	100. 0
Mother Lode Coalition	215, 016	4, 305, 527	2. 002	13. 567	11. 565	96. 6
Nevada Consolidated [6]	1, 994, 264	56, 249, 047	2. 821	13. 687	10. 866	97. 3
New Cornelia [7]	431, 367	13, 580, 320	3. 148	13. 743	10. 595	96. 4
Old Dominion [8]	210, 685	−3, 504, 749	−1. 664	13. 644	15. 308	94. 7
Phelps Dodge	1, 879, 513	−24, 548, 941	−1. 306	13. 228	14. 534	56. 4
Utah Copper	2, 110, 147	121, 637, 313	5. 764	13. 713	7. 949	93. 4
United Verde [9]						
United Verde Extension	460, 696	[10] 41, 788	[10] . 009	13. 347	13. 356	93. 6
Total	13, 104, 071	329, 998, 028	2. 518	13. 530	11. 012	
Total United States mine production	15, 836, 749					
Percent of total	82. 7					
Smelting and refining companies:						
American Metal	3, 921, 094	23, 386, 910	0. 596	13. 283	12. 687	49. 4
American Smelting & Refining	9, 514, 684	122, 344, 304	1. 286	13. 684	12. 398	46. 0
Total	13, 435, 778	145, 731, 214	1. 085	13. 567	12. 482	
Total United States smelter production	16, 913, 451					
Percent of United States total	79. 4					
Foreign mining companies:						
Andes Copper [11]	522, 092	18, 903, 764	3. 621	14. 139	10. 518	98. 0
Cerro de Pasco	903, 666	20, 791, 491	2. 301	13. 240	10. 939	54. 5
Chile Copper	2, 207, 995	102, 216, 396	4. 629	13. 559	8. 930	100. 0
Granby Consolidated	490, 416	−3, 830, 125	−. 781	13. 149	13. 930	93. 7
Greene Cananea	388, 909	6, 628, 818	1. 704	13. 102	11. 398	92. 3
Hudson Bay [12]	75, 599	−597, 896	−. 791	6. 841	7. 632	45. 0
Kennecott (Braden and Alaska)	2, 140, 593	97, 784, 364	4. 568	13. 119	8. 551	99. 3
Noranda Mines, Ltd.[13]	285, 669	15, 464, 395	5. 413	11. 376	5. 963	65. 5
Katanga	2, 092, 936	36, 522, 175	1. 745	[14] 12. 132	10. 387	100. 0
Total	9, 107, 875	293, 883, 382	3. 227	12. 963	9. 736	
Total shown, United States and foreign	22, 211, 946					
Total world mine production	34, 845, 907					
Percent of world mine production	63. 7					

[1] Take into account in most instances depreciation and depletion.
[2] Average selling price for each company is weighted according to company yearly output, based on average New York selling price for the year: also the average for all companies combined is weighted accordingly.
[3] Calumet & Arizona, 1922–30, inclusive.
[4] Calumet & Hecla, 1923–32, inclusive.
[5] Champion Copper Co., 1922–30, inclusive.
[6] Nevada Consolidated, includes Ray and Chino mines prior to 1926.
[7] New Cornelia, 1922–28, inclusive.
[8] Old Dominion, 1922–31, inclusive.
[9] No data available.
[10] Not actual deficit but is due to high rate charged for depletion.
[11] Andes Copper, 1927–32, inclusive.
[12] Hudson Bay, 1930–32, inclusive.
[13] Noranda, 1928–32, inclusive.
[14] London price.

Table 63 shows production costs at 11 United States copper mining companies, 1922–32, prorated on the basis of the various metals comprising the output.[98]

Table 64 shows the same data for eight foreign companies.

[98] Keiser, H. D., Production Costs Prorated on Basis of Metals Comprising Output: Eng. and Min. Jour., vol. 135, May 1934, p. 207.

TABLE 63.—*Prorated production costs of 11 United States copper mining companies*

	Copper	Lead	Zinc	Silver	Gold	Total
Anaconda Copper[1] (1922-32, inclusive):						
Production (pounds and ounces)	2, 186, 072, 000	191, 241, 352	2, 000, 862, 000	101, 009, 737	354, 000	
Sales receipts (estimate) percent	59. 3	2. 7	24. 5	12. 0	1. 5	$495, 848, 559
Based on dividends:						
Prorated dividends	$56, 469, 971	$2, 571, 145	$23, 330, 764	$11, 427, 313	$1, 428, 414	$95, 227, 607
Sales price per unit cents	13, 448	7. 104	6. 072	58. 871	$20. 67183	
Dividends per unit do	2. 583	1. 344	1. 166	11. 313	$4. 03507	
Cost per unit do	10. 865	5. 760	4. 906	47. 558	$16. 63676	
Based on dividends plus bond interest:						
Prorated dividends plus bond interest	$94, 267, 121	$4, 292, 095	$38, 946, 787	$19, 075, 977	$2, 384, 497	$158, 966, 477
Sales price per unit cents	13. 448	7. 104	6. 072	58. 871	$20. 67183	
Dividends plus bond interest per unit do	4. 312	2. 244	1. 947	18. 885	$6. 73587	
Cost per unit do	9. 136	4. 860	4. 125	39. 986	$13. 93596	
Based on earnings:						
Prorated earnings	$62, 380, 940	$2, 840, 279	$25, 772, 901	$12, 623, 462	$1, 577, 933	$105, 195, 515
Sales price per unit cents	13. 448	7. 104	6. 072	58. 871	$20. 67183	
Net earnings per unit do	2. 854	1. 485	1. 288	12. 497	$4. 45743	
Cost per unit do	10. 594	5. 619	4. 784	46. 374	$16. 22440	
Calumet & Arizona (1922-30, inclusive; 1931 consolidated with Phelps Dodge):						
Production (pounds and ounces)	524, 913, 120			9, 829, 015	356, 668	
Sales receipts (estimate) percent	85. 6			6. 2	8. 2	$90, 209, 271
Based on dividends:						
Prorated dividends	$25, 229, 441			$1, 827, 366	$2, 416, 839	$29, 473, 646
Sales price per unit cents	14. 711			57. 123	$20. 67183	
Dividends per unit do	4. 806			18. 592	$6. 77616	
Cost per unit do	9. 905			38. 531	$13. 89567	
Based on dividends and bond interest: No bonded debt.						
Based on earnings:						
Prorated earnings	$21, 263. 717			$1, 540, 129	$2, 036, 945	$24, 840, 791
Sales price per unit cents	14. 711			57. 123	$20. 67183	
Net earnings per unit do	4. 051			15. 669	$5. 71104	
Cost per unit do	10. 660			41. 454	$14. 96079	
Kennecott (Utah, Braden, and Alaska; 1922-32, inclusive):						
Production (pounds and ounces)	4, 250, 740, 227			10, 192, 158	795, 371	
Sales receipts (estimate) percent	96. 2			1. 0	2. 8	$592, 442, 283
Based on dividends:						
Prorated dividends	$193, 957, 135			$2, 016, 186	$5, 645, 322	$201, 618, 643
Sales price per unit cents	13. 414			56. 920	$20. 67183	
Dividends per unit do	4. 563			19. 782	$7. 09772	
Cost per unit do	8. 851			37. 138	$13. 57411	

Based on dividends and bond interest:						
Prorated dividends and bond interest	$198,084,962			$2,059,095	$5,765,467	$205,909,524
Sales price per unit ... cents	13.414			56.920	$20.67183	
Dividends and bond interest per unit ... do	4.660			20.203	$7.24878	
Cost per unit ... do	8.754			36.717	$13.42305	
Based on earnings:						
Prorated earnings	$211,083,653			$2,194,217	$6,143,807	
Sales price per unit ... cents	13.414			56.920	$20.67183	
Net earnings per unit ... do	4.966			21.528	$7.72445	
Cost per unit ... do	8.448			35.392	$12.94738	
Mother Lode Coalition mines (1922–32, inclusive):						
Production (pounds and ounces)	215,016,860			1,683,678		
Sales receipts (estimate) ... percent	96.6			3.4		$30,199,776
Based on dividends:						
Prorated dividends	$13,524,000			$476,000		$14,000,000
Sales price per unit ... cents	13.567			61.062		
Dividend per unit ... do	6.290			28.271		
Cost per unit ... do	7.277			32.791		
Based on dividends and bond interest: No bonded debt.						
Based on earnings:						
Prorated earnings	$4,159,139			$146,388		$4,305,527
Sales price per unit ... cents	13.567			61.062		
Earnings per unit ... do	1.934			8.695		
Cost per unit ... do	11.633			52.367		
Magma Copper (1922–32, inclusive):						
Production (pounds and ounces)	274,061,188			7,931,704	90,395	
Sales receipts (estimate) ... percent	86.3			9.4	4.3	$43,388,176
Based on dividends:						
Prorated dividends	$7,705,134			$839,262	$383,917	$8,928,313
Sales price per unit ... cents	13.650			51.809	$20.67183	
Dividends per unit ... do	2.811			10.581	$4.24710	
Cost per unit ... do	10.839			41.308	$16.42473	
Based on dividends and bond interest:						
Prorated dividends and bond interest	$8,348,533			$909,342	$415,976	$9,673,851
Sales price per unit ... cents	13.650			51.809	$20.67183	
Dividends plus bond interest per unit ... do	3.046			11.465	$4.60175	
Cost per unit ... do	10.604			40.344	$16.07008	
Based on earnings:						
Prorated earnings	$6,858,362			$747,029	$341,726	$7,947,117
Sales price per unit ... cents	13.650			51.809	$20.67183	
Net earnings per unit ... do	2.502			9.418	$3.78036	
Cost per unit ... do	11.148			42.391	$16.89147	
Nevada Consolidated [2] (1922–32, inclusive):						
Production (pounds and ounces)	1,994,264,157			1,308,404	335,203	
Sales receipts (estimate) ... percent	97.3			0.2	2.5	$280,596,451
Based on dividends:						
Prorated dividends	$45,765,800			$94,072	$1,175,894	$47,035,766
Sales price per unit ... cents	13.687			53.759	$20.67183	
Dividends per unit ... do	2.295			7.190	$3.50801	
Cost per unit ... do	11.392			47.569	$17.16382	

[1] Exclusive of Andes Copper Co., Chile Copper Corporation, and Greene Cananea Copper Co., but does include International Smelting Co. and all other subsidiaries.
[2] Ray and Chino mines included in figures prior to 1926; after that year consolidated with Nevada Consolidated.

TABLE 63.—*Prorated production costs of 11 United States copper mining companies*—Continued

	Copper	Lead	Zinc	Silver	Gold	Total
Nevada Consolidated (1922–32, inclusive)—Continued.						
Based on dividends and bond interest:						
Prorated dividends and bond interest	$47, 058, 218	----	----	$96, 728	$1, 209, 101	$48, 364, 047
Sales price per unit ....cents	13. 687	----	----	53. 759	$20. 67183	----
Dividends plus bond interest per unit ....do	2. 359	----	----	7. 393	$3. 60707	----
Cost per unit ....do	11. 328	----	----	47. 366	$17. 06476	----
Based on earnings:						
Prorated earnings	$54, 730, 323	----	----	$112, 498	$1, 406, 226	$56, 249, 047
Sales price ....cents	13. 687	----	----	53. 759	$20. 67183	----
Earnings per unit ....do	2. 744	----	----	8. 598	$4. 19515	----
Cost per unit ....do	10. 943	----	----	45. 161	$16. 47668	----
New Cornelia (1922–28, inclusive; consolidated with Calumet & Arizona in 1929):						
Production (pounds and ounces)	431, 367, 514	----	----	978, 934	77, 726	----
Sales receipts (estimate) ....percent	96. 4	----	----	1. 0	2. 6	$61, 501, 868
Based on dividends:						
Prorated dividends	$15, 790, 320	----	----	$163, 800	$425, 880	$16, 380, 000
Sales price per unit ....cents	13. 743	----	----	62. 505	$20, 67183	----
Dividends per unit ....do	3. 661	----	----	16. 732	$5. 47925	----
Costs per unit ....do	10. 082	----	----	45. 773	$15. 19258	----
Based on dividends and bond interest: No bonded debt.						
Based on earnings:						
Prorated earnings	$13, 091, 429	----	----	$135, 803	$353, 088	$13, 580, 320
Sales price per unit ....cents	13. 743	----	----	62. 505	$20. 67183	----
Earnings per unit ....do	3. 035	----	----	13. 873	$4. 54273	----
Cost per unit ....do	10. 708	----	----	48. 632	$16. 12910	----
Old Dominion (1922–31, inclusive; closed down in 1932):						
Production (pounds and ounces)	210, 685, 031	----	----	1, 160, 025	43, 553	----
Sales receipts (estimate) ....percent	94. 7	----	----	2. 3	3. 0	$30, 353, 492
Based on dividends or deficit:						
Prorated deficit	$3, 318, 997	----	----	$80, 609	$105, 143	$3, 504, 749
Sales price per unit ....cents	13. 644	----	----	61. 006	$20. 67183	----
Deficit per unit ....do	1. 575	----	----	6. 949	$2. 36822	----
Cost per unit ....do	15. 229	----	----	67. 955	$23. 04005	----
Based on dividends and interest on bonded indebtedness: No bonded debt.						
Based on earnings: No earnings.						
Utah Copper (1922–32, inclusive):						
Production (pounds and ounces)	2, 110, 147, 163	----	----	7, 024, 101	795, 371	----
Sales receipts (estimate) ....percent	93. 4	----	----	1. 3	5. 3	$309, 720, 920
Based on dividends:						
Prorated dividends	$105, 450, 145	----	----	$1, 467, 921	$5, 983, 788	$112, 901, 654
Sales price per unit ....cents	13. 713	----	----	55. 855	$20. 67183	----
Dividend per unit ....do	4. 997	----	----	20. 895	$7. 52327	----
Cost per unit ....do	8. 716	----	----	34. 960	$13. 14856	----

Based on dividends and bonded indebtedness: No bonded debt.						
Based on earnings:						
Prorated earnings	$113, 609, 250			$1, 581, 285	$6, 446, 778	$121, 637, 313
Sales price per unit ____cents	13. 713			55. 855	$20. 67183	
Earnings per unit ____do	5. 384			22. 512	$8. 10537	
Cost per unit ____do	8. 329			33. 343	$12. 56646	
United Verde (1922–31, inclusive; closed down in 1932):						
Production (pounds and ounces)	903, 552, 152			16, 909, 059	461, 621	
Sales receipts (estimate) ____percent	86. 9			6. 7	6. 4	$148, 600, 750
Based on dividends:						
Prorated dividends	$33, 369, 600			$2, 572, 800	$2, 457, 600	$38, 400, 000
Sales price per unit ____cents	14. 285			59. 067	$20. 67183	
Dividend per unit ____do	3. 693			15. 216	$5. 32383	
Cost per unit ____do	10. 572			43. 851	$15. 34800	
Based on dividends and bonded indebtedness: No bonded debt.						
Based on earnings: No data available.						
United Verde Extension (1922–32, inclusive):						
Production (pounds and ounces)	460, 696, 000			3, 854, 685	102, 908	
Sales receipts (estimate) ____percent	93. 6			3. 2	3. 2	$65, 710, 484
Based on dividends:						
Prorated dividends	$23, 228, 300			$794, 130	$794, 130	$24, 816, 560
Sales price per unit ____cents	13. 347			54. 287	$20. 67183	
Dividend per unit ____do	5. 042			20. 602	$7. 71689	
Cost per unit ____do	8. 305			33. 685	$12. 95494	
Based on dividends and bonded indebtedness: No bonded debt.						
Based on deficit:						
Prorated deficit	$39, 114			$1, 337	$1, 337	$41, 788
Sales price per unit ____cents	13. 347			54. 287	$20. 67183	
Deficit per unit ____do	0. 008			0. 035	$0. 01299	
Cost per unit ____do	13. 355			54. 252	$20. 65884	

TABLE 64.—*Prorated production costs of eight foreign copper-mining companies*

	Copper	Lead	Zinc	Silver	Gold	Total
Andes Copper [1] (1927–32, inclusive):						
Production (pounds and ounces)	522, 092, 254			853, 802	51, 551	
Sales receipts (estimate) percent	98. 0			0. 6	1. 4	
Based on dividends:						
Prorated dividends	$10, 611, 928			$64, 971	$151, 599	$10, 828, 498
Sales price per unit cents	14. 139			49. 643	$20. 67183	
Dividends per unit do	2. 033			7. 609	$2. 94076	
Cost per unit do	12. 106			42. 034	$17. 73107	
Based on dividends and bond interest:						
Prorated dividends and bond interest	$16, 084, 836			$98, 479	$227, 783	$16, 413, 098
Sales price per unit cents	14. 139			49. 643	$20. 67183	
Dividends and bond interest per unit do	3. 081			11. 534	$4. 45739	
Cost per unit percent	11. 058			38. 109	$16. 21444	
Based on earnings:						
Prorated earnings	$18, 525, 689			$113, 423	$264, 652	$18, 903, 764
Sales price per unit cents	14. 139			49. 643	$20. 67183	
Net earnings per unit do	3. 548			13. 284	$5. 13381	
Cost per unit do	10. 591			36. 359	$15. 53802	
Cerro de Pasco (1922–32, inclusive):						
Production (pounds and ounces)	903, 666, 000	[2] 123, 042, 000	[2] 85, 510, 000	137, 528, 000	363, 577	
Sales receipts (estimate) percent	54. 5	3. 4	2. 3	36. 4	3. 4	$219, 351, 507
Based on dividends:						
Prorated dividends	$23, 224, 794	$1, 448, 886	$980, 129	$15, 511, 606	$1, 448, 886	$42, 614, 301
Sales price per unit cents	13. 240	6. 039	5. 877	57. 148	$20. 67183	
Dividends per unit do	2. 570	1. 178	1. 146	11. 117	$3. 98509	
Cost per unit do	10. 670	4. 861	4. 731	46. 031	$16. 68674	
Based on dividends and bond interest:						
Prorated dividends and bond interest	$23, 779, 255	$1, 483, 476	$1, 003, 528	$15, 881, 926	$1, 483, 476	$43, 631, 661
Sales price per unit cents	13. 240	6. 039	5. 877	57. 148	$20. 67183	
Dividends and bond interest per unit do	2. 631	1. 206	1. 173	11. 383	$4. 08025	
Cost per unit do	10. 609	4. 833	4. 704	45. 765	$16. 59158	
Based on earnings:						
Prorated earnings	$11, 331, 363	$706, 911	$478, 204	$7, 568, 103	$706, 910	$20, 791, 491
Sales price per unit cents	13. 240	6. 039	5. 877	57. 148	$20. 67183	
Earnings per unit (net) do	1. 254	0. 575	0. 559	5. 424	$1. 94432	
Cost per unit do	11. 986	5. 464	5. 318	51. 724	$18. 73751	
Granby Consolidated (1922–32, inclusive):						
Production (pounds and ounces)	490, 416, 158			4, 391, 093	90, 795	
Sales receipts (estimate) percent	93. 7			3. 5	2. 8	$68, 789, 933
Based on dividends:						
Prorated dividends	$7, 510, 602			$280, 545	$224, 437	$8, 015, 581
Sales price per unit cents	13. 149			55. 306	$20. 67183	
Dividends per unit do	1. 531			6. 389	$2. 47191	
Cost per unit do	11. 618			48. 917	$18. 19992	

Based on dividends and bond interest:						
Prorated dividends and bond interest	$8, 926, 542	----	----	$333, 436	$266, 748	$9, 526, 726
Sales price per unit ... cents	13. 149	----	----	55. 306	$20. 67183	----
Dividends per unit ... do	1. 820	----	----	7. 593	$2. 27708	----
Cost per unit ... do	11. 329	----	----	47. 713	$18. 39475	----
Based on deficit:						
Prorated deficit	$3, 588, 827	----	----	$134, 054	$107, 244	$3, 830, 125
Sales price per unit ... cents	13. 149	----	----	55. 306	$20. 67183	----
Deficit per unit ... do	0. 732	----	----	3. 053	$1. 18117	----
Cost per unit ... do	13. 881	----	----	58. 359	$21. 75300	----
Green Cananea (1922–32, inclusive):						
Production (pounds and ounces)	388, 909, 021	----	----	4, 985, 001	66, 687	----
Sales receipts (estimate) ... percent	92. 3	----	----	5. 2	2. 5	$55, 190, 494
Based on dividends:						
Prorated dividends	$6, 576, 375	----	----	$370, 500	$178, 125	$7, 125, 000
Sales price per unit ... cents	13. 102	----	----	57. 320	$20. 67183	----
Dividends per unit ... do	1. 691	----	----	7. 432	$2. 67106	----
Cost per unit ... do	11. 411	----	----	49. 888	$18. 00077	----
Based on dividends and bond interest: No bonded debt.						
Based on earnings:						
Prorated earnings	$6, 118, 399	----	----	$344, 699	$165, 720	$6, 628, 818
Sales price per unit ... cents	13. 102	----	----	57. 320	$20. 67183	----
Earnings per unit ... do	1. 573	----	----	6. 917	$2. 48504	----
Cost per unit ... do	11. 529	----	----	50. 403	$18. 18679	----
Hudson Bay (1930–32, inclusive; production began in 1930):						
Production (pounds and ounces)	75, 598, 781	----	80, 698, 199	1, 636, 111	155, 565	----
Sales receipts (estimate) ... percent	45. 0	----	23. 0	4. 0	28. 0	$11, 503, 508
Based on dividends: No dividends paid.						
Based on bond interest:						
Prorated bond interest	$304, 299	----	$155, 531	$27, 049	$189, 342	$676, 221
Sales price per unit ... cents	6. 841	----	3. 289	28. 239	$20. 67183	----
Bond interest per unit ... do	0. 403	----	0. 193	1. 653	$1. 21712	----
Cost per unit ... do	6. 438	----	3. 096	26. 586	$19. 45471	----
Based on deficit:						
Prorated deficit	$269, 053	----	$137, 516	$23, 916	$167, 411	$597, 896
Sales price per unit ... cents	6. 841	----	3. 289	28. 239	$20. 67183	----
Deficit per unit ... do	0. 356	----	0. 170	1. 462	$1. 07615	----
Cost per unit ... do	7. 197	----	3. 359	29. 701	$21. 74798	----
Kennecott (Braden and Alaska; 1922–32, inclusive):						
Production (pounds and ounces)	2, 140, 593, 064	----	----	3, 168, 057	----	----
Sales receipts (estimate) ... percent	99. 3	----	----	0. 7	----	$282, 721, 363
Based on dividends:						
Prorated dividends	$88, 095, 920	----	----	$621, 019	----	$88, 716, 989
Sales price per unit ... cents	13. 119	----	----	58. 960	----	----
Dividends per unit ... do	4. 115	----	----	19. 602	----	----
Cost per unit ... do	9. 004	----	----	39. 358	----	----
Based on dividends and bond interest:						
Prorated dividends and bond interest	$92, 356, 815	----	----	$651, 055	----	$93, 007, 870
Sales price per unit ... cents	13. 119	----	----	58. 960	----	----
Dividends and bond interest per unit ... do	4. 314	----	----	20. 551	----	----
Cost per unit ... do	8. 805	----	----	38. 409	----	----

[1] 1927, first full year of production.

[2] Lead and zinc production began in 1926.

TABLE 64.—*Prorated production costs of eight foreign copper-mining companies*—Continued

	Copper	Lead	Zinc	Silver	Gold	Total
Kennecott (Braden and Alaska; 1922–32, inclusive)—Continued.						
Based on earnings:						
Prorated earnings	$97,099,873			$684,491		$97,784,364
Sales price per unit ... cents	13.119			58.960		
Earnings per unit ... do	4.536			21.606		
Cost per unit ... do	8.683			37.354		
Noranda Mines, Ltd. (1928–32, inclusive; production began in 1928):						
Production (pounds and ounces)	285,669,328			2,204,597	790,838	
Sales receipts (estimate) ... percent	65.5			1.6	32.9	$49,619,487
Based on dividends:						
Prorated dividends	$5,164,283			$126,150	$2,593,968	$7,884,401
Sales price per unit ... cents	11.376			35.124	$20.67183	
Dividends per unit ... do	1.808			5.722	$3.28002	
Cost per unit ... do	9.568			29.402	$17.39181	
Based on dividends and bond interest:						
Prorated dividends and bond interest	$5,313,295			$129,791	$2,668,815	$8,111,901
Sales price per unit ... cents	11.376			35.124	$20.67183	
Dividends and bond interest per unit ... do	1.860			5.887	$3.37467	
Cost per unit ... do	9.516			29.237	$17.29716	
Based on earnings:						
Prorated earnings	$10,129,178			$247,430	$5,087,787	$15,464,395
Sales price per unit ... cents	11.376			35.124	$20.67183	
Earnings per unit ... do	3.546			11.223	$6.43341	
Cost per unit ... do	7.830			23.901	$14.23842	
International Nickel & Mond Nickel, 1926–29:		*Nickel*	*Platinum*			
Production (pounds and ounces)	252,298,294	360,278,000	92,135	1,036,175	21,635	
Sales receipts (estimate) ... percent	22.34	72.60	4.47	0.33	0.26	$73,696,121
Based on dividends:						
Prorated dividends	$6,832,735	$22,204,859	$1,367,159	$100,931	$79,521	$30,585,205
Selling price per unit ... cents	15.380	35.000	$84.30	56.053	$20.67183	
Dividends per unit ... do	2.708	6.163	$14.84	9.741	$3.67557	
Cost per unit ... do	12.672	28.837	$69.46	46.312	$16.99626	
Based on dividends and bond interest:						
Prorated dividends and bond interest	$7,267,614	$23,618,119	$1,454,173	$107,355	$84,583	$32,531,845
Selling price per unit ... cents	15.380	35.000	$84.30	56,053	$20.67183	
Dividends and bond interest per unit ... do	2.881	6.556	$15.78	10.361	$3.90954	
Cost per unit ... do	12.499	28.444	$68.52	45.692	$16.76229	
Based on earnings:						
Prorated earnings	$11,791,989	$38,321,326	$2,359,454	$174,188	$137,239	$52,784,196
Sales price per unit ... cents	15.380	35.000	$84.30	56.053	$20.67183	
Earnings per unit ... do	4.674	10.637	$25.61	16.811	$6.34338	
Cost per unit ... do	10.706	24.363	$58.69	39.242	$14.32845	

Tables 65 and 66 show the investment in 14 United States copper-producing companies, 2 United States smelting companies, and 9 foreign producing companies at the end of 1929.[99] The investment necessary to produce a pound of copper is indicated. The data for each company are not strictly comparable as transportation charges comprise a substantial part of plant costs. Two methods were used in preparing the preceding two tables to show clearly the investment per pound of output. Table 65 gives the investment in mine development, plant, and equipment and table 66 the fixed investment, which includes fluctuations in the value of the mining property as well as investment in mine development, plant, and equipment.

[99] Keiser, H. D., Investment Metal Costs: Eng. and Min. Jour. vol. 135, August 1934, p. 353.

TABLE 65.—*Investment in copper-producing companies—Dec. 31, 1929*

	Ore mined (tons)	Actual output of copper (thousands of pounds)	Rated capacity (thousands of pounds)	Investment in mine development, plant, and equipment [1]				Working capital				Total investment [2]		
				Total (dollars)	Per ton of ore mined (dollars)	Per pound of actual output (cents)	Per pound of rated capacity (cents)	Total (dollars)	Per ton of ore mined (dollars)	Per pound of actual output (cents)	Per pound of rated capacity (cents)	Per ton of ore mined (dollars)	Per pound of actual output (cents)	Per pound of rated capacity (cents)
Mining companies (United States):														
Anaconda Copper	(3)	281,938	324,000	[4] 105,052,514	(3)	37.3	32.4	75,654,978	(3)	26.8	23.4	(3)	64.1	55.8
Calumet & Arizona	4,428,746	127,140	140,000	18,471,061	4.17	14.5	13.2	8,485,890	1.92	6.7	6.1	6.09	21.2	19.3
Calumet & Hecla	2,444,000	134,694	140,000	[5] 64,622,866	26.44	48.0	46.2	13,966,465	5.71	10.4	10.0	32.15	58.4	56.2
Inspiration Consolidated	5,773,858	107,308	120,000	21,160,586	3.66	19.7	17.6	5,272,785	.91	4.9	4.4	4.57	24.6	22.0
Isle Royale Copper	514,024	10,864	11,000	[6] 9,832,628	19.12	90.5	89.3	1,441,174	2.80	13.3	13.1	21.92	103.8	102.4
Magma Copper	269,579	38,256	48,000	[7] 3,495,838	12.97	9.1	7.3	5,094,664	18.90	13.3	10.6	31.87	22.4	17.9
Miami Copper	5,017,983	58,841	75,000	8,055,829	1.61	13.7	10.7	6,101,791	1.22	10.4	8.1	2.83	24.1	18.8
Mohawk	637,159	20,044	(3)	[8] 13,456,190	21.12	67.1	(3)	3,632,735	5.70	18.1	(3)	26.82	85.2	(3)
Mother Lode Coalition	57,177	12,242	(3)	[9] 10,130,140	177.17	82.7	(3)	591,911	10.35	4.8	(3)	187.52	87.5	(3)
Nevada Consolidated	12,888,289	266,274	350,000	58,278,185	4.52	21.9	16.6	27,236,023	2.11	10.2	7.8	6.63	32.1	24.4
Old Dominion	415,891	18,944	(3)	[10] 17,664,150	42.47	93.2	(3)	2,257,756	5.43	11.9	(3)	47.90	105.1	(3)
Phelps Dodge	4,604,716	167,180	240,000	44,683,395	9.70	26.7	18.6	32,241,698	7.00	19.3	13.4	16.70	46.0	32.0
United Verde Extension	358,654	64,112	(3)	7,680,841	21.42	12.0	(3)	4,820,347	13.44	7.5	(3)	34.86	19.5	(3)
Utah Copper	17,724,100	296,626	375,000	[11] 48,642,869	2.74	16.4	13.0	26,761,374	1.51	9.0	7.1	4.25	25.4	20.1
Total	55,134,176	1,604,463	1,823,000	431,227,092	[12] 5.08	[12] 27.0	[13] 16.3	213,559,591	[12] 2.56	[12] 10.6	[13] 8.6	[12] 7.64	[12] 31.6	[13] 24.9
Total United States mine production		1,995,079												
Percent of total		80.4												
Smelting and refining companies:														
American Metal	(3)	491,712	(3)	[14] 33,405,286	(3)	6.8	(3)	34,335,538	(3)	7.0	(3)	(3)	13.8	(3)
American Smelting & Refining	[16] 4,620,185	1,238,796	1,290,000	[15] 122,432,472	[16] 26.50	9.9	9.5	67,445,265	[16] 14.60	5.4	5.2	[16] 41.10	15.3	14.7
Total	(3)	1,730,508	(3)	155,837,758	(3)	9.0	(3)	101,780,803	(3)	5.9	(3)	(3)	14.9	(3)
Total United States smelter production		2,201,930												
Percent of total		78.6												
Foreign mining companies:														
Andes Copper	8,000,000	192,664	200,000	49,746,353	6.22	25.8	24.9	1,656,007	.21	.9	.8	6.43	26.7	25.7
Cerro de Pasco	(3)	99,986	100,000	[17] 86,759,629	(3)	86.8	86.8	30,186,973	(3)	30.2	30.2	(3)	117.0	117.0
Chile Copper	11,000,000	299,576	375,000	59,611,427	5.42	19.9	15.9	19,280,177	1.75	6.4	5.1	7.17	26.3	21.0
Granby Consolidated	310,716	60,854	(3)	17,332,666	55.78	28.6	(3)	5,854,819	18.84	9.6	(3)	74.62	38.2	(3)
Greene Cananea	(3)	58,827	(3)	[18] 52,532,103	(3)	89.3	(3)	2,617,907	(3)	4.5	(3)	(3)	93.8	(3)
Hudson Bay	1,090,596	31,068	(3)	[19] 30,903,984	28.34	99.5	(3)	1,346,984	1.24	4.3	(3)	29.58	103.8	(3)

Kennecott (Braden and Alaska)	5,320,474	204,508	275,000	[20] 224,002,361	42.10	109.5	81.5	38,787,417	7.29	19.0	14.1	49.39	128.5	95.6
Noranda Mines, Ltd	428,221	51,224	65,000	[21] 10,418,203	24.33	20.3	16.0	5,625,502	13.14	11.0	8.7	37.47	31.3	24.7
Katanga	(3)	302,012	(3)	[22] 19,435,136	(3)	6.4	(3)	20,285,432	(3)	6.7	(3)	(3)	13.1	(3)
Total	26,150,007	1,300,719	1,015,000	550,741,862	[23] 6.56	[23] 22.9	[24] 19.0	125,641,218	[23] 1.39	[23] 4.8	[24] 3.6	[23] 7.95	[23] 27.7	[24] 22.6
Total foreign and United States mine production		2,905,182												
Total world mine production		4,298,970												
Percent of total mine production		67.6												

[1] Investment in mine development, plant, and equipment includes book valuation of these items as of Dec. 31, 1929, plus deferred charges of that date plus accumulated depreciation.

[2] Total investment represents sum of investment in mine, development, plant, and equipment plus working capital.

[3] No data available.

[4] Anaconda Copper item under mine development, etc., represents Dec. 31, 1929, book valuation of buildings and machinery plus deferred charges, as shown in comparative consolidated balance sheet of the company, less items credited in this table to Andes, Chile, and Cananea. For Anaconda Copper accumulated depreciation is not taken into account. No data available on this subject.

[5] Calumet & Hecla item under mine development, etc., includes mine lands and plant plus deferred charges and accumulated depletion and depreciation as of Dec. 31, 1929. No data available showing value of mine development, plant, and equipment separately.

[6] Isle Royale Copper item under mine development, etc., is book valuation as of Dec. 31, 1929, of real estate and construction plus deferred charges plus accumulated depletion and depreciation. No data available for plant and equipment alone.

[7] Magma Copper item under mine development, etc., represents value of mine claims and plant as of Dec. 31, 1929, without addition of accumulated depletion and depreciation. No other data available.

[8] Mohawk item under mine development, etc., represents value of item shown under property account as of Dec. 31, 1929, and since accumulated depletion and depreciation was not shown it could not be added. Also, no separate data as to value of plant and equipment are available.

[9] Mother Lode Coalition does not show accumulated depreciation on development and equipment in 1929, so figures for 1923 for this item are used, as this is the first year after operations were started in 1919 in which this item was shown separately.

[10] Old Dominion item under mine development, etc., shows valuation of property and equipment as of Dec. 29, 1929, without the addition of accumulated depletion and depreciation. No data available as to value of plant and equipment alone or as to accumulated depletion and depreciation.

[11] Utah Copper item under mine development, etc., represents book valuation as of Dec. 31, 1929, of mining and mill property and equipment plus accumulated depletion and depreciation plus deferred charges.

[12] These average figures take into account only the 7 companies (Calumet & Arizona, Inspiration, Miami, Mother Lode, Nevada Consolidated, Phelps Dodge, and United Verde Extension) for which data as to value of mine development, equipment, etc., were available separated from value of mine. Total figures for these 7 companies were as follows: Ore mined, 33,129,423 tons; actual output, 803,097,000 pounds; investment, $168,460,037; and working capital, $84,750,445.

[13] These average figures on rated capacity take into account only 5 companies (Calumet & Arizona, Inspiration, Miami, Nevada Consolidated, Phelps Dodge) for which information as to capacity was available. Such information was not obtainable for the 2 other of the 7 companies used for average figures. Total figures used for these five companies were: Rated capacity, 925,000,000 pounds of copper; investment, $150,649,056; and working capital, $79,338,186.

[14] American Metal item under mine development, etc., represents book valuation of mines, smelters, etc., as of Dec. 31, 1929, plus deferred charges plus accumulated depletion and depreciation. No detailed data available.

[15] American Smelting & Refining item under mine development, etc., represents property account book valuation as of Dec. 31, 1929, with depreciation written off. No data available as to amount of depletion and depreciation.

[16] Figures represent ore smelted by American Smelting & Refining.

[17] Cerro de Pasco item under mine development, etc., is mines and properties book valuation as of Dec. 31, 1929, plus accumulated depletion and depreciation. No other data of this nature available.

[18] Greene Cananea item under mine development, etc., represents Dec. 31, 1929, book valuation of property account after depreciation. No other data available.

[19] Hudson Bay item under mine development, etc., is book valuation, Dec. 31, 1931, of claims, building, and machinery plus deferred charges. No data available as to accumulated depletion and depreciation nor as to value of machinery and equipment. Year 1931 used for this company, as it was the first year the mine was in full production.

[20] Kennecott (Braden and Alaska) item under mine development, etc., represents the book valuation as of Dec. 31, 1929, of property and equipment plus stripping and mine development plus other deferred charges plus accumulated depreciation and depletion from the consolidated statement of the Kennecott company less similar items representing the amount accounted for under Utah Copper.

[21] Noranda Mines, Ltd., item under mine development, etc., represents book valuation of property and equipment as of Dec. 31, 1929, plus investment in hotels, etc., plus deferred charges. No data available as to accumulated depletion and depreciation, nor as to value of plant, equipment, and mine development.

[22] Katanga item under mine development, etc., represents Dec. 31, 1929, book valuation of property, equipment, etc., without the addition of accumulated depletion and depreciation, as no data are available in existing records for this last item.

[23] These average figures take into account those 3 (Andes, Chile, and Granby) foreign companies for which detailed information regarding investment in mine development, etc., is available. For these 3 companies total figures were as follows: Ore mined, 19,310,716 tons; actual output, 553,090,000 pounds; investment, $126,700,446; and working capital $26,791,003.

[24] These average figures on rated capacity take into account only 2 (Chile and Andes) companies, as no data as to rated capacity are available for the third foreign company included in other foreign average figures. Total figures were: Rated capacity, 575,000,000 pounds; investment, $109,367,780; and working capital, $20,936,184.

TABLE 66.—*Investment in copper-producing companies—Dec. 31, 1929*

	Ore mined (tons)	Actual output of copper (thousands of pounds)	Rated capacity (thousands of pounds)	Fixed investment [1]				Working capital				Total investment [2]		
				Total (dollars)	Per ton of ore mined (dollars)	Per pound of actual output (cents)	Per pound of rated capacity (cents)	Total (dollars)	Per ton of ore mined (dollars)	Per pound of actual output (cents)	Per pound of rated capacity (cents)	Per ton of ore mined (dollars)	Per pound of actual output (cents)	Per pound of rated capacity (cents)
Mining companies (United States):														
Anaconda Copper	([3])	281, 938	324, 000	303, 892, 193	([3])	107. 8	93. 8	75, 654, 978	([3])	26. 8	23. 4	([3])	134. 6	117. 2
Calumet & Arizona	4, 428, 746	127, 140	140, 000	71, 435, 482	16. 13	56. 2	51. 0	8, 485, 890	1. 92	6. 7	6. 1	18. 05	62. 9	57. 1
Calumet & Hecla	2, 444, 000	134, 694	140, 000	40, 111, 289	16. 42	29. 8	28. 7	13, 966, 465	5. 71	10. 4	10. 0	22. 13	40. 2	38. 7
Inspiration Consolidated	5, 773, 858	107, 308	120, 000	40, 802, 073	7. 07	38. 0	34. 0	5, 272, 785	. 91	4. 9	4. 4	7. 98	42. 9	38. 4
Isle Royale Copper	514, 024	10, 864	11, 000	4, 305, 693	10. 32	48. 8	48. 2	1, 441, 174	2. 80	13. 3	13. 1	13. 12	62. 1	61. 3
Magma Copper	269, 579	38, 256	48, 000	3, 516, 731	13. 05	9. 2	7. 3	5, 094, 664	18. 90	13. 3	10. 6	31. 95	22. 5	17. 9
Miami Copper	5, 017, 983	58, 841	75, 000	22, 152, 771	4. 41	37. 6	29. 5	6, 101, 791	1. 22	10. 4	8. 1	5. 63	48. 0	37. 6
Mohawk	637, 159	20, 044	([3])	13, 649, 229	21. 42	68. 1	([3])	3, 632, 735	5. 70	18. 1	([3])	27. 12	86. 2	([3])
Mother Lode Coalition	57, 177	12, 242	([3])	1, 484, 552	25. 96	12. 1	([3])	591, 911	10. 35	4. 8	([3])	36. 31	16. 9	([3])
Nevada Consolidated	12, 888, 289	266, 274	350, 000	61, 702, 569	4. 79	23. 2	17. 6	27, 236, 023	2. 11	10. 2	7. 8	6. 90	33. 4	25. 4
Old Dominion	415, 891	18, 944	([3])	17, 711, 983	42. 59	93. 5	([3])	2, 257, 756	5. 43	11. 9	([3])	48. 02	105. 4	([3])
Phelps Dodge	4, 604, 716	167, 180	240, 000	205, 024, 694	44. 52	122. 6	85. 4	32, 241, 698	7. 00	19. 3	13. 4	51. 52	141. 9	98. 9
United Verde Extension	358, 654	64, 112	([3])	10, 474, 478	29. 20	16. 3	([3])	4, 820, 347	13. 44	7. 5	([3])	42. 64	23. 8	([3])
Utah Copper	17, 724, 100	296, 626	375, 000	66, 118, 181	3. 73	22. 3	17. 6	26, 761, 374	1. 51	9. 0	7. 1	5. 24	31. 1	24. 7
Total	55, 134, 176	1, 604, 463	1, 823, 000	863, 381, 918	[4] 10. 15	53. 8	[4] 45. 0	213, 559, 591	[4] 2. 68	13. 3	[4] 11. 1	[4] 12. 83	67. 1	56. 1
Total United States mine production		1, 995, 079												
Percent of total		[3] 80. 4												
Smelting and refining companies:														
American Metal	([3])	491, 712	([3])	41, 527, 134	([3])	8. 4	([3])	34, 335, 538	([3])	7. 0	([3])	([3])	15. 4	([3])
American Smelting & Refining	[5] 4, 620, 185	1, 238, 796	1, 290, 000	150, 245, 698	32. 52	12. 1	11. 6	67, 445, 265	14. 60	5. 4	5. 2	47. 12	17. 5	16. 8
Total	4, 620, 185	1, 730, 508	1, 290, 000	191, 772, 832	[4] 32. 52	11. 1	[4] 11. 6	101, 780, 803	[4] 14. 60	5. 9	[4] 5. 2	[4] 47. 12	17. 0	[4] 16. 8
Total United States smelter production		2, 201, 930												
Percent of total		78. 6												
Foreign mining companies:														
Andes Copper	8, 000, 000	192, 664	200, 000	86, 561, 820	10. 82	44. 9	43. 3	1, 656, 007	0. 21	0. 9	0. 8	11. 03	45. 8	44. 1
Cerro de Pasco	([3])	99, 986	100, 000	28, 151, 003	([3])	28. 2	28. 2	30, 186, 973	([3])	30. 2	30. 2	([3])	58. 4	58. 4
Chile Copper	11, 000, 000	299, 576	375, 000	144, 203, 234	13. 11	48. 1	38. 5	19, 280, 177	1. 75	6. 4	5. 1	14. 8	54. 5	43. 6
Greene Cananea	([3])	58, 827	([3])	52, 665, 011	([3])	89. 5	([3])	2, 617, 907	([3])	4. 5	([3])	([3])	94. 0	([3])

Granby Consolidated	310, 716	60, 854	(3)	8, 775, 195	28. 24	14. 4	(3)	5, 854, 819	18. 84	9. 6	(3)	47. 08	24. 0	(3)
Hudson Bay	1, 090, 596	31, 068	(3)	30, 913, 985	28. 35	99. 5	(3)	1, 346, 984	1. 24	4. 3	(3)	25. 59	103. 8	(3)
Kennecott (Braden and Alaska)	5, 320, 474	204, 508	275, 000	282, 277, 179	53. 05	138. 0	102. 6	38, 787, 417	7. 29	19. 0	14. 1	60. 34	157. 0	116. 7
Noranda Mines, Ltd	428, 221	51, 224	65, 000	12, 497, 096	29. 18	24. 4	19. 2	5, 625, 502	13. 14	11. 0	8. 7	42. 32	35. 4	27. 9
Katanga	(3)	302, 012	(3)	24, 204, 600	(3)	8. 0	(3)	20, 285, 432	(3)	6. 7	(3)	(3)	14. 7	(3)
Total	26, 150, 007	1, 300, 719	1, 015, 000	670, 249, 103	[4] 21. 61	51. 5	[4] 54. 6	125, 641, 218	[4] 2. 77	9. 7	9. 4	[4] 24. 38	61. 2	[4] 64. 0
Total foreign and United States		2, 905, 182												
Total world mine production		4, 298, 970												
Percent of world total		67. 6												

[1] Represents difference between total assets and current assets.
[2] Represents fixed assets plus working capital.
[3] No data available.
[4] Companies which show no data available are not considered in this average.
[5] Ore smelted.

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