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GOLD MINING AND MILLING IN THE UNITED STATES AND CANADA

Current Practices and Costs

BY

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FOREWORD

This bulletin deals with the prospecting, development, mining, and milling of lode-gold ores, and contains a brief discussion of placer mining. It is the first of a series of summary bulletins, which will deal particularly with production methods, as well as costs per ton of the different metallic ores mined, and per unit of metal recovered. During periods of reduced demand and lowered prices for base metals, interest in gold mining invariably revives. This paper is therefore of timely interest.

Some of the details here summarized have already been published by the bureau during the past three years, in various information circulars which dealt with practices and costs at each of a large number of properties. Generally, those circulars were written by men in charge of operations, acting as special consultants to the bureau. To details thus accumulated have been added facts secured by field engineers of the bureau at these and other properties, supplemented by data from the bureau files, company reports, and the technical press. A bulletin devoted to each metal will be published, in which will be consolidated the pertinent facts regarding exploitation and operation, and discussion of production trends, ore occurrence, and costs.

From comparative insignificance as a gold producer until 1848, the United States came to the forefront with the discovery of gold in California in that year and continued in first or second place among gold-producing countries until 1930, when, if the Philippine production is omitted, Canada took second place and increased its lead over the United States in 1931.

The United States production reached its peak in 1915, when 4,823,672 fine ounces of gold ($99,703,300) were produced exclusive of Philippine production. Since 1915, production steadily declined until 1931, when a small increase was recorded. In 1931, our most productive gold mine had a record of slightly less than $9,000,000.

Canada first assumed real importance as a gold-mining country during the Yukon rush in 1898, and since that date has become an increasingly important producer. In 1931, each of two Canadian lode mines produced gold in excess of $10,000,000.

Some lode deposits have been remarkably persistent in size and grade to great depths and have been profitably operated over a long period of years. Others have been of the bonanza type, and for a few years have yielded large quantities of gold from comparatively shallow depths.

The cost of producing an ounce of gold at different American mines varies between wide limits and depends upon a number of factors. Among these factors the grade of the ore mined is of importance, as well as size of operations, and physical conditions, such as type of deposit, continuity of ore, distribution of gold, and mineral association. One American mine, operating under conditions suited to wholesale methods, is now producing gold profitably from an ore which, over a period of years, has yielded less than $1 per ton in gold.

Most of the larger producers are mining ore yielding $6 to $12 per ton, but only a few are mining ore yielding over $12 per ton. At a number of properties operating costs per ounce of gold range from about $6.50 to $18.50, the average cost being roughly $10 to $12. If depreciation, taxes, overhead, and marketing are added, these costs range from about $8.30 to $19.50 and average roughly $12 to $14.

These are current costs at profitable mines, and it is obvious that if losses at nonprofitable operations and the cost of unsuccessful exploration for gold could be known and were added, the total cost of producing gold in the United States and Canada would be found to be much higher.

It is hoped that this bulletin will be found useful to mining men generally. Helpful suggestions would be welcomed by the bureau, and new or improved data might be incorporated in a later revised edition of this gold bulletin.

SCOTT TURNER, *Director.*

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GOLD MINING AND MILLING IN THE UNITED STATES AND CANADA: CURRENT PRACTICES AND COSTS[1]

By Charles F. Jackson[2] and John B. Knaebel[3]

INTRODUCTION

Gold mining is a subject that not only appeals to the popular imagination but has vital importance to the world's economic structure. However, a discussion of the use of gold as a medium of exchange and a basis for credit between nations and for monetary systems is not within the scope of this paper. The importance of gold in carrying on the commerce of the world, the present supply and rate of production, the probable trend of future production, the probable future requirements, and related subjects have recently received much attention from bankers and economists, and many articles on these subjects have been published.

During slack periods of base-metal production due to falling prices, accompanied by scaling down of base-metal mining operations and often by the closing down of at least some mines, there usually result revival of interest in gold mining, search for previously undiscovered deposits, and reopening of some old gold mines.

At present there is such revival of interest in gold mining, and it is hoped that this paper may contain information and suggestions of value in this connection.

OBJECT AND SCOPE OF PAPER

This paper attempts to assemble and discuss briefly in one volume a number of subjects relating to the mining of gold, particularly from lode deposits, in the United States and Canada.

The history of gold mining in the United States and Canada, the present rate of output, and the location of the principal gold-producing districts are touched upon very briefly. Then follows a general discussion of types of gold deposits, geological occurrence, mineralogical associations of the ores, and changes in lode deposits with depth. Next is a discussion of such subjects as methods of prospecting and exploration, methods and costs of mine development, stoping methods and costs, and total ore-production costs, followed by a résumé dealing with the milling of the ores, the recovery of gold therefrom, and the costs of milling.

No attempt has been made to deal exhaustively with any one of these related subjects, which would defeat the object of condensing

[1] Work on manuscript completed December, 1931.
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some of the essentials of each within a single volume. Numerous references are made throughout the discussions to more complete works on the various topics. Highly technical treatment and terminology have been avoided to increase the value of the paper to the general public.

ACKNOWLEDGMENTS

The authors desire to acknowledge the helpful criticism and suggestions offered by D. F. Hewett, of the United States Geological Survey, on the section dealing with geology. The section on gold milling was critically reviewed by Edward S. Leaver, of the Bureau of Mines. The cooperation of the mining companies which have furnished data is also gratefully acknowledged. The technical press and the publications of technical societies, as well as the annual reports of mining companies, have been drawn upon to supplement data obtained during field studies.

PRODUCTION OF GOLD IN UNITED STATES, ITS TERRITORIAL POSSESSIONS, AND CANADA

HISTORY

The history of gold production in the United States and Canada, as well as in all the principal producing regions of the world, has been briefly discussed in a recent report,[4] and detailed figures on the same subject have been prepared by Ridgway.[5] Inasmuch as a lengthy discussion of the statistical history of gold mining is not within the province of this bulletin, it is sufficient to say that from comparative insignificance as a gold producer until 1848 the United States came to the forefront with the discovery of the California placers in that year and ever since has remained in first or second place among the producing nations of the world (if the Philippine production is omitted, however, Canada surpassed the United States in 1930). Except for 20 years of gradual decline in output between 1855 and 1876, production in this country mounted continuously—though rather erratically—to an all-time peak of 4,823,672 fine ounces ($99,703,300) in 1915, exclusive of the Philippine contribution. The steady rise was due to the important discoveries of Cripple Creek, Colo.; Lead, S. Dak.; Alaska; Goldfield; and other famous districts which more than compensated for the declining output from older mining regions. Since 1915 production in the United States has turned downward and in 1930 was slightly exceeded by that from Canadian mines.

Canada first assumed real importance with the Yukon rush of 1898 and has become a factor of steadily increasing importance in the present century, chiefly because of the spectacular performance of a few mines at Porcupine and Kirkland Lake in northern Ontario. Canada will probably soon definitely surpass the United States as a producer of gold, and there is every reason to expect a continued high rate of production for some years.

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[4] Grant, R. J., and Knaebel, John B., Sources and Trends in Gold Production: Annual meeting, Am. Inst. Min. and Met. Eng., New York, Feb. 17, 1931.

[5] Ridgway, Robert H., Summarized Data of Gold Production: Ec. Paper 6, Bureau of Mines, 1929, 63 pp.

LOCATION AND STATUS OF PRODUCING REGIONS

The production of gold in 1929 is given by States and Provinces, and by counties or districts producing over $35,000, in Table 1. State, Province, and county totals for 1930 are included as well, and these afford some indication of present trends in each general mining region, although due regard must be given the influence of curtailed base-metal production in 1930, since considerable gold is derived from this source, particularly from large-scale copper-mining projects.

TABLE 1.—*Gold production in the United States, its Territories, and Canada for 1929*

[1930 totals for States, Provinces, and counties are included under the column headed "Remarks"]

UNITED STATES, ALASKA, AND PHILIPPINES

State	County	District	Number of lode mines	Metals in ore in order of value	Tons of ore mined or sold	Total value of ore produced	Value of lode gold	Number of placer mines	Value of placer gold	1929 total value of lode and placer gold	Remarks[1]
Arizona	Cochise	Tombstone	8	Pb, Ag, Au, Cu	15,601	$155,959	$34,530			$34,530	D.
	do	Warren	2	Cu, Ag, Pb, Au	2,660,479		124,003			124,003	2 mines only; represent most of district production.
	Gila	Banner	5	Cu, Pb, Ag, Au	144,346	1,347,648	33,072			33,072	D.
	do	Globe	20	Cu, Ag, Au, Pb	11,293,656	33,958,735	116,106			116,106	D.
	do	Summit	3	Cu, Pb, Ag, Au	11,439,598	35,334,152	149,588			149,588	D.
	Greenlee	Greenlee	3	Cu, Au, Ag	1,742,589	10,141,667	86,367			86,367	D.
	Maricopa	Vulture	2	Pb, Au, Cu, Ag	17,283	186,821	36,760			36,760	D.
	Mohave	San Francisco	1	Au	4,300	109,550	109,550			109,550	1 mine only; represents most of district production.
	Pima	Ajo	1	Cu, Au, Ag	3,578,403	12,884,356	304,723			304,723	Do.
	Pinal	Pioneer	1	Cu, Ag, Au	269,922	7,669,000	177,180			177,180	Do.
	Santa Cruz	Oro Blanco	1	Pb, Zn, Ag, Au, Cu.	34,130	525,630	35,659			35,659	D.
	Yavapai	Verde	3	Cu, Au, Ag	2,191,459	39,160,430	1,563,322			1,563,322	D.
	do	Weaver	1	Au, Ag, Cu, Pb	9,186	76,486	71,612			71,612	D.
All[2]				Cu, Au, Ag, Pb, Zn.	25,860,772	155,561,459	4,176,635	Sluice; dry washing	$5,652	4,182,287	1930 total, $3,073,500.
Idaho	Boise	Boise Basin	7	Au, Ag, Pb, Cu	21,779	211,691	206,667				D.
	Custer	Alder Creek	1	Cu, Au, Ag, Pb	66,573	577,475	45,650				1 mine; represents most of district production.
All[2]				Pb, Zn, Ag, Cu, Au.	2,174,125	31,018,116	333,172	(?) dredge; (?) sluice; (?) hydraulic.	85,373	418,545	1930 total, $438,200.
Washington	Whatcom	Mt. Baker	1	Au, Ag	4,694	55,349	55,274				D.
All[2]				Cu, Zn, Au, Pb, Ag.	93,597	546,067	70,784	(?) dredge; (?) some sluice.	6,114	76,898	1930 total, $75,400.
Utah	Juab and Utah	Tintic	[3] 30	Pb, Ag, Au, Cu, Zn.	455,316	10,545,407	1,017,293			1,017,293	D.
	Salt Lake	West Mt.	14	Cu, Pb, Au, Zn, Ag.	18,515,213	69,803,927	3,328,035			3,328,035	D.
	Summit	Uintah	[4] 5	Pb, Ag, Zn, Cu, Au.	452,398	6,022,345	81,726			81,726	D.
	Tooele	Rush Valley	5	Pb, Ag, Au, Zn, Cu.	61,662	1,923,546	124,806			124,806	D.

	Wasatch	Blue Ledge	3	Zn, Pb, Ag, Au, Cu.	170,970	3,242,231	291,150			291,150	Chiefly 1 mine.
	do	Snake Creek	2	Pb, Zn, Ag, Cu, Au.	154,541	3,945,196	55,488			55,488	D.
All[3]				Cu, Pb, Ag, Zn, Au.	19,831,975	95,984,237	4,968,959	3	956	4,969,915	1930 total, $4,319,100.
Oregon	Curry	Cheto	1	Au, Ag	1,097	37,899	37,731			37,731	1 mine, D.
	Grant	Canyon	1	do				1 dredge	35,176	35,176	Most of district.
	Jackson	Gold Hill	1	do				do	151,738	151,738	Do.
All[3]				do	10,509	238,083	106,354	3 dredges; 38 hydraulic; 67 sluice; 3 drift.	246,969	353,323	1930 total, $295,600.
California	Amador	Jackson, Sutter Creek, Plymouth, and Volcano.	[6] 14	do	226,445	1,554,874	1,545,650	1 dredge; 12 sluice; 2 hydraulic; 2 drift.	56,211	1,601,861	3 lode districts; county total.
	Calaveras	13, chiefly Esmerelda.	[6] 28	Cu, Au, Ag	29,944	291,094	77,609			77,609	3 copper mines, 13 districts; county total.
	Eldorado	8, chiefly Spanish Diggings.	17	Au, Ag	6,104	41,708	41,515			41,515	8 districts; county total.
	Kern	13, chiefly Randsburg.	30	do	30,694	150,043	147,736			147,736	13 districts; county total. (See San Bernardino County.)
	Mariposa	9, chiefly Coulterville.	[6] 18	Au, Cu, Ag	18,331	89,913	88,171			88,171	18 districts; county total.
	Nevada	5, chiefly Grass Valley.	13	Au, Ag, Cu, Pb	207,256	1,797,980	1,774,816			1,774,816	50 districts; county total.
	Plumas	6, chiefly Spring Garden and Engels.	9	Cu, Au, Ag	853,817	4,980,324	375,083			375,083	6 districts; county total.
	San Bernardino	19, chiefly Randsburg.	[6] 32	Ag, Au, Cu, Pb	18,441	251,571	44,305			44,305	19 districts. (See Kern County.)
	Sierra	4, chiefly Alleghany.	14	Au, Ag	41,387	357,310	355,560			355,560	4 districts; county total.
	Tuolumne	11, chiefly Sonora	[7] 39	Au, Ag, Cu	2,317	86,387	69,139			69,139	11 districts; county total.
	Butte							2 dredge; 34 sluice; 7 drift.	70,276	70,276	County total.
	Merced	Snelling						1 dredge	84,188	84,188	Do.
	Sacramento	Folsom and Natomas.						8 dredge	1,492,083	1,492,083	2 districts; most of county.
	Shasta	Redding						1 dredge	61,161	61,161	Most of county.
	Siskiyou							61 hydraulic; 11 sluice	42,540	42,540	County total.
	Stanislaus	La Grange						1 dredge	123,002	123,002	Most of county.
	Trinity	Eastman Gulch and Lewiston.						3 dredge; 27 hydraulic; 10 sluice.	326,957	326,957	Chiefly 2 dredges; county total.
	Yuba	Chiefly Marysville.						7 dredge; 18 sluice; 1 hydraulic.	1,449,758	1,449,758	Chiefly 6 dredges; county total.
All[3]			324	Au, Cu, Ag, Pb	1,657,069	11,212,887	4,656,096	478	3,870,607	8,526,703	1930 total, $9,308,300.

See footnotes at end of table.

Table 1.—*Gold production in the United States, its Territories, and Canada for 1929*—Continued

[1930 totals for States, Provinces, and counties are included under the column headed "Remarks"]

UNITED STATES, ALASKA, AND PHILIPPINES

State	County	District	Number of lode mines	Metals in ore in order of value	Tons of ore mined or sold	Total value of ore produced	Value of lode gold	Number of placer mines	Value of placer gold	1929 total value of lode and placer gold	Remarks
Colorado	Boulder			Au, Ag, Pb	6,176	39,976	33,528			33,528	County total.
	Clear Creek	Empire	7	Au, Pb, Ag	11,028	35,592	35,460			35,460	D.
	do	Idaho Springs	19	Au, Ag, Pb, Cu, Zn	5,685	82,104	52,625			52,625	D.
	Eagle	Battle Mountain	1	Zn, Cu, Ag, Au, Pb.	54,627	1,193,343	71,703			71,703	D.
	Gilpin	Southern	2	Au, Ag, Cu, Pb, Zn.	57,249		99,670			99,670	2 mines; most of district.
	La Plata	California	7	Au, Ag, Pb, Zn	19,950	165,128	116,360			116,360	D.
	Ouray	Sneffels	2	Au, Ag, Pb, Cu, Zn.	7,159	378,163	363,285			363,285	D; chiefly 1 mine.
	Park	Mosquito		Au, Ag, Pb, Zn, Cu.	3,449	218,119	205,889			205,889	Do.
	San Juan	Animas	12	Au, Cu, Ag, Pb, Zn.	37,974	420,970	180,469			180,469	Do.
	do	Eureka	1	Pb, Zn, Ag, Cu, Au.	308,892	3,560,997	250,935			250,935	D; 1 mine.
	San Miguel	Upper San Miguel.	1				308,892			308,892	D; Smuggler-Union mine, now shut down.
	Teller	Cripple Creek		Au, Ag	288,590	2,644,961	2,640,034			2,640,034	D.
	Summit	Breckenridge						3 dredge; 1 power shovel; sluicing.	40,719	40,719	Chiefly 1 dredge; district and county total.
All[2]				Au, Zn, Pb, Ag, Cu.	1,172,193	15,293,343	4,417,358		45,850	4,463,208	1930 total, $4,511,800.
Montana	Lewis & Clark	Helena	1	Au, Ag, Cu	40,178	183,222	182,427			182,427	D; 1 mine.
	do	Ottawa (Marysville).	10	Au, Ag, Cu	5,279	65,631	47,886			47,886	D; chiefly 1 mine.
	Madison	Mineral Hill		Au, Cu, Ag	31,712	206,700	178,755			178,755	1 mine; most of district.
	Silver Bow	Summit Valley	([6])	Cu, Zn, Ag, Pb, Au.	4,271,207	65,687,609	538,419			538,419	D.
All[2]				Cu, Zn, Ag, Pb, Au.	4,723,445	71,767,175	1,119,615	Sluice; hydraulic; drifting.	12,334	1,131,999	1930 total; $969,500.
Nevada	Elko	Gold Circle	4	Au, Ag	13,913	164,097	120,078			120,078	D; chiefly 1 mine.
	do	Jarbidge	3	Au, Ag	51,209	672,627	581,741			581,741	Do.

	Esmeralda	Goldfield		Au, Ag		269,018	263,947			263,947	D; chiefly tailings.
	Esmeralda and Nye.	Tonopah	14	Ag, Au	121,447	1,462,291	414,628			414,628	D.
	Nye	Manhattan	10	Au, Ag	7,013	116,850	116,662			116,662	D; chiefly 1 mine.
	Eureka	Eureka	3	Pb, Au, Ag, Cu	4,981	103,677	37,611			37,611	Do.
	Storey	Comstock	13	Au, Ag, Cu	11,244	127,608	109,879			109,879	Do.
	White Pine	Robinson		Cu, Au, Ag, Pb	6,378,138	24,447,782	1,321,640			1,321,640	D; chiefly 2 mines.
	Nye	Round Mountain						1 hydraulic	32,517	32,517	D; chiefly 1 mine.
All [2]				Cu, Au, Ag, Pb, Zn.	7,355,500	32,985,929	3,340,449	15 hydraulic; drifting	43,762	3,384,211	1930 total, $2,898,-600.
New Mexico	Grant	Central	21	Cu, Zn, Pb, Ag, Au.	4,135,864	17,677,723	131,631			131,631	D; chiefly 1 mine.
	Hidalgo	Lordsburg	7	Cu, Au, Ag, Pb	83,091	1,055,597	219,359			219,359	D.; chiefly 2 mines.
	San Miguel	Willow Creek	1	Zn, Pb, Cu, Au, Ag.	216,809	4,798,090	343,064			343,064	D.; entirely 1 mine.
All [2]				Cu, Zn, Pb, Au, Ag.	4,506,807	24,472,023	725,512	Sluicing; some hand rocking.	1,650	727,162	1930 total, $653,000.
South Dakota	Laurence	Whitewood	1	Au, Ag	1,437,935	6,536,482	6,491,184				D.
	Bennington	Keystone	1	do	25,224	58,519	58,415				D.
All [2]			2	do	1,463,159	6,595,001	6,549,599	(?)	(?)		1930 total, $8,398,-900.
Eastern States.[9]				Zn, Cu, Pb, Ag, Au.	4,345,149	28,803,959	33,577			36,793	1930 total, $42,200.
Alabama, Georgia, and North Carolina.								Sluicing	3,216		
Alaska		Juneau	1	Au, Pb, Ag	3,836,440		3,399,743	(?)	(?)	3,399,743	D.; 1 mine.
		Sitka	1	Au, Ag, Pb	2,100		54,634	(?)	(?)	54,634	Do.
		Fairbanks	5	Au, Ag	14,257		97,906	(?)	(?)	97,906	D.
All					3,900,000 (gold ore only).		3,644,000	(?)	4,117,000	7,761,000	1930 total, $8,420,-800.
Philippines											1930 total, $3,828,-600.
Texas [2]				Ag, Cu, Pb, Au	63,872	683,920	26,439			26,439	1930 total, $180,463.
Total									46,651,400		1930 total, $47,247,-600.

See footnotes at end of table.

TABLE 1.—*Gold production in the United States, its Territories, and Canada for 1929*—Continued

[1930 totals for States, Provinces, and counties are included under the column headed "Remarks"]

CANADA

Province	District	Metals in ore in order of value	Value of lode gold	Value of placer gold	Total value of lode and placer gold	Remarks
Ontario	Porcupine district	Au, Ag	$19,900,000		$19,900,000	1930 total, $17,822,365.
	Kirkland Lake district	do	13,900,000		13,900,000	1930 total, $17,231,709.
	Other gold districts	do				1930 total, $464,788.
	Base-metal ores					
All		Au, Cu, Ag	33,500,000		33,500,000	1930 total, $35,518,862.
British Columbia			3,004,419	$118,711	3,123,130	1930 estimate: Placer, $154,700; lode, $3,183,457; total, $3,338,157.
Quebec	Chiefly Rouyn	Cu, Au	[3] 798,863		[10] 798,863	First half 1930, $1,078,780.
Manitoba		do				1930 total, $206,907.
Yukon				[10] 151,284		First half 1930, $121,788.
Nova Scotia					[11] 47 47,541	1930 total, $36,005.[4]
Total					39,840,722	1930 total, $43,199,000.[4]

[1] D means district.
[2] These are State totals, including districts producing less than $35,000.
[3] 4 clean-ups.
[4] 5–2 tails.
[5] 3 tails.
[6] 1 clean-up.
[7] 2 tails.
[8] Many.
[9] Alabama, Georgia, North Carolina, Pennsylvania, Tennessee.
[10] First 6 months.
[11] Estimated.

TABLE 2.—*Location and production of important gold mines in the United States and Canada* [1]

1. LODE-GOLD MINES

Mine	Location	Type of deposit	Bullion production		Remarks
			Dollars	Year	
Hollinger	Ontario	Lenticular veins in basaltic schist.	10,264,504	1930	1931 production, $10,097,975; total to end of 1931, $177,237,489.
Lake Shore	do	Veins in fault zones in syenite, porphyry, and lamprophyre.	9,152,935	[2] 1931	1931 production, $11,065,618.
Homestake	South Dakota	Wide replacement bodies in folded beds of dolomite and schist.	8,426,195	1930	Total, 1879 to 1931, inclusive, $263,704,304.
Teck-Hughes	Ontario	Vein in fault zone in syenite, porphyry, and lamprophyre.	5,973,120	[3] 1931	
McIntyre	do	Lenticular veins in basaltic schist.	4,633,328	[4] 1931	Calendar year 1931, $4,756,880.
Dome	do	Lenticular veins in metamorphosed sediments and greenstone.	3,914,883	1928	Year, 1931, $3,512,066.
Alaska-Juneau	Alaska	Stringer lodes in broad shear zones in slate and metagabbro.	3,375,659	1930	Year 1931, $3,710,927.
Wright-Hargreaves.	Ontario	Veins in syenite porphyry	2,428,008	1930	Year, 1931, $2,909,837.
Premier	British Columbia.	Lenticular veins in shear zones in tuffs and greenstone.	[5] 1,997,000	1929	2,258,729 ounces silver not included.
Empire-Star	California	Veins in granodiorite	1,637,401	1930	
Vipond	Ontario	Lenticular veins in basaltic schist.	907,141	1930	Year 1931, $564,291.
Sylvanite	do	Veins in syenite porphyry	791,803	1930	Year 1931, $901,168.
Coniaurum	do	Veins in basaltic schist	737,233	1930	Year 1931, $785,708.
Kirkland Lake Gold.	do	Lenticular veins in lamprophyre, syenite, and porphyry.	582,583	1930	9 months, 1931, $438,750.
Tom Reed	Arizona	Veins in andesite and on andesite-tuff contact.	539,115	1930	Does not include custom ore milled.
Portland	Colorado	Fissure veins in volcanic breccias	467,755	1929	From Mines Handbook.
Three companies.	Cripple Creek, Colo.	do	1,739,844	1930	
Howey	Ontario	Portions of sheared porphyry dike.	457,810	1930	Year 1931, $866,606.
Central-Eureka	California	Veins in slates and greenstone	489,453	1930	From Mines Handbook.
Siscoe	Quebec	Veins in granodiorite			Year 1931, $743,747.
Elkoro	Nevada	Quartz veins in lava flows	458,000	1930	
Pioneer	British Columbia.	Quartz veins in diorite and albitite.	350,000	[4] 1931	Current rate, $50,000 per month.
Three mines, Mother Lode district.	California	Veins in slate and greenstone	1,613,476	1930	Three mines.
Nevada City, Grass Valley, and Alleghany districts.	do	Veins in granodiorite, diabase, and schists.	2,285,707	1930	Do.
Nine mines	Nevada	Various	1,612,429	[6] 1930	

2. PLACER MINES

Twelve companies.	California	Placer	$3,451,801	1930	Combined output 12 dredging companies.[7]
Five companies	Oregon	do	174,470	1930	Combined production 5 companies.[8]

[1] List not complete as some companies do not publish production figures. In some cases value includes a little silver.
[2] Year ended June 30.
[3] Year ended Aug. 31.
[4] Year ended Mar. 31.
[5] Approximate.
[6] One mine over $400,000, 3 mines $200,000 to $300,000 each, 5 mines $50,000 to $100,000 each.
[7] Two companies over $1,000,000 each, one company between $300,000 and $400,000, four companies between $100,000 and $200,000 each, five companies less than $100,000 each.
[8] From $20,000 to $80,000 each.

TABLE 2.—*Location and production of important gold mines in the United States and Canada*—Continued

3. BASE-METAL MINES PRODUCING IMPORTANT AMOUNTS OF GOLD

Mine	Location	Type of deposit	Bullion production		Remarks
			Dollars	Year	
Noranda	Ontario	Large lenses of massive sulphides in altered rhyolite cut by dikes of quartz diorite and gabbro.	$2,423,700	1930	Year 1931, $5,237,000.
Utah Consolidated Copper Co.	Utah	Sulphides disseminated in porphyry.	[9] 1,284,798	1930	
Nevada Consolidated Copper Co.	Nevada, Arizona, New Mexico.	do	[9] 647,343	1930	Gold and silver.
Calumet and Arizona Mining Co.	Arizona, New Mexico.	Various	[9] 879,573	1930	
Park Utah	Utah	Fissure veins in quartzite and limestone.	[9] 365,425	1930	
Consolidated Coppermines.	Nevada	Disseminated sulphides in porphyry.	[9] 287,676	1930	
Granby Consolidated Mining Co.	British Columbia.	1. Copper sulphides ores in greenstone and on contact. 2. Porphyry.	[9] 218,234	1929	
Tintic Standard.	Utah	Lead-silver ores in limestone. Irregular bodies.	[9] 75,593	1930	
Silver King-Coalition.	do	Complex ores in fissures and in form of bedded replacements.	[9] 72,332	1930	

[9] From published annual reports.

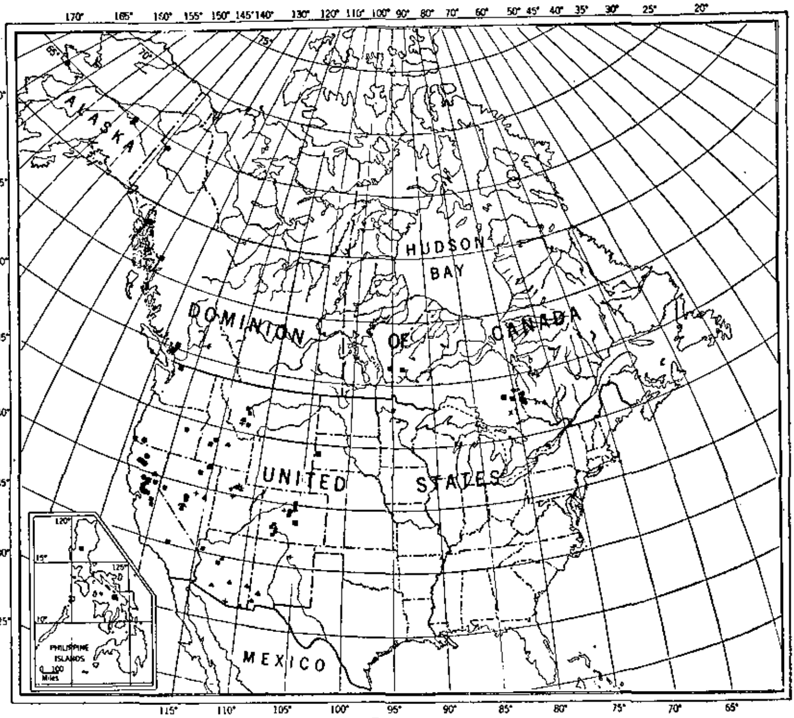
The location and yearly production of some of the important individual gold mines are presented in Table 2. Other important mines do not publish their annual production, and the list is therefore incomplete.

The figures employed in preparing Table 1 were derived from reports of the director of the United States Mint, from the United States Bureau of Mines, from the Canadian Department of Mines, and Dominion Bureau of Statistics. Table 2 is based on the annual reports of the companies and on authoritative information from other sources.

LOCATION OF MINES

Except for its occurrence in association with the base metals, where it is won as a by-product of base-metal mining operations, gold is usually found in a form amenable to recovery from the associated gangue by means which require comparatively small tonnages of fuel and supplies. Furthermore, it can usually be extracted and converted into bullion or into concentrates of very high tenor at the mine, so that shipment of product is confined to small tonnage and bulk. Gold mining is therefore possible in regions and under conditions where freight and haulage rates (for incoming supplies and outgoing product) would make it impossible to mine profitably base-metal ores, coal, and nonmetallic ores of equal dollar value at the market.

The discovery of gold in remote regions thus has a better chance, as a rule, for subsequent successful exploitation than would the discovery of deposits containing the base metals, size of the deposit and value at the market being comparable. It is not strange, therefore, that the quest for gold and the exploitation of gold deposits have been the advance guard in pushing back the fringes of civilization and in the development of natural resources.



LEGEND

● Placer mine. ■ Lode mines dry and siliceous gold ores. ▲ Lodes producing appreciable gold, principal revenue from copper. + Lodes producing appreciable gold, principal revenue from lead and zinc. < Lodes producing appreciable gold, principal revenue from silver. × Lodes producing appreciable gold, principal revenue from nickel and copper

FIGURE 1.—Map showing location of principal gold-producing districts in the United States and Canada, 1930–31

The accompanying map (fig. 1) shows the location of the principal gold-mining districts in the United States and its Territorial possessions and in Canada, with the principal base-metal mining districts producing important amounts of gold as a by-product from base-metal mining.

Part 1.—GEOLOGY

TYPES OF GOLD DEPOSITS

Before the subject of gold mining is discussed a brief review of the types of gold deposits of the United States and Canada is desirable, since the nature of the deposit determines the methods of prospecting, exploration, development, and mining, and the treatment of the ore, and affects the mining and milling costs.

The workable deposits of gold are of two principal types: 1, Placer deposits; and 2, lode deposits.

PLACER DEPOSITS

Placer deposits are of two general types, residual placers and transported placers; they result from the disintegration of lode deposits by weathering and erosional forces.

Residual placers are relatively unimportant and result simply from the disintegration of gold-bearing rock without transportation of the material from its original location.

Transported placers result from the disintegration and erosion of lode deposits, followed by transportation of the resulting débris (principally by running water), sorting and segregation of the gold and other heavy minerals by the action of moving water, and finally deposition of the gold in places where the velocity, and hence the carrying power, of the water was reduced.

Briefly then, weathering and other erosional forces break down the lode material and the surrounding rocks, and the débris moves down the slopes toward the stream beds and is carried along by the streams. The lightest and finest material is quickly washed out and carried away while the heaviest material, including gold, and the largest material are deposited in the stream channels where the velocity is sufficiently reduced. The largest and heaviest material will obviously be deposited first (that is, nearest its source in the lode) and the lightest and finest material will be transported farther.

Re-sorting of placers may occur due to changes in stream courses or in the volume and velocity of the currents; sometimes they may be worked over several times by natural agencies before they reach a final resting place.

CLASSIFICATION

Placer deposits have been classified by Brooks[1] as follows:

Creek placers.—Gravel deposits in the beds and intermediate flood plains of small streams.

Bench placers.—Gravel deposits in ancient stream channels and flood plains which stand from 50 to several hundred feet above the present streams.

[1] Brooks, A. H., Outline of Economic Geology; The Gold Placers of Parts of Seward Peninsula: U. S. Geol. Survey Bull. 328, 1908, pp. 114–145.

Hillside placers.—A group of gravel deposits intermediate between the creek and bench placers. Their bedrock is slightly above the creek bed, and the surface topography shows no indication of benching.

River-bar placers.—Placers on gravel flats in or adjacent to the beds of large streams.

Gravel-plain placers.—Placers found in the gravels of the coastal or other lowland plains.

Sea-beach placers.—Placers reconcentrated from the coastal-plain gravels by the waves along the seashore.

Ancient beach placers.—Deposits found on the coastal plain along a line of elevated beaches.

Lake-bed placers.—Placers accumulated in the beds of present or ancient lakes that were generally formed by landslides or glacial damming.

Buried placers.—Ancient placers of the above types have sometimes become buried by thick alluvial or detrital deposits which may or may not have become solidified or by lava flows of later geological age, giving rise to buried placers which may be at considerable depth below the present surface. Some of these have been worked by what is termed "drift mining" underground, especially in California, Oregon, Idaho, and Alaska.

The gold in transported placer deposits commonly is found concentrated on the underlying bedrock and in fractures in the bedrock itself but may occur at several horizons representing different stream-bed levels or be more or less disseminated through a considerable vertical range. In decomposed or fractured bedrock the gold often works down in cracks and crevices in the rock, sometimes for several feet.

The map (fig. 1) shows the location of the principal placer deposits which have been worked in the United States and Canada.

LODE DEPOSITS

A lode deposit has been defined as "strictly a fissure in the country rock filled with mineral; usually applied to metalliferous lodes. In general miners' usage, a lode, vein, or ledge is a tabular deposit of valuable mineral between definite boundaries."[2]

The term is also employed in a still broader sense to quartz or other rock in place which carries valuable mineral, and in the present pages the term is used in this broad sense.

Lode deposits of the United States and Canada carrying gold as the principal valuable mineral are of several types. Thus we have tabular deposits (wide in two dimensions and narrow in the other) of the fissure-vein type (fig. 2) and other tabular deposits of the lens type occurring in schistose (usually pre-Cambrian) rocks (fig. 3). Figure 4 shows quartz stringers in a vein at Grass Valley, Calif., typical of the fissures in that district.

Another form of lode deposit found in the United States is the wide and homogeneous pitching deposit typified by the Homestake (S. Dak.) ore body, which is a replacement in a calcareous bed in a

[2] Fay, Albert H., Glossary of the Mining and Mineral Industry: Bull. 95, Bureau of Mines, 1920, pp. 405–406.

series of ancient schistose rocks and which is formed around the nose of a faulted plunging anticlinal structure.[3] (Fig. 6.)

Another type of gold deposit is the broad shear zone in which the gold occurs in a series of stringer lodes made up of veinlets and

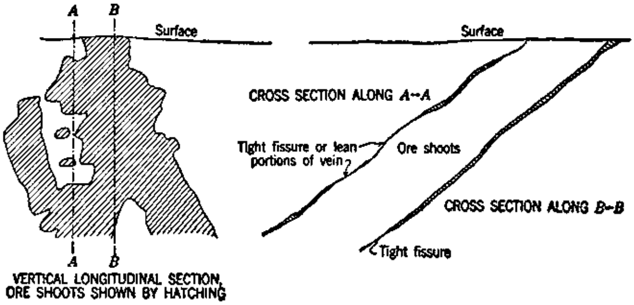


FIGURE 2.—Simple fissure veins, showing ore shoots, Grass Valley district, California

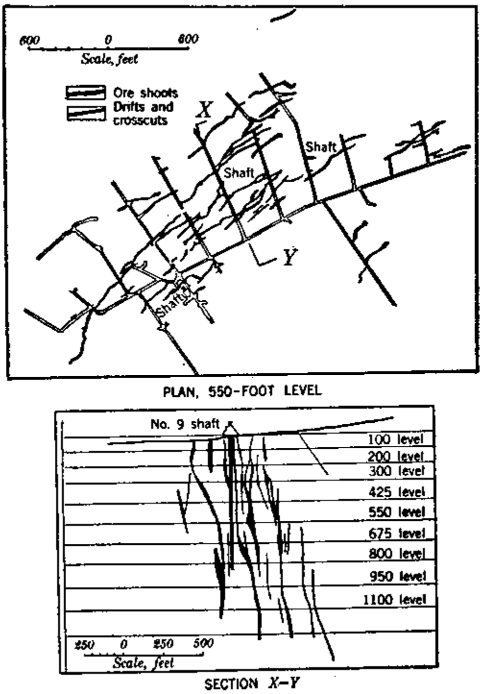


FIGURE 3.—Vein system, Hollinger mine (After A. G. Burrows, Ontario Department of Mines, vol. 33, part 2, 1924)

part 2, 1924)

irregular isolated lenses of quartz, with intervening bands of barren rock. This type is exemplified by the deposits at Juneau, Alaska. (Fig. 5.)

[3] McLaughlin, B. F., The Homestake Enterprise, Ore Genesis and Structure: Eng. and Min. Jour., vol. 132, Oct. 12, 1931, pp. 324–329.

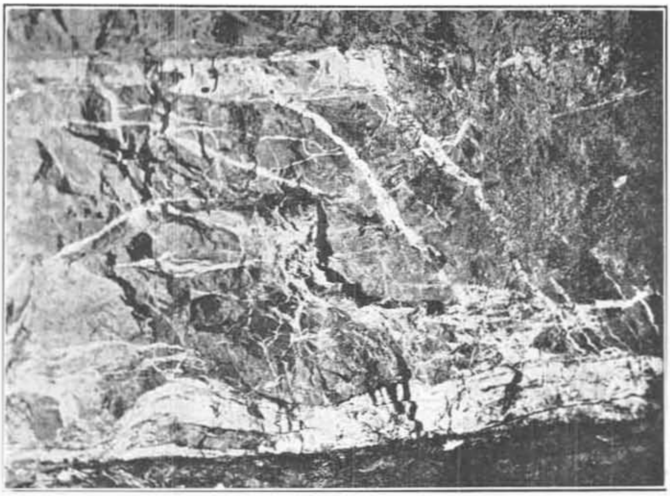


FIGURE 4.—Stope face showing quartz stringers in vein at Grass Valley, Calif. (After W. D. Johnston, jr.)

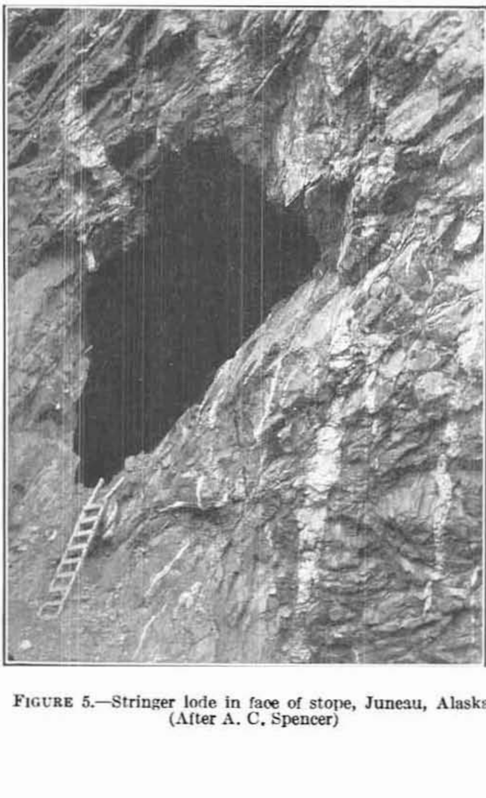


FIGURE 5.—Stringer lode in face of stope, Juneau, Alaska. (After A. C. Spencer)

The contact-metamorphic type is also represented among the gold-lode deposits of North America.[4] (Fig. 7.)

By far the greatest number of gold lodes mined in North America are of the tabular type. Such gold lodes are of different subtypes;

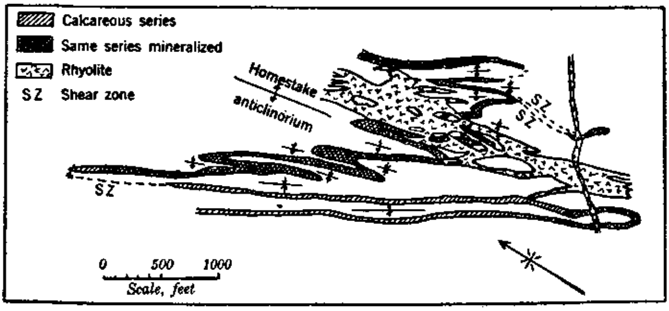


FIGURE 6.—Geology and part of 200 level, Homestake mine. (After J. O. Husted and L. B. Wright, Eng. and Min. Jour., May 12, 1923)

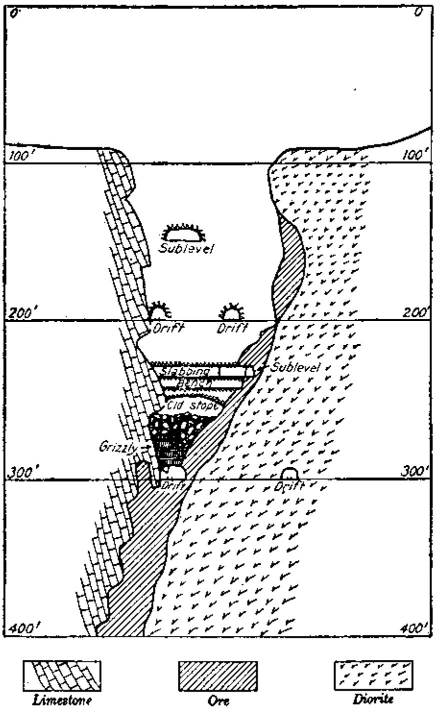


FIGURE 7.—Contact metamorphic type of ore body, Spring Hill, Mont.

thus, the ores may occur as simple fissure fillings, often banded, or they may occur in fault zones filling fissures and small cracks in the

[4] Pierce, A. L., Mining Methods and Costs at the Spring Hill Mine, Montana Mines Corporation, Helena, Mont.: Inf. Circ. 6402, Bureau of Mines, 1930, p. 2. Camsell, Charles, The Geology and Ore Deposits of Hedley Mining District, British Columbia; Canada Geol. Survey Mem. 2, 1910, 218 pp.

country rock and partly replacing it. Another subtype occurs in the form of lenticular lodes in schisted and sheared rocks, the gold being usually present in quartz stringers (often with sulphides), filling fissures and numerous small irregular cracks and seams in the sheared country rock and sometimes partly replacing it. Individual lenses are often of considerable lateral extent and are sometimes wide. More often they are narrow and occur in echelon, there being a series of lenses overlapping one another in plan but separated by barren rock. They generally parallel the schistosity roughly in strike and dip, but some trend at small angles thereto. This type merges into the broad shear-zone type represented by the Juneau deposits, where individual lenses are too small and irregular, both in form and distribution, to be worked individually by selective mining.

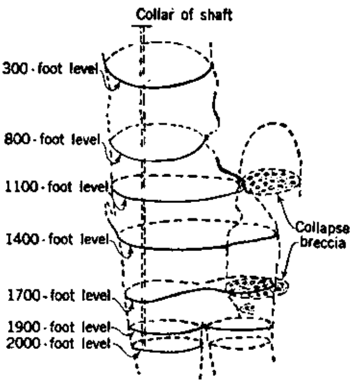


FIGURE 8.—Outline of Cresson pipe, Cripple Creek, Colo. (After G. F. Loughlin)

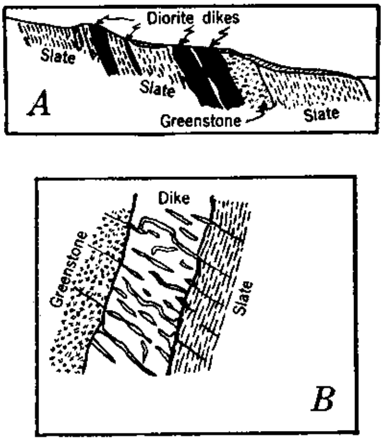


FIGURE 9.—Type of gold lodes, Douglas Island, Alaska. (After A. C. Spencer.) *A*, Cross section through Alaska-Treadwell mine; *B*, diorite dike, showing veinlets of quartz and calcite

Gold lodes in or around pipes or chimneys also have been worked. (Fig. 8.)

The deposits of Douglas Island, Alaska, which occur in albite-diorite dikes represent another type of deposit. (Fig. 9.)

Other types of lode-gold deposits occur in the United States and Canada but are of scientific rather than economic interest.

GEOLOGIC AGE OF LODE-GOLD DEPOSITS

Gold is found in North America in rocks of all ages ranging from Keewatin to Tertiary, but the commercially important deposits are confined to two principal groups. The first group embraces those deposits formed in ancient pre-Cambrian rocks during pre-Cambrian times and the second group, deposits formed in Mesozoic or later time, largely in rocks of Mesozoic and earlier ages. The second group may be divided into two subgroups—those formed in Cretaceous time and those formed in late Cretaceous or early Tertiary time, commonly in the igneous rocks (with some exceptions) of the same period.

From a purely practical rather than scientific standpoint the significance of these occurrences is that the formation of valuable auriferous lode deposits (as distinguished from placers) occurred in times of intense igneous activity, so that the search for lode gold should be confined to areas where igneous rocks are prominent, although they need not necessarily be exposed on the surface. In such areas the fringes, domes, and pendants of the igneous mass are the most favorable locations. Also, the porphyritic types of rocks have generally been more productive than the granites.

The known important deposits of the pre-Cambrian occur in regions of intense dynamic metamorphism as evidenced by folding, shearing, and often by recrystallization of the rocks. The Cretaceous deposits are characteristic of the western Cordillera extending from California up through Washington and Oregon, in parts of which they are buried by later volcanic rocks, and into Alaska. The Tertiary type predominates in the central and eastern Cordillera in Nevada, Washington, Colorado, and parts of Idaho and Montana.

MINERALOGY OF LODE-GOLD DEPOSITS

Gold in lode deposits nearly always occurs in the native state. It may, in some cases, be unaccompanied by sulphide minerals, as at Oatman, Ariz.,[5] and Seven Troughs, Nev.,[6] but generally the gold is accompanied by pyrite, with which it is partly or wholly associated in intimate admixture. In some deposits chalcopyrite is the most important carrier of the gold; and in others arsenopyrite, galena, or even sphalerite, may be important.

Gold, as it is found in lodes, is usually alloyed with silver in proportions which range from nearly pure gold to nearly pure silver. When silver is present in excess of 16 per cent by weight the pale natural alloy electrum is formed. Silver is more abundant in the Tertiary deposits, which were formed at relatively shallow depths, than in pre-Cambrian or Mesozoic veins. The ores at Tonopah were chiefly valued for their content of silver; and the deposits at Republic, Wash., likewise of Tertiary age, carried 3 to 8 parts of silver to 1 of gold. The Mesozoic veins of California yield gold with a fineness usually between 850 and 900, and the pre-Cambrian gold of Ontario is of similar quality. Gold purer than 900 fine is rarely found in nature. Small amounts of platinum, palladium, iridosmine, copper, iron, and bismuth sometimes occur alloyed with gold.

Gold is one of the most inert of known chemical elements and in nature enters into very few compounds; compounds with tellurium are the only stable natural ones in which gold is definitely known to occur, and these are rare. The tellurides—sylvanite, calaverite, krennerite, and nagyagite—are the best-known varieties. They formed the most important source of gold in a few camps, of which Cripple Creek is the classic example, and are known to occur, generally in insignificant amounts, in many gold deposits.

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[5] Ransome, F. L., Geology of the Oatman Gold District, Arizona; U. S. Geol. Survey Bull. 743, 1923, 58 pp.

[6] Ransome, F. L., Notes on Some Mining Districts in Humboldt County, Nev.: U. S. Geol. Survey Bull. 414, 1909, 75 pp.

Selenium has been reported from three districts in the United States;[7] and, while definite knowledge of the nature of its occurrence is lacking, it is believed that gold selenides may be present in the ores.

The mineral associations of gold are simple in older veins of pre-Cambrian and Mesozoic ages but are in many cases more complex in Tertiary deposits. Search of the published literature has provided the basis for Table 3, in which are listed most of the minerals known to occur in gold deposits; symbols are employed in the table to indicate the relative abundance of each species in the primary zone, in the zone of supergene enrichment, and in the zone of oxidation for each of the three general classes of deposits (pre-Cambrian, Mesozoic, and Tertiary).

The literature studied includes reports on 60 districts and covers several hundred mines. Publications of the geological surveys of the United States and Canada, the Bureau of Mines, and the technical and scientific press have been drawn upon. The data presented in these various reports naturally are not comparable in the degree of detail with which they discuss the mineralogy of the lodes they describe. All, however, mention the more important minerals, so that a fairly accurate measure of the degree of prominence of these commoner varieties is obtainable. Many of the reports make no mention of the rarer minerals, so that the table doubtless understates their prominence; however, it serves to indicate the extent to which they occur in a general way. The numbers given after each mineral in the table represent the number of districts out of the 60 studied from which each was reported.

NONMETALLIC GANGUE MINERALS

Of the 60 districts covered, quartz was reported from every one; in the great majority of gold lodes it is the most prominent gangue mineral present. It may occur as a simple fissure filling or replacing the shattered, brecciated country rock adjacent to the vein, or both. Banding and crustification are common structures.

Carbonate minerals are next to quartz in abundance. Of these calcite is the most prominent; but dolomite and rhodochrosite are common, and the mineral ankerite (an indefinite mixture of lime, magnesium, and iron carbonates) is abundant in certain veins, such as those of the Mother lode[8] and some of the Ontario deposits. In many instances values are poor where carbonates abound, as at Grass Valley, Calif.[9]

Sericite is widely distributed in gold quartz veins; it occurs as small flakes scattered through the quartz and as seams and partings and is abundant in gouge and altered wall rocks. Chlorite, talc, mariposite, and other micaceous minerals are often found in similar associations.

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[7] Bastin, E. S., and Laney, F. B., The Genesis of the Ores at Tonopah: U. S. Geol. Survey Prof. Paper 104, 1918, 50 pp. Bancroft, Howland, The Ore Deposits of Northeastern Washington: U. S. Geol. Survey Bull. 550, 1914, 215 pp. Umpleby, J. B., Geology and Ore Deposits of Lemhi County, Idaho: U. S. Geol. Survey Bull. 528, 1913, 182 pp.

[8] Knopf, Adolph, The Mother Lode System: U. S. Geol. Survey Prof. Paper 157, 1929, 88 pp.

[9] Knaebel, J. B., The Veins and Crossings of the Grass Valley District, California: Econ. Geol., vol. 26, No. 4, 1931, pp. 375–398.

Barite is a common gangue mineral, and adularia (vein orthoclase) is quite characteristic of many Tertiary veins, such as those of Jarbidge, Nev.[10]

Kaolinite is common in oxidized portions of veins and is a notable constituent of gouge and altered wall rocks in depth.

Fluorite is quite often present in the gangue.

Tourmaline, garnet, vesuvianite, and other high-temperature minerals are rather rare in gold deposits and are found only in veins of the deep-seated, high-temperature type or in those rare gold ore bodies formed by contact metamorphism.

GOSSAN MINERALS

The oxidized gangue minerals characteristic of gossans, or fractured portions of veins which are subjected to alteration by percolating surface waters, consist chiefly of limonite with minor amounts of manganese oxides, malachite, chrysocolla, cerussite, and other secondary minerals derived from base-metal sulphides. Quartz, being very resistant to change, is invariably present in some abundance.

METALLIC MINERALS

Pyrite is nearly always present in gold veins. In typical gold lodes it usually makes up 2 to 10 per cent of the vein filling in good ore, although there are exceptional instances, such as that represented by the Haile mine in South Carolina, where gold ore occurs in a gangue of dense, massive pyrite. In a few instances, such as at Oatman, Ariz., pyrite is practically absent from the ore, and in others it is subordinate in importance to chalcopyrite or arsenopyrite. The best ore usually contains pyrite in crushed or irregular patches and specks, and well-crystallized cubes of the mineral are regarded as an unfavorable sign. Like all generalizations, however, this one is not always true. Pyrite is a common constituent of altered wall rocks.

As noted above, chalcopyrite is sometimes an important associate of gold. Small amounts are generally present in typical siliceous veins, but in the so-called copper gold deposits, such as those of Rouyn, Quebec, it is an important source of gold.

Arsenopyrite is regarded as a favorable indication in the California mines, where it sometimes occurs in erratic patches and shoots. In such instances, it often accompanies high-grade ore. It is the chief metallic mineral at a few mines, of which the Nickel Plate mine in British Columbia is the classic example. This is one of the few contact metamorphic deposits that have been large producers; the gold is intimately associated with arsenopyrite. Arsenopyrite is of common occurrence in the veins of the Alleghany district, California.

Galena is perhaps more widespread than chalcopyrite but in typical gold deposits is found in subordinate quantities; it often indicates good ore. Galena is commonly associated with the gold at Grass Valley, Calif., and at Juneau, Alaska.

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[10] Schrader, F. C., The Jarbidge Mining District, Nevada: U. S. Geol. Survey Bull. 741 1923, 86 pp.

Sphalerite is common in minor amounts. In gold veins it usually is confined to ore shoots.

Gold and silver have been discussed on pages 17 to 18. In the oxidized zone gold, if present, is found as small needles, scales, foils, or nuggets in the quartz-limonite gangue. In the primary zone its abundance is usually, but not always, proportionate to that of its close associates, the sulphides. In high-grade shoots gold is often intergrown with cracked and crushed quartz, which has no sulphides whatever. Silver is generally alloyed with gold, but in some Tertiary veins or in the zones of oxidation and enrichment in all classes of deposits it may be locally developed as the native metal, as horn silver, or in the form of the rich silver sulphides or sulpho-salts, argentite, polybasite, proustite, pyrargyrite, stephanite, and pearceite.

As has been mentioned, tellurides are of rare occurrence, although locally they are of great importance in a few districts.

The prominence of the rarer minerals is indicated by the figures in Table 3.

TABLE 3.—*Prominence of minerals in various occurrences of gold deposits*

Minerals [1]	Type of ore occurrence [2]								
	Pre-Cambrian deposits			Mesozoic deposits			Tertiary deposits		
	Zone of oxidation	Zone of supergene enrichment	Primary zone	Zone of oxidation	Zone of supergene enrichment	Primary zone	Zone of oxidation	Zone of supergene enrichment	Primary zone
1. Metallic minerals									
Altaite			[3]						
Anglesite (1)	R			O			O		
Argentite (6)		O	R		O	R		C	C
Arsenopyrite (17)		C	C		C	C		O	O
Azurite (2)	O			O			O		
Bismite (2)							O		
Bismuthinite (4)									O
Bornite (5)								O	O
Bournonite (1)									O
Braunite (1)							O		
Calaverite (5)		O	O		O	O		[4]	[4]
Cerargyrite (5)		R			R		O	O	
Cerussite (4)	R			O			O		
Chalcanthite (3)							R		
Chalcocite (7)					R			O	R
Chalcopyrite (35)		C	C		C	C		C	C
Chrysocolla (3)				R			R		
Cinnabar (1)									R
Copper (native) (3)				R	R		R	R	
Cosalite (1)									R
Covellite (1)									R
Cuprodescloizite (1)							R		
Cuprite (1)	R			R			R		
Emmonsite (2)							R		
Enargite (3)								R	
Freibergite (1)								R	
Galena (37)		C	C		C	C		C	C
Gold (60)	C	C	C	C	C	C	C	C	C
Goslarite (1)							R		
Hematite (7)	O			O			O		
Hinsdalite (1)							R		

[1] Numerals after name of each mineral represent number of times reported out of 60 reports studied.
[2] P, Very prominent or widespread in occurrence; C, common; O, occasional or sporadic; R, rare; L, locally (thus, CL means common locally).
[3] Common, 1 district.
[4] Occasional to common locally.

TABLE 3.—*Prominence of minerals in various occurrences of gold deposits.*—Con.

Minerals	Type of ore occurrence								
	Pre-Cambrian deposits			Mesozoic deposits			Tertiary deposits		
	Zone of oxidation	Zone of supergene enrichment	Primary zone	Zone of oxidation	Zone of supergene enrichment	Primary zone	Zone of oxidation	Zone of supergene enrichment	Primary zone
1. Metallic minerals									
Hübnerite (1)									R
Jamesonite (1)									R
Krennerite (4)		R	R		R	R		[5]	[5]
Limonite (23)	C			P			P		
Leadhillite (1)							R		
Magnetite (6)		O	O		O	O		R	R
Malachite (9)	O			O			O		
Marcasite (3)			R			R			R
Melanterite (2)							R		
Molybdenite (6)		O	O		O	O		O	O
Molybdite (1)	R			R			R		
Nagyagite (2)									R
Naumaunite (1)									R
Pearceite (3)								R	R
Petzite (4)		R	R		R	R		R	R
Pitchblende (1)							R		
Polybasite (6)								O	O
Proustite (5)								O	O
Psilomelane (8)	O			O			O		
Pyrargyrite (3)								O	O
Pyrite (53)		P	P		P	P		P	P
Pyrolusite (5)	O			O			O		
Pyromorphite (2)							R		
Pyrrhotite (11)		O	O		O	O		O	O
Realgar (1)								R	R
Rhodochrosite (10)		O	O		O	O		O	O
Scorodite (2)							R		
Scheelite (1)					R	R		R	R
Selenides (of Au and Ag) (3)								R	R
Silver (native) (55)	O	C	C	O	C	C	O	C	C
Smithsonite (2)	R			R			R		
Sphalerite (31)		C	C		C	C		C	C
Stephanite (8)		R	R		R	R		O	O
Stibnite (6)		R	R		R	R		R	R
Sylvanite (5)		[5]	[5]		[5]	[5]		[5]	[5]
Tennantite (6)					O			O	O
Tellurite (1)							R		
Tetrahedrite (13)		R	R	O	O			O	O
Wad (8)	O			O			O		
Wolframite (1)						R			
Wulfenite (1)							R		
Yukonite (1)				R					
2. Nonmetallic minerals									
Actinolite (1)								R	R
Adularia (8)		R	R		R	R		C	C
Albite (14)		R	R		[4]	[4]			
Allanite (1)					R	R			
Alunite (3)	R	R	R	R	R	R	O	O	O
Amphibole (1)						R			
Ankerite (6)		O	O		O	O			
Antigorite (1)				R	R	R	R	R	R
Apatite (4)	R	R	R	R	R	R			
Barite (16)	O	C	C	O	C	C	O	C	C
Biotite (3)		R	R		R	R			
Calcite (34)	C	P	P	C	P	P	C	P	P
Celestite (1)								R	R
Chalcedony (5)	O	O	R	O	O	R	O	O	O
Chlorite (11)	C	C	C	C	C	C	C	C	C
Chrysoprase (1)							R		
Cummingtonite (1)		R	R						
Diopside (1)		R	R						
Dolomite (8)		O	O		O	O		O	O
Epidote (3)		R	R		R	R			
Epsomite (1)							R		
Fluorite (13)	O	O	O	O	O	O	O	O	

[5] Rare to occasionally locally.

TABLE 3.—*Prominence of minerals in various occurrences of gold deposits*—Con.

Minerals	Type of ore occurrence								
	Pre-Cambrian deposits			Mesozoic deposits			Tertiary deposits		
	Zone of oxidation	Zone of supergene enrichment	Primary zone	Zone of oxidation	Zone of supergene enrichment	Primary zone	Zone of oxidation	Zone of supergene enrichment	Primary zone
2. Nonmetallic minerals									
Garnet (4)	R	R	R	R	R	R			
Gypsum (5)	R	R	R	R	R	R	R	R	R
Halloysite (1)								R	R
Hyalite (1)							R	R	R
Jarosite (1)							R		
Jasperoid (2)								R	R
Kalinite (1)							R	R	
Kaolinite (12)	C	C	C	C	C	C	C	C	C
Laumontite (1)							R	R	
Mariposite (6)	R	R	R	O	O	O			
Muscovite (2)	R	R	R	R	R	R			
Opal (3)	R	R		R	R		R	R	R
Orthoclase (4)								R	R
Pectolite (1)								R	R
Prehnite (1)				R	R	R			
Quartz (60)	P	P	P	P	P	P	P	P	P
Roscoelite (2)								R	R
Rutile (2)	R	R	R	R	R	R			
Sericite (20)	C	C	C	C	C	C	C	C	C
Serpentine (5)	O	O	O	O	O	O	O	O	O
Siderite (9)	O	O	O	O	O	O	O	O	O
Sulphur (1)							R	R	
Talc (4)	O	O	O	O	O	O	O	O	O
Thuringite (1)								R	R
Tourmaline (4)		R	R		R	R		R	R
Vesuvianite (1)		R	R		R	R		R	R
Wavellite (1)								R	R
Wollastonite (1)		R	R		R	R		R	R

CHANGES IN GOLD DEPOSITS WITH DEPTH

CHANGES NEAR SURFACE

Lode deposits which outcrop at the surface commonly exhibit different physical and mineral characteristics at and near the surface from those found at depth, except where recent glaciation or other erosional force has planed off the surface and exposed fresh underlying rock. The depth to which surface conditions prevail differs widely in different districts and may range from only a few to several or many hundreds of feet.

Mechanical weathering agencies affect the deposits to only relatively shallow depths and serve to rupture and erode the rocks through expansion and contraction, due to temperature changes and the action of ice, running water, etc. The influence of chemical agencies is much more profound, especially where the level of ground water is relatively deep, as in arid or semiarid districts, and commonly extends to and a short distance below the ground-water level or water table. The principal effect of these agencies is to oxidize the sulphide minerals present and to dissolve and carry off or carry down to lower depths the more soluble constituents of the lode. Gold is little affected by natural chemical agencies and is usually left behind in its approximate original position with the other less soluble materials of the lode. Emmons [11] has pointed out, however, that in the presence of chlorides and the higher oxides of manganese gold may be dissolved and carried in ground waters to lower levels, where it is redeposited as native gold by the chemical action of ferrous sulphate, organic compounds, etc. Under these circumstances, a zone of gold enrichment might be expected at or near the ground-water level. It may be noted also that deposits where gold has been dissolved in the oxidized zone seldom give rise to important placer deposits. Likewise, if the gold particles are very fine they may be carried away and widely dispersed without the formation of placers.

Since most gold-lode deposits contain iron sulphides, which when oxidized are converted into red and yellow oxides or hydrated oxides of iron, the outcrops commonly have a rusty appearance and, due to removal of certain of the more soluble lode minerals, are usually honeycombed. Such an appearance does not signify that the lode contains gold, since the minerals responsible therefor are present in most lodes containing base metals and even in iron-rich country rocks.

Other sulphide minerals become altered in the oxidized zone to their oxide, carbonate, and sulphate derivatives, and changes take place in the gangue minerals. Quartz, being little affected, remains in place.

In lode deposits containing gold these rusty, stained, and honeycombed outcrops, commonly called gossans, are often richer than the primary ore, due to the removal of part of the gangue material as described above. On the other hand, if chlorides, such as sometimes are found in waters in desert regions, and the higher oxides of manganese have been present at any time, gold may have been dissolved from the upper parts of the lode, leaving a gossan leaner than the primary ore. The presence of manganese oxides is evidenced by the familiar "desert bloom" known to the prospector—a black or dark brown coating on the rocks or seams or veinlets in the gossan. It thus follows that gold-lode deposits may differ considerably in the upper zone of oxidation and in the lower primary ore zone, in physical character, in chemical character, and in the amount of gold and other minerals present.

In northern districts where glaciation has occurred and where the water table stands near the surface the oxidized zone may be absent or very shallow. Thus, in the gold districts of northern Ontario and at Juneau, Alaska, the primary ores are practically at the surface and have only superficially oxidized outcrops.

On the other hand, primary ores may occur close to the surface in arid climates or regions which lie too far south to have been subjected to glacial erosion, provided that very steep topography and intermittent torrential rains have combined to remove the surface material as fast as weathering and chemical alteration of the vein material worked downward.

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[11] Emmons, W. H., The Agency of Manganese in the Superficial Alteration and Secondary Enrichment of Gold Deposits in the United States: Am. Inst. Min. and Met. Eng. Bull. 46, 1910, pp. 789–791.

Thus the depth to which the upper portions of gold deposits may be leached and oxidized depends on the relative influence of several natural agencies.

In the Granite-Bimetallic lode,[12] Philipsburg, Mont., the gold is fine, and considerable manganese carbonate is present in the primary ore. Upon oxidizing, this gives a notable amount of manganese oxide, which is present in nearly every outcrop and in the oxidized zone to considerable depths. No placers are associated with these deposits, and the outcrop carries less gold than the lode at a depth of 50 to 200 feet below surface. Oxidation extends to a depth of 800 feet in places.

At the Cable mine, however, the gold is for the most part less finely divided, and manganese oxide is entirely absent or extremely rare. These deposits yield rich placers and have been stoped to the surface. On the Comstock lode, Nevada, according to King,[13] "A zone of manganese oxide occupies the entire length of the lode from the outcrop 200 feet down." In this lode oxidation extends in some places to a depth of 500 feet. At Tonopah, Nev., oxidation extends to a depth of 700 feet in the Mizpah mine.

At Creede,[14] Colo., in the Amethyst vein, the largest secondary gold deposits lie 200 to 700 feet below the surface, and some oxidation occurs in this vein at 1,000 feet in depth. In the Appalachian region the oxidized zone usually extends 80 to 200 feet below the surface. In the Nevada City and Grass Valley districts, California,[15] the upper part of a vein is generally decomposed, forming a mass of limonite and quartz. The decomposition seldom extends more than 150 feet vertically below the surface. Spurr[16] states that at Silver Peak, Nev., no decided enrichment of the ores by oxidation can be established. At Douglas Island, Alaska,[17] nothing in the character of the ore indicates any considerable concentration by oxidizing waters.

In the Georgetown quadrangle, Colorado,[18] the auriferous deposits are mainly at Idaho Springs and in the Empire district. The lodes are usually oxidized at the surface and from 15 to 70 feet downward. In several mines the oxidized ore is much richer than the average ore. In the Summit district, Colorado,[19] the zone of incompletely oxidized ore extends to a depth ranging from a few feet to 300 feet. All the bonanzas were confined to this zone. At Bullfrog, Nev.,[20] the outcrops were comparatively poor, but good ore was encountered within a few feet of the surface, and some deposits were worked by open-

[12] Emmons, William H., Outcrops of Ore Bodies: Min. and Sci. Press, vol. 99, 1909, pp. 751–754, 782–787.

[13] King, Clarence, The Comstock Lode, in Hague, J. A., Mining Industry: U. S. Geol. Expl., 40th Par., vol. 3, 1870, pp. 11–96.

[14] Emmons, W. H., and Larsen, E. S., Geology and Ore Deposits of Creede, Colo.: U. S. Geol. Survey Bull. 718, 1923, 198 pp.

[15] Lindgren, Waldemar, The Gold-Quartz Veins of Nevada City and Grass Valley Districts, California: U. S. Geol. Survey 17th Ann. Rept., Pt. II, 1896, 863 pp.

[16] Spurr, J. E., Ore Deposits of the Silver Peak Quadrangle, Nevada: U. S. Geol. Survey Prof. Paper 55, 1906, 174 pp.

[17] Spencer, A. C., The Juneau Gold Belt, Alaska: U. S. Geol. Survey Bull. 287, 1906, 161 pp.

[18] Spurr, J. E., and Garrey, G. H., Economic Geology of the Georgetown Quadrangle, Colorado: U. S. Geol. Survey Prof. Paper 63, 1908, pp. 99–101.

[19] Hills, R. C., Ore Deposits of Summit District, Rio Grande County, Colo.: Proc. Colorado Sci. Soc., vol. 1, 1883, pp. 20–26.

[20] Ransome, F. L., Emmons, W. H., and Garrey, G. H., Geology and Ore Deposits of the Bullfrog District, Nevada: U. S. Geol. Survey Bull. 407, 1910, 130 pp.

cut. Some of the ore deposits decrease in value below the 400-foot level.

At Cripple Creek, Colo.,[21] the oxidation extended downward to a depth generally less than 200 feet, and even in the oxidized zone residual sulphides are present. In the Ontario gold districts there is very little surface oxidation, and if there ever was an oxidized zone, practically all of it has been planed off by glacial erosion.

CHANGES BELOW ZONE OF OXIDATION

Aside from the changes due to weathering and oxidation in the upper parts of the lodes, variations in size, dip, and character of wall rocks often occur as depth is reached on a given lode, and the mineral associations and value of the ores may change. In some instances two or more veins in the upper levels come together at greater depth to form a single vein, which may be richer or of leaner grade than the separate veins; often there is an enrichment at or near the junction, although this does not necessarily follow. Lodes commonly pinch and swell both on the dip and along the strike, and the richer sections or ore shoots may be separated by barren or lean ground. Sometimes certain wall rocks have been more favorable to replacement by mineralizing solutions than others, and as the lode passes from one formation to another the width and grade of ore may change, or the ore may spread out along a favorable contact. Likewise, some rocks are more susceptible to shearing or shattering than others, and in these the ore bodies are often richer and wider. This feature, indeed, is of the very greatest importance in controlling the movement of ore-bearing solutions and the deposition of ore. In most if not all of the important lode-gold districts in North America the workable deposits are closely related and confined to zones of intense dynamic action where folding or faulting, fissuring, and shearing have taken place, thus providing channels for the circulation of mineralizing solutions and openings in which deposition could take place.

Thus, the rich gold ores of Kirkland Lake, Ontario, are found in a broad faulted zone where the rocks in and adjacent to the plane of the main faults have been sheared and shattered. In the Porcupine district, Ontario, the ore occurs in lenses in rocks which have been intensely sheared in a general region of intense folding. At Lead, S. Dak., where the rocks have been folded and sheared, the great Homestake ore body occurs around the nose of a plunging anticlinal fold with lesser ore bodies around minor folds. At Juneau, Alaska, the ores are found in sheared rocks in the form of stringer lodes and lenses.

As a lode passes from a tough or plastic rock formation to a more brittle one conditions become more favorable for deposition of ore, and these changes may occur vertically, along the strike, or along diagonally plunging zones.

Some well-known gold deposits in the United States and Canada have proved to be remarkably persistent with depth and with little if any change in average tenor.

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[21] Lindgren, Waldemar, and Ransome, F. L., Geology and Gold Deposits of the Cripple Creek District, Colorado: U. S. Geol. Survey Prof. Paper 54, 1906, p. 129.

HOMESTAKE MINE, SOUTH DAKOTA

The Homestake ore body has persisted from the outcrop on surface to a vertical depth of over 2,600 feet (or about 3,500 feet down the plunge of the anticline) with no diminution in grade of the ore. The ore body is somewhat narrower on the 2,150 and 2,300 levels. On the 2,600-level the ore body again widens.

KIRKLAND LAKE DISTRICT, ONTARIO, AND MOTHER LODE, CALIFORNIA

In the Kirkland Lake district there is no sign of diminution of value at 4,700 feet; and in some of the lower levels values are if anything somewhat higher on the average than in the upper levels, though lean zones have occurred at several elevations, with good ore coming in again below. On the Mother lode, California, at a depth of over a mile on the dip and some 4,900 feet vertically, the values in two or three mines are reported to be as good as the average of the upper levels. Here, too, there have been lean horizons with good ore below.

ALASKA-JUNEAU MINE, ALASKA, AND GRASS VALLEY DISTRICT, CALIFORNIA

At the Alaska-Juneau property average values 600 feet below the active producing section of the mine are, at this writing, reported to be averaging several times those of the producing section. At Grass Valley, Calif., development has proceeded at a depth of 9,000 feet on the dip, or about 4,500 feet vertically, in the North Star mine.

In the instances cited above the deposits are of pre-Cambrian or Mesozoic age, and it may be stated that in North America the most persistent known gold deposits belong in a general way to these eras. It must not be inferred that all lodes of these periods are persistent or that the discovery of a lode belonging thereto will be followed by successful exploitation. There are many examples to the contrary. Another characteristic of the deposits of these groups is the comparative simplicity of the mineralization—chiefly quartz, pyrite (and galena in some cases), and only a minor amount of other sulphides, and some calcite and other carbonates with native gold.

The Tertiary deposits, as previously pointed out, seem to be more erratic, have generally a rather complex mineral association, and are less persistent with depth.

BODIE DISTRICT, CALIFORNIA

At Bodie, Calif., the grade of the ore apparently decreased rapidly below about 500 feet in vertical depth in many of the veins, and practically all the production came from above 1,000 feet in depth. This may have been due to the conditions at the time deposition of ore took place or, more probably, to secondary enrichment in the upper horizon.[22] The vein systems and faulting are very complex in this district. (Fig. 10.) One group of veins contains considerable black oxide of manganese, and in this group the gold values die out at about 500 feet in depth.

[22] Brown, R. Gilman, The Vein System of the Standard Mine, Bodie, Calif.: Trans. Am. Inst. Min. and Met. Eng., vol. 38, 1907, pp. 343–357.

GOLDFIELD DISTRICT, NEVADA

At Goldfield, Nev., the bonanza ores of the upper levels became perceptibly leaner at depths below 300 feet, although good ore was produced from below this zone to depths exceeding 800 feet in some of the mines. Ore has been mined from greater depths, but this exception does not disprove the generalization that the bulk of the rich ore in this famous district was derived from stopes within 1,000 feet of the surface.

TONOPAH DISTRICT, NEVADA

At Tonopah the rich ore is all oxidized in the upper levels, and in some zones of shattering the oxidation is complete at the 800 and partial at the 1,500 levels. The primary ore decreases in grade with depth, carrying in general more quartz and a smaller amount of sulphide. It is believed that passing of the veins from rhyolite into trachyte is at least partly responsible for the general decrease in tenor.[23]

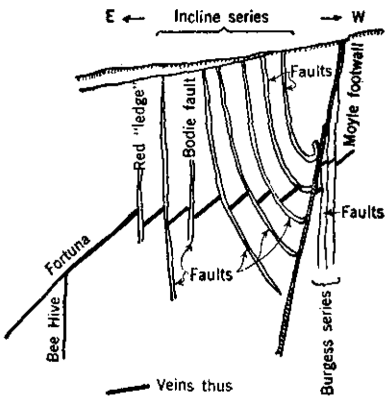

FIGURE 10.—Cross section of vein system of the Standard mine, Bodie, Calif. (After R. Gilman Brown, Trans. Am. Inst. Min. and Met. Eng., vol. 38, 1907)

ROUND MOUNTAIN DISTRICT, NEVADA, AND BRECKENRIDGE DISTRICT, COLORADO

At Round Mountain, Nev.[24] the principal vein dipped 15° and was productive for 900 feet down the dip, or less than 350 feet vertically. The profitable ore came chiefly from the zone of oxidation and enrichment. The small rich veins of Breckenridge,[25] which owed their value chiefly to the process of secondary enrichment, were found to be unproductive below 450 feet in depth. The oxidized ore was of high grade but very pockety and erratic in occurrence.

OATMAN DISTRICT, ARIZONA

At Oatman, Ariz.,[26] the Tertiary gold veins afford a rather unusual example, in that sulphides are practically absent. Little or

[23] Bastin, E. S., and Laney, F. B., work cited.
[24] Ransome, F. L., and Burchard, E. F., Contributions to Economic Geology: U. S. Geol. Survey Bull. 725, 1922, 440 pp.
[25] Ransome, F. L., Geology and Ore Deposits of the Breckenridge District, Colo.: U. S. Geol. Survey Prof. Paper 75, 1911, 187 pp.
[26] Ransome, F. L., Geology of the Oatman Gold District, Arizona: U. S. Geol. Survey Bull. 743, 1923, 86 pp.

no downward enrichment has taken place, and most of the many veins in the district have proved barren or lean, or, at best, pockety. A few, however, of which the Tom Reed, Big Jim, Gold Road, and United Eastern are the principal ones, have been highly productive. All these veins occupy fault fissures and have themselves been displaced by postmineral movement. Very little good ore outcropped at the surface, and little has been found below 1,000 feet. However, at present (1931) a small but high-grade ore body is being mined at 1,200-foot depth. The greatest single ore body developed in the United Eastern mine was found in a fork of the Tom Reed vein and had maximum dimensions of 1,000 feet in length, 50 feet in width, and 800 feet in depth, and averaged about $22 a ton in value. (Fig. 11.)

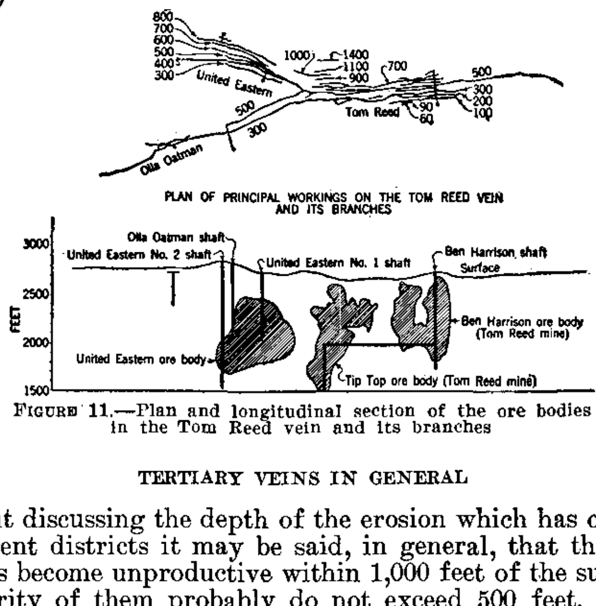

FIGURE 11.—Plan and longitudinal section of the ore bodies in the Tom Reed vein and its branches

TERTIARY VEINS IN GENERAL

Without discussing the depth of the erosion which has occurred in the different districts it may be said, in general, that the Tertiary gold veins become unproductive within 1,000 feet of the surface, and the majority of them probably do not exceed 500 feet. However, there are several notable exceptions, such as Cripple Creek, Colo., where good ore has been mined at a depth of 2,900 feet; the Comstock lode, Nevada; Telluride and Camp Bird, Colo.; and, to a less striking extent, Goldfield and Tonopah, Nev.

Secondary enrichment of the upper portions is in many cases responsible for the existence of workable ore bodies, but by no means universally so in these veins. In a recent summary of Nevada mining districts, Ferguson [27] expresses the belief that little chance now exists for increased production from known or new veins of the pre-Tertiary system in western Nevada, but that there is more likelihood of new discoveries in the Tertiary lavas than in the older deposits. He further states that, of the Tertiary types, the silver and silver-gold veins of pre-Esmeralda age are more likely to persist than the Pliocene gold veins; workable ore in Tertiary lava veins usually persists below the zone of supergene enrichment.

[27] Ferguson, Henry G., The Mining Districts of Nevada: Econ. Geol., vol. 24, No. 2, 1929, p. 115.

A rather exhaustive search of the geologic literature of American gold deposits leads the writers to the belief that only meager possibilities exist for the extension of Tertiary gold deposits to depths below 2,000 feet, but that a thorough reprospecting of old districts in the light of present knowledge will produce more or less new ore—in particular, ore that apexes underground and occurs in blind lodes parallel to known veins. The facts that better transportation facilities now exist in many remote regions and that mining and milling technique have made rapid strides during the last 15 years may also render profitable old mines which were shut down when costs mounted above mill returns and which are now full of water.

Later flows of lava are known to cover in part some lodes, as at Tonopah, and this fact lends credence to the possible existence of important deposits that have lain undiscovered because their lava capping has remained sufficiently intact to conceal the veins.

RELATION OF OUTCROPS TO ORE SHOOTS

There seems to be a popular impression among prospectors that if values are found at or near the surface better values should be found at depth, but the reverse has more often been the case. As previously pointed out, outcrops of some gold lodes may have lost part of their constituents and thus become enriched by the increased relative amount of gold in the upper portions of the lode. This is most apt to be the case where the sulphide content is high and there is a deep zone of oxidation.

On the other hand, there are important occurrences of gold lodes where this generalization does not apply. In the gold districts of Ontario and Quebec many of the best ore shoots do not outcrop; and some are small and tight at and near the surface, becoming better at greater depths. This condition has been found in enough instances for mining men in these districts to concede that, in general, exploration to considerable depths is often warranted on rather meager surface showings. Although expenditures for deep work on the strength of such showings probably will result in many more disappointments than successes, a hope, at least, is often justified that profitable ore may be found. So far as the authors of the present paper are aware, no adequate explanation for these conditions has yet been offered.

EMMONS'S THEORY

The theory regarding changes in depths as indicated by outcrops advanced by Emmons[28] might possibly be applied as a partial explanation, at least. Without attempting a discussion of erosion as related to lode outcrops, which is a large subject and not within the scope of the present paper, Emmons's postulation is stated as follows:

> If the lode is very resistant and the country rock is easily eroded, then the lode outcrops above the surface, and the wider part of the lode will outcrop for a longer period of time than the narrower part; hence, if there be several deposits of this character, most of them will be found at a time when a maximum amount of the hard rock is exposed to erosion. (Fig. 12, *A*.) If, on the

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[28] Emmons, William H., Outcrops of Ore Bodies: Min. and Sci. Press, vol. 99, 1909, pp. 751–754, 784–787.

other hand, the deposit be less resistant than the country rock, and if it vary in width down the dip, the narrow portion is likely to remain at the surface longer, as shown by figure 12, *B*, where the solid line represents the "permanent" outcrop, and the dotted line the "temporary" outcrop. Such a deposit is likely to increase in size as it is followed downward. In other words, erosion is such that a maximum amount of the most resistant material, be it ore or country rock, tends to remain longest at the surface, and as far as possible to monopolize the outcrop. Not all, but a majority of such deposits will increase in size with depth.

Applying this to the gold deposits of Ontario and Quebec, and substituting the words "shear zone" for lode or deposit, a possible explanation may be offered for the occurrences of some of the non-outcropping lodes.

As previously mentioned, the important gold deposits in this region are all closely related and confined to broad zones of intense shearing and shattering of the rocks. In many instances there are indications

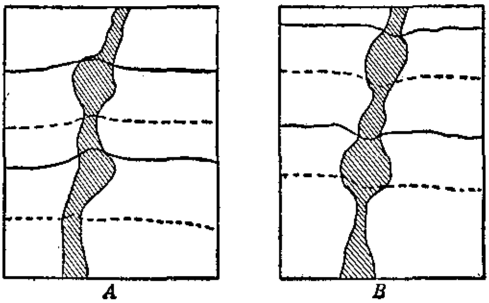

FIGURE 12.—Cross sections showing erosion of lode and wall rocks. (After W. H. Emmons.) *A*, Lode more resistant than country rock; *B*, lode less resistant than country rock

of considerable postmineral movement and shattering, and even though the ore deposits themselves are often highly silicified the net result has probably been to leave the shear zones generally less resistant to erosion than the surrounding rocks. Thus, according to Emmons's hypothesis, the shear zones (in distinction from the ore bodies contained within portions of the shear zones) will, as a result of erosive action, tend to be eroded more rapidly than the surrounding rocks, and more of the latter than of the former will remain exposed at any given time. Carrying this a step further, the portions of the shear zones which have suffered the most intense shattering (and which are most favorable to ore deposition) will be less permanent than the more resistant portions in their exposures.

OATMAN, ARIZ.

At Oatman, Ariz.,[29] many veins outcrop boldly and form conspicuous surface features, but most of these have not been highly productive and many of them were barren. On the other hand, the large, important ore bodies were nearly always found beneath relatively inconspicuous surface showings which carried small or negligible amounts of gold.

[29] Ransome, F. L., Geology of the Oatman Gold District, Arizona: U. S. Geol. Survey Bull. 743, 1923, 86 pp.

DEPTH OF FISSURES

The influence of dynamic action in producing shattering, shearing, and fissuring of rock masses tends to decrease with depth, so that ultimately ore-bearing lodes may be assumed to die out at some point in depth. This depth may, in some instances, be so great as to be well below the limits of practical mine operation. In other instances, however, the dying out or diminution in size or number of fissures is observable at mining depth.

CRIPPLE CREEK, COLO.

At Cripple Creek, Colo.,[30] while telluride ore has been mined at a depth of 3,000 feet below the present surface, or about 5,000 feet below the original surface of the Cripple Creek volcano, the veins which were numerous at higher horizons have converged at depth and are few in number, and the ore shoots are smaller.

APPALACHIAN REGION

In the Appalachian region, particularly in Alabama, Georgia, the Carolinas, and Virginia, gold deposits of the deep-seated type are widely scattered through areas of pre-Cambrian schists and granite. They occur as fissure veins in granite or as lenticular bodies in the old schists. Quartz, pyrite, and often garnet form the principal gangue constituents of the rather simple mineralization, with gold both free and intimately mixed with pyrite. The oxidized and enriched upper portions of these lodes have been profitably worked in the past to depths of 80 to 200 feet or so, but the primary ore below the level of ground water generally has been low grade. In some places the mineralization has been erratic, but elsewhere the distribution and width have been uniform. This fact, coupled with poor response of the ore to treatment by metallurgical processes of the last century, has precluded deep exploration of the deposits. Considerable placer gold has been won from the Appalachian region in former years, as well as a fair amount from the free-milling ores of the oxidized zones. Since 1800, a total output of more than $50,000,000 has been mined, of which $30,000,000 is credited to placers. It should be noted, however, that much of the gold was derived by placer methods from disintegrated weathered material in place (residual rather than transported placers). This alluvial and residual gold probably contributed to the recorded production of both lode and placer gold, the method of working more than the nature of the occurrence being responsible for the classification employed.

In recent years Appalachian gold deposits have become insignificant as factors in total production. The great thickness of material known to have been eroded from the mountains since these ores were deposited is considered by many geologists to be a strong reason for concluding that the veins now exposed represent the roots or lower portions of formerly extensive lodes. It should be borne in mind, however, that many of these veins were probably formed at profound depths, estimated by Lindgren[31] at 3 or 4 miles, and that, further-

[30] Loughlin, G. F., Ore at Deep Levels in the Cripple Creek District, Colorado: Trans. Am. Inst. Min. and Met. Eng., vol. 75, 1927, pp. 42–73.

[31] Graton, L. C., Reconnaissance of Some Gold and Tin Deposits of the Southern Appalachians, with Notes on the Dahlonega Mines, by Waldemar Lindgren: U. S. Geol. Survey Bull. 293, 1906, 134 pp.

more, our present knowledge of them is largely confined to the shallow limits of oxidation.

Partly in consequence of the widespread interest in gold mining which has developed since 1929, a good deal of desultory prospecting and exploration by persons of limited resources has been recently taking place in the Appalachian region. The results have been negligible because very little real work has been done. It is significant, however, that one strong company, experienced in gold mining, has conducted many months of active exploration work in the Southeastern States without bringing to light anything of more than moderate promise. Many properties were examined, several were unwatered and reopened, and considerable underground work was done, according to information received informally by the authors. Many of the veins showed stoping widths of good ore (most of which had been taken out) above water level; but the ore shoots were short, averaging perhaps 25 to 40 feet along the strike. The old workings are seldom found to go below the zone of oxidation. Whether the early miners were forced to stop solely because of being unable to extract the gold from the sulphide ores, or whether the ore in the primary zone is markedly leaner than that above water level is a matter of some doubt. The company mentioned was planning to explore one rather promising vein in depth at the time of a visit to the district (September, 1931) to determine the nature of the unoxidized material. Should deep work reveal the existence of minable sulphide ore below the oxidized zone serious exploration of some of the other old mines might be justified. General knowledge of the grade, size, and persistence of the primary ore bodies will not be possible until considerable deep exploration has been done. That some small or moderate size mines may be developed is not improbable; but it seems doubtful, in the light of present knowledge, that great ore bodies lie undiscovered in the Appalachian region.

OCCURRENCE OF PRODUCTIVE AND NONPRODUCTIVE HORIZONS

The pinching of an ore shoot or a decrease in the grade of the ore at a given horizon does not necessarily mean that the bottom of the workable ore has been reached, as these may be local conditions due to one or more of the causes previously mentioned. Several examples illustrating this point are given below.

TECK-HUGHES MINE, ONTARIO

At the Teck-Hughes mine, Kirkland Lake, Ontario, discouragement was met at the ninth and tenth levels, where very little ore was found. (Fig. 13.) Below these levels, however, the ore opened up again and was wide and of high grade. Another poor zone has been encountered on the twenty-sixth, twenty-seventh, and twenty-eighth levels, while on the thirtieth level (3,600 feet below surface) the ore shoot is again large and of good grade.

MOTHER LODE, CALIFORNIA

On the Mother lode of California, in the vicinity of Jackson, there was a lean zone with no ore on the 2,300 level, but the ore was encountered again on the 2,750 level and extended almost up to the 2,300 level. The largest ore shoot was at or near the 3,900 level, and good ore is still being mined at a vertical depth of 4,900 feet.

SIERRA COUNTY, CALIF.

At a mine in Sierra County, Calif., there was a barren zone at the 1,000 level extending almost up to the 800 level. A short distance below the 1,000 level ore was again found, and to-day the mine is being profitably worked on and above the 2,100 level, with development going on at much deeper levels.

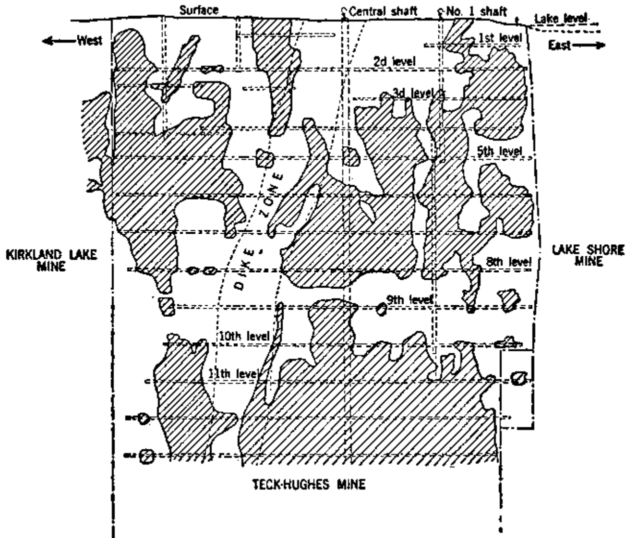

FIGURE 13.—Ore shoots in upper part of Teck-Hughes mine showing poor zone between eighth and tenth levels. (After E. W. Todd)

PORTLAND MINE, COLORADO

At the Portland mine in the Cripple Creek (Colo.) district, the upper vein system pinched out around the fifth level, below which another vein system was encountered. Another unproductive horizon occurred below the 1,200 level, but on the 1,500 good ore shoots were again found. Another vein system was found extending from the 2,300 to the 3,000 level, the other vein system having pinched out.

It is probable that had the barren or lean zones mentioned in the foregoing examples occurred at or close to the surface, these mines would not have been discovered and developed. Likewise, it is highly probable that there are valuable gold deposits in the United States and Canada which have not yet been discovered, due to the fact that the valuable horizons do not outcrop or come close to the surface. The search for such deposits is apt to prove expensive. It is to be hoped, however, that as a better knowledge of ore deposits, their habits, and structural relationships is gained, and as improved methods of exploration, aided perhaps by advances in the science of geophysical prospecting, are developed, such deposits will eventually be discovered.

Part 2.—EXPLORATION, DEVELOPMENT, AND MINING

METHODS OF PROSPECTING, EXPLORATION, AND SAMPLING

This paper is concerned principally with the exploitation of lode deposits; gold placer mining will be discussed comprehensively in a bulletin now in course of preparation by a specialist in this type of mining. Therefore, only brief mention will be made here of placer mining and the prospecting, exploration, and sampling of placer deposits. The occurrence and types of placer deposits have been briefly touched upon in a preceding chapter.

PROSPECTING

GOLD PLACERS

After placer gold is discovered along the bed of a stream, river bar, bench, terrace, or beach, as the case may be, the method of prospecting will depend upon not only the type, depth, and water conditions (whether under water or in water-saturated ground or not) of the deposit, but upon such questions as location and accessibility for supplies and fuel, relative costs of different prospecting methods, amount of money available, and probable method of working.

If, as is usually the case, the original discovery is made by a prospector of small means, the discovery will be further investigated by simple panning or by washing the sand and gravel in a rocker (fig. 14, A) or "tom" (fig. 14, B), picking or screening out the coarse gravel, washing away the light sands and soil, and concentrating the gold and heavy sands in the pan or behind riffles or collecting it on blankets, the gold finally being recovered by picking out the "colors" or by amalgamation. The prospector will usually excavate the material by hand. Under such circumstances the prospecting and working of the placer are coincident. If the gold is very fine or the values low, coarser gold and higher average values may be reasonably expected farther upstream, if the placer is recent. (If it is ancient the present drainage may differ from that at the time the placer was formed and may even be in the opposite direction.) At any rate, the coarsest gold and richest ground may be expected nearest the original source of the gold; hence the prospector will usually work in that direction from the original discovery, if it is possible to do so. Also, bearing in mind that the richest concentration usually occurs at bedrock and in the low spots thereof, care should be taken to carry all excavations to bedrock and to follow erosion courses therein. In some wide bar deposits, however, the gold occurs at several horizons on beds of hardpan or quite uniformly distributed throughout the deposit.

If large-scale operations, such as hydraulicking, dredging, and steam-shovel or drag-line work, involving a large capital outlay, are contemplated, the original discovery and preliminary crude exploration should be followed by a systematic exploration program before the mining plant and equipment are installed. In this work it is desired to investigate the area, depth, total volume, and average value of the ground, the location of the pay dirt, the amount and character of the overburden, and the character of the bedrock.

The common methods of prospecting are by trenching or test pitting and by drive pipe or drilling. Each has its particular field of

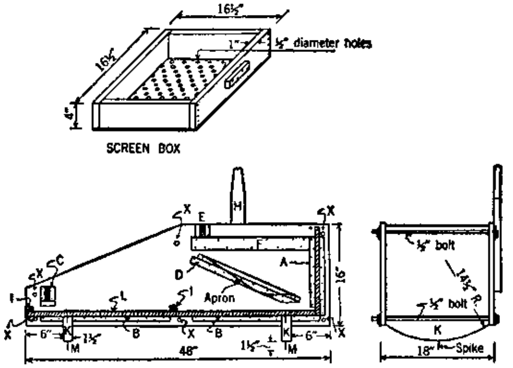

FIGURE 14, A.—Knockdown rocker. (After Eng. and Min. Jour.) *A*, Cleats for holding back of rocker; *B*, cleat for holding bottom of rocker; *C*, cleats for holding back of rocker; *D*, cleat for holding canvas apron frame; *E*, cleats for holding brace at top of rocker; *E*, cleat for holding brace at top of rocker; *F*, cleat for holding spring box; *H*, handle for rocking; *I*, riffles ¾ inch high by 1 inch wide; *K*, rockers; *L*, bottom board of rocker; *M*, spike projecting 1½ inches to prevent rocker from slipping; *X*, bolt holes for ½-inch iron bolts

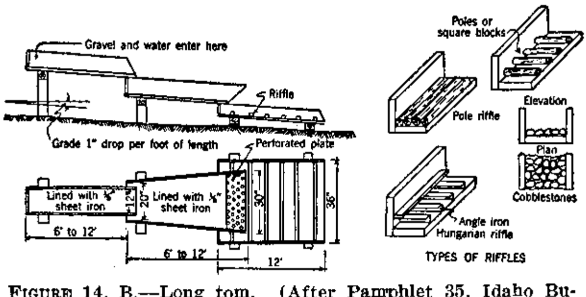

FIGURE 14, B.—Long tom. (After Pamphlet 35, Idaho Bureau of Mines)

application; but more than one method may be used at a given property to advantage, either on account of variations in conditions or to check results obtained by one method with those by another.

The pattern on which the test pits or drill holes should be laid out depends largely upon the shape of the deposit. This may not be known at the outset, so lines of holes are usually spaced relatively long distances apart, with individual holes rather far apart, from which some idea of the shape and trend of the pay gravel may be gained. Closer test pitting or drilling based upon the results of the first work may then be done to delimit the deposit and to form a basis for estimating its average tenor and total value.

If the deposit is broad, holes may be sunk at the corners of a checkerboard of squares; or if the deposit is long and narrow, following a channel, lines of holes may be sunk at right angles to the long dimension.

In shallow ground containing little water test pitting usually will be the cheapest and most accurate method of sampling the deposit. In deep or very wet placer ground the drive-pipe or churn-drill method would generally be the best to employ, the drills being operated by hand, steam, or gasoline engine. In wet ground not only would the cost of test pits be very high, but the accuracy of the samples would be vitiated by caving of the sides and washing in of extraneous material. The gold is recovered by panning or rocking each sample representing a definite depth of hole, and from the amount of gold recovered and the volume of the sample the value per cubic yard of material is calculated for each sample. By properly combining the values of all the samples from a given pit or drill hole the average value of the ground per cubic yard represented by that hole may be computed.

Estimation of the value of placer ground in advance of working is not simple, and many sources of error exist. These may be due to improper technique in taking the samples, incorrect interpretation of the results, or characteristics of the deposit beyond the control of the investigator. Suffice it to say for the purpose of the present paper that experience and judgment are required to obtain reliable results.

The following references are of interest in connection with the exploration and sampling of placer ground:

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STALEY, W. W. Elementary Methods of Placer Mining. Pamphlet 35, Idaho Bureau of Mines and Geology, 1931.

JACKSON, CHARLES F. AND KNAEBEL, JOHN B. Sampling and Estimation of Ore Deposits. Bull. 356, U. S. Bureau of Mines, 1932, 154 pp.

LODE DEPOSITS

As pointed out in a previous chapter, the important known deposits of gold ores in the United States and Canada all occur in regions where there are unmistakable evidences that there has at some time been intense igneous activity, and such regions offer the most promising opportunities for new discoveries by prospecting.

IN HILLY COUNTRY

The first clue to the presence of gold-bearing lodes in a region is often the discovery of gold placer deposits, although the discovery of placer gold in a given region has not always been followed by successful efforts to discover the original source of the placer gold in the mother lode.

A second common clue to the presence of gold-bearing lodes is "float"—pieces of rock broken off from the outcrop of the lode by erosional forces and moved from their original position by gravity and running water—commonly quartz or rock containing quartz and some gold. Bearing in mind the common character of the outcrops of gold lodes as previously discussed, float consisting of honeycombed, iron-stained quartz may be worthy of attention, even though there may be no visible gold in it. It may be well to pulverize such pieces of rock and pan the pulverized material for colors.

By following placer gold upstream or the float uphill the prospector will be moving in the general direction of the source whence the gold or float originally came, unless the direction of the drainage or slope of the ground has changed materially since the float was in its present position. When he reaches an upstream point where no more gold is found or an uphill horizon where no more float is found, its source in the lode is probably near at hand.

If the clue followed has been gold along a stream bed, search should be made for float on the slopes in the vicinity of the point where the last gold was found. If the float has been followed to its highest horizon the next step is to dig trenches into the hillside from the line along which the last float was found, removing soil and loose rock in an endeavor to uncover the lode. Sometimes the lode material is more resistant to weathering and erosion than the country rock and may thus stand up above the wall rocks, in which case little, if any, trenching may be required to uncover it. In other instances, the lode may be less resistant than the inclosing rocks, and its location may be marked by a depression and require considerable excavation to uncover it. It may be that the lode has been entirely removed by erosion, and the search for it will necessarily be fruitless.

If the surface is covered thickly with mantle rock and soil the first trenches may pass entirely over the existing top of the lode, so that deeper trenches or tunnels may have to be driven from points lower down on the hillside to reach bed rock and intercept the lode. If the lode is dipping into the hillside, a vertical shaft, started at the point where the highest float was found, will fail to strike it. Therefore, since the dip and even approximate location are not known at this stage and since the cost of sinking will ordinarily be greater than that of trenching or tunneling, the latter methods for uncovering the lode are preferable.

Trenching from the point where the last surface float was found may reach bedrock, but not the lode. If the mantle rock contains float in the face of the cut, however, it indicates that the lode may be further ahead. Figure 15 shows trenches and other prospect openings in a hillside in the Cripple Creek district, Colorado.

The foregoing indicates, in a very brief and general manner, the procedure in prospecting for lode gold in a mountainous or very hilly country. Further exploration of a discovery is touched upon in a later section.

IN FLAT COUNTRY

In relatively flat country lacking pronounced slopes, and especially in a glaciated country largely covered by glacial drift, the finding of gold float offers a yet more puzzling clue to its source and may not furnish any guide of practical value whatever, if it is found in the general glacial detritus of the region. Float may be carried hundreds of miles before coming to rest in the débris left behind by the glacier as it retreats. If found in an unglaciated, flat, or only slightly hilly country the direction from which the float came may not be evidenced by the present slope of the ground, because in such a region it is likely that the slopes have changed since the float was deposited. On the other hand, its source in the lode may be quite close at hand, although the direction in which to look may not be indicated. Large angular blocks containing lode material probably have originated from near-by sources, and all rock exposures in the near vicinity should be investigated. In such districts, trenching or test pitting must often be employed to uncover bedrock; and the lodes are as apt to lie under surface depressions, partly filled with soil, sand, and bowlders, as under higher ground. Indeed, in some very productive areas the original discoveries have been made on hillside outcrops, but the largest and richest lodes have been near by under depressions in the bedrock occupied by detrital material, or even by lakes. Since productive gold lodes occur most frequently in rocks which have been sheared and shattered by dynamic movements, a condition which favors rapid erosion, it follows that the major lodes are quite as apt to lie under depressions in the surface as under higher ground. The effect of shattering of the rocks may, on the other hand, have been to provide access to solutions which through intense silicification of the lode and immediate walls would tend to make the lode more resistant to the forces of erosion than the surrounding rocks, in which case the lode may project above the surrounding country.

Since the rock outcrops in such districts usually occur on the hilltops or along beds of streams these locations are the easiest to prospect and invariably receive first attention in an undeveloped area. Prospecting of the low areas is usually done from underground workings in adjacent lodes outcropping on higher ground and may not be attempted for years after the development of the latter, if at all. The implication in this is that some of the older districts may still offer attractive opportunities for prospecting in the vicinity of producing or worked-out mines.

EXPLORATION

The term "exploration" is often and quite correctly employed synonymously with the term "prospecting," but in this report it is used to cover operations subsequent to the finding of a gold-bearing lode that are conducted to reveal the nature and extent of the find and the grade or value of the rock before mine development and productive operations are begun.

Assuming that a lode has been discovered, the procedure will then depend upon a number of considerations, such as its position relative to surface topography; its shape, size, dip, and other geological features; and such factors as location, accessibility for men, supplies, fuel, water, etc., and the amount of money available for exploration.

It may be said at the outset that once a discovery has been made, even though high-grade material has been found, no reliable forecast is possible of the expenditure that will be required to prove or disprove the existence of a profitable deposit. The best that can be

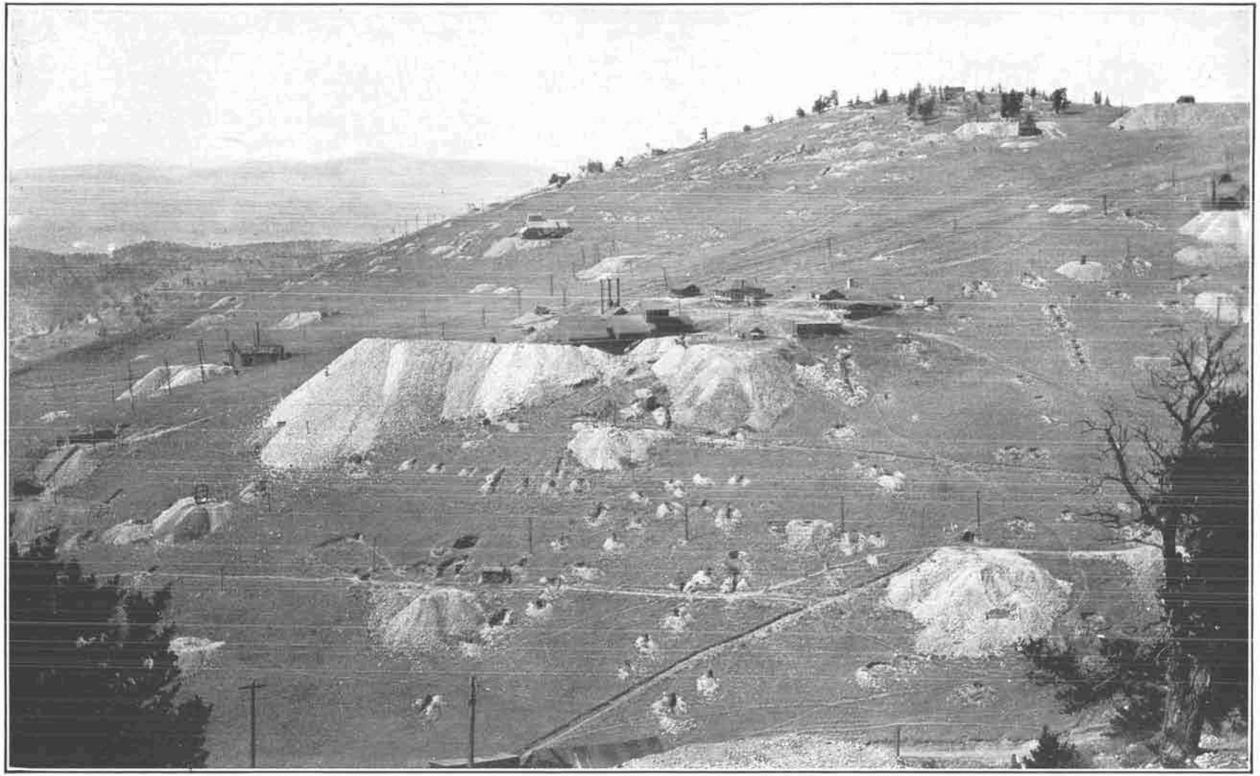

FIGURE 15.—Raven Hill, Cripple Creek district, Colorado, showing trenching and other prospect openings in hillside. (After F. L. Ransome)

FIGURE 16.—Outcrop of lode showing distortion at surface due to disintegration, slump, and creep. (After F. L. Ransome and W. H. Emmons)

done is to use the money available to the best advantage in further exploration of the prospect, basing each step on the results of the preceding work as progress discloses more and more information concerning the nature of the lode. To do this exercise of the best judgment, based upon experience, and knowledge of the nature of ore deposits and of the methods and costs of exploratory work are essential. It very rarely happens, especially in later years, after the easily discovered high-grade surface outcrops have been combed over, that a prospect can be made to "pay its way" from discovery to the point of profitable exploitation.

Vast sums of money have been spent on gold prospects, only too often unwisely. Exclusive of the fraudulent promotions which in the aggregate have taken huge sums of money from a credulous public, large amounts have been unwisely, though honestly, spent in the endeavor to develop profitable gold mines.

It would be impossible to outline a program of exploration to fit all types of deposits or even to cover the combinations of conditions that might be met in exploring a single type and to anticipate all the contingencies that might have to be met. In a great many instances exploratory work on showings of gold-bearing rock has resulted, as it progressed, in disclosures good enough to warrant further work but inadequate to meet expenses and make operation profitable. These "teasers" have absorbed large capital expenditures, a natural consequence of the lure of the presence of precious metal which may lie along the trail of profitable ore from which it is hoped the earlier expenditures may be recouped and a final profit obtained. Here again a high order of wisdom and judgment, experience, and knowledge of the habit of ore deposits is required in deciding the proper time to stop.

SHALLOW WORK

In following up the disclosures of the original discovery, the position and dip of the lode with respect to the topography have an important bearing on procedure. In any event, it is usually advisable, from the standpoint of cost, to confine the earliest work to shallow excavations, if possible, by uncovering the outcrop either continuously along its strike, especially if it is narrow, or trenching across it at right angles to the strike at frequent intervals, if it is wide. Bearing in mind the facts that lodes commonly pinch and swell, are barren in some portions, or may be cut off by faults and other structural variations, it is well to space the initial openings at close intervals. If a long interval between the last disclosure of the lode and the next excavation does not uncover the lode, it will more likely than not fail to furnish a clue to what has taken place in the intervening area.

Frequently the structure of the lode, such as the direction of banding or of shear planes, a smooth wall, or a slip revealed in an excavation, will indicate its direction and dip at that point and its probable position beyond. Here, in surface trenching, the slope of the ground must be taken into consideration in following the outcrop, since, if the lode is not vertical, its direction will depend on its dip and the surface contour. Figure 17 shows a very simple case of two lodes dipping in opposite directions but paralleling each other in strike, and outcrops diverging due to the contour of the hill. A common effect of distorted dip near surface, due to slump or creep, is shown in Figure 16. A yet more irregular topography than that shown in Figure 17 would result in more irregular outcrops.

Separate pannings of pulverized material chipped from each band, exposed in the excavations, should be made to gain an idea as to the mineralization, and the results should be recorded in a notebook, with a sketch showing the location of the samples. In addition, carefully cut channel samples should be taken from each band in the lode and in the exposed wall rocks by cutting grooves of uniform width and depth at right angles to the dip, taking about 1 pound of material per foot of channel. The positions of these samples should be recorded in a notebook with suitable sketches and the samples assayed.

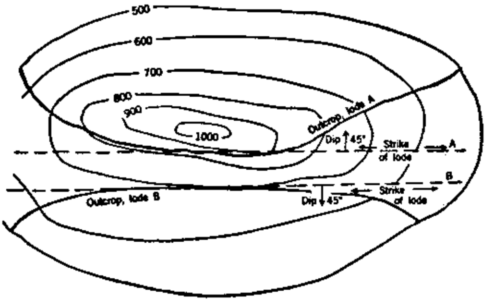

FIGURE 17.—Outcrop of two lodes on hillside, lodes dipping in opposite directions

DEEPER WORK

If the results of the preliminary shallow work are encouraging the next step is consideration of deeper exploration. In this connection, a good general rule to follow is to adopt a method which will keep the work, particularly in its initial stages, confined to the lode itself, unless certain adverse conditions mentioned later prevent this procedure.

ADIT LEVEL

The most ideal condition is where the lode strikes across the slope of the hill and an adit level can be driven in the lode from a point low on the hillside which will gain increasing depth on the lode as the level advances. (Fig. 18.) An adit is cheaper to drive than a shaft, as no hoisting or pumping is required; and if driven in the lode it gives information every foot of the way.

SHAFTS *v.* CROSSCUTS

If, on the other hand, the lode outcrops along the hill, as in Figure 19, *A*, depth may be gained either by a shaft or a crosscut tunnel. An inclined shaft in and following the lode commends itself from the fact that, as with an adit, the work is all done in the lode, barring dislocation by faulting and, except for this contingency, furnishes information regarding the lode as work progresses. A shaft, however, usually costs at least two or three times as much per foot to sink as a tunnel; and if water is encountered in appre-

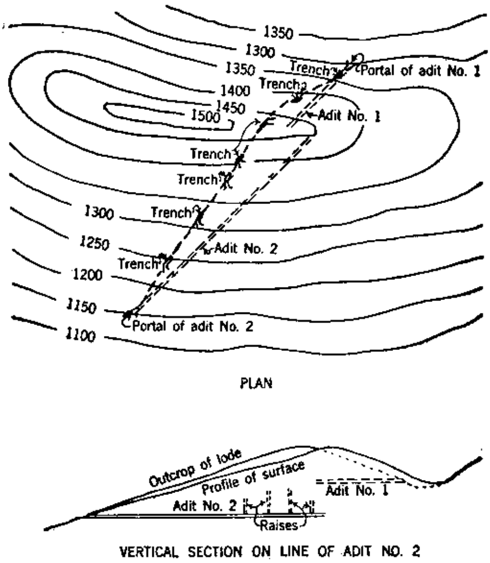

FIGURE 18.—Exploration by adit levels driven on the vein

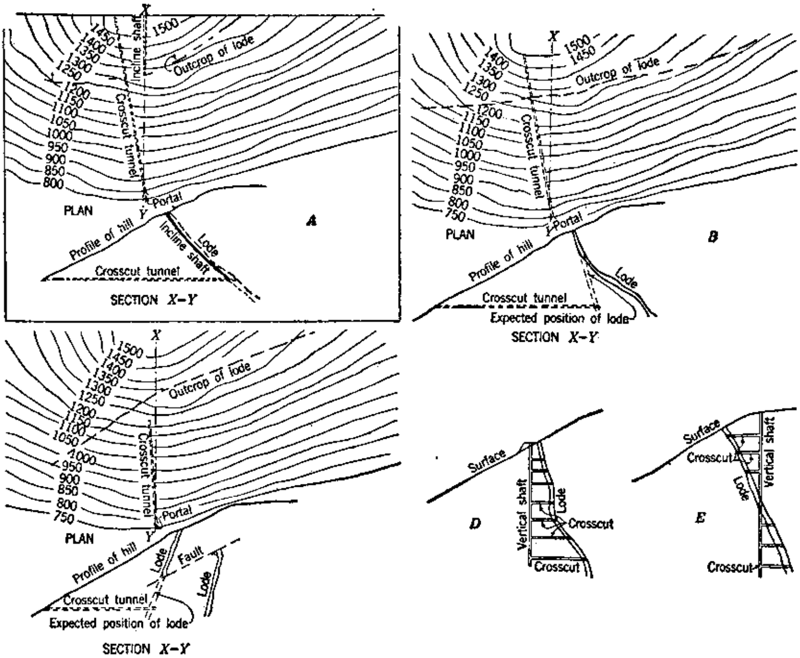

FIGURE 19.—Methods of exploring lodes at depth: *A*, Lode outcropping along hill side, exploration by incline shaft or tunnel; *B*, lode dipping into hill, flattening with depth; *C*, lode faulted; *D*, lode explored from vertical shaft in footwall; *E*, lode explored from vertical shaft in hanging wall

ciable quantities or the dip of the vein changes frequently and appreciably in degree, sinking may be very difficult and expensive. If a shaft of any considerable depth is required it is necessary to install hoisting equipment which is not needed for tunnel work, and this may require considerable cash outlay.

Based solely on the consideration of cost and assuming no unusual difficulties in shaft sinking the decision as between an inclined shaft and a crosscut tunnel would depend largely on the relative length of tunnel and shaft to reach the desired depth objective and the relative cost of handling the material excavated in the lode once it is encountered. The amount of money available must be considered here, and obviously if the funds are all used up in driving the tunnel before striking the lode their expenditure would have added nothing to the knowledge of the deposit.

Furthermore, if the lode dips away from the portal of the tunnel the length of the tunnel would increase with the depth of the objective point. If the dip were flat this increase would be rapid. (Fig. 19, *A*.) Changes in dip may result in the tunnel reaching the objective point only to find the lode is not there (fig. 19, *B*), and perhaps insufficient funds may remain with which to drive ahead to intersect the lode. Faulting (fig. 19, *C*) or a change in the strike may upset the calculations even more seriously; and since little is known regarding the lode at this juncture, there may be no evidence on which to predict in which direction it lies. In other words, exploration by a crosscut tunnel is blind work which may easily fail entirely to pick up the lode, while an incline shaft in the vein, unless it is cut off by a fault, explores it all the way as far as the money available will drive it. If the vein is nearly vertical or stands at a steep angle of dip a vertical shaft may be the best form of exploration opening. (Figs. 19, *D*, and 19, *E*.) In this event, unless the dip of the lode changes or it is displaced by a fault only a small amount of dead work will be necessary in the form of short crosscuts from the shaft. A vertical shaft is often more convenient than an incline for hoisting, and as it is not expected to stay in the lode can be kept straight. Its length to attain a given depth will be also less. An incline can be driven straight; but in this case, since most lodes do not dip uniformly, it may require crosscuts to be driven to the lode at frequent intervals.

If a vertical shaft is sunk the question arises as to whether to sink in the hanging wall or the foot wall of the lode. If sunk in the foot wall the crosscuts will be longer at each successively lower level. (Fig. 19, *D*.) If sunk in the hanging wall the levels will be shorter as the lode is approached by the shaft and then become longer after the shaft has passed into the foot wall. (Fig. 19, *E*.) A shaft in the lode or in the hanging wall may tie up some ore which can not be extracted without ruining the shaft for operating purposes later on; but since the object of the exploratory work is primarily to gain a knowledge of the lode, usually this would not be a serious consideration.

In making a decision as to the mode of exploration at greater depth all the factors involved, including the possibilities named above, must be considered. If it is decided to explore by a long crosscut tunnel or a vertical shaft, it may be advisable to do some preliminary work with the diamond drill to determine the position of the lode from the outcrop to and at the depth to which it is desired to explore. If the deposit is flat lying, so that it may be explored to best advantage by vertical drill holes, the churn drill may be substituted for the diamond drill.

DRILL EXPLORATION

Drill exploration can usually be done much faster and at a fraction of the cost per foot of tunneling or shaft sinking, and while the results therefrom are often not conclusive as to the grade of ore they will usually give at least a rough idea thereof (sometimes a very close approximation) and will locate the lode and determine its dip and its width. As in the case of tunneling, drill holes may fail to locate the lode, due to change in dip or strike, faulting, or pinching out;

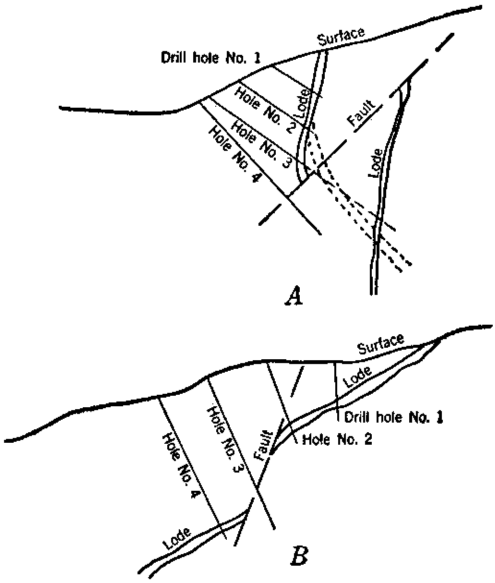

FIGURE 20.—Exploration by drill holes at close intervals: *A*, Faulted steep vein; *B*, faulted flat vein

but if this happens, the expense usually will have been much less, and the cost of drilling deeper or drilling additional holes in other directions will be less than that for crosscutting.

In drilling, it is usually advisable to drill a series of holes following the lode down on the dip and along the strike, the first being pointed to intersect it a short distance below its known location and thus decreasing the probability of missing it, due to change in dip or strike or because of faulting. (Fig. 20, *A*.) If, in the illustration, drill hole *4* were the first one drilled and were put down with a view to cutting the lode a considerable distance below the outcrop, it would fail to cut the lode at or near the expected depth and would not offer much encouragement for further exploration. This would happen if the lode were faulted as shown by the full lines or if the dip changed as shown by the dotted lines. If, however, holes *1*, *2*, and *3* were drilled in the order indicated, cutting the lode at about 100-foot intervals, they would have shown the lode to be continuous for a considerable distance and offer encouragement to drill hole *4* deeper, or perhaps to deepen hole *3*, which, as shown by the dotted line, would intersect the lode again, whether it were faulted or the dip changed, thus giving a key to the structure. Figure 20, *B*, shows a similar condition, where a flat vein has been faulted. If drill hole *3* were sunk first and near the location shown it would miss the lode entirely and even if continued would not intercept it. The drill cores might or might not indicate the presence of a fault, depending upon the nature of the fault.

The lode may also be dislocated in a similar manner along the strike, an argument for close spacing of drill holes in that direction.

Close spacing of holes obviously entails a greater expenditure for drilling to cover a given area but will explore the property more intensively. In any particular case, conditions will govern the spacing of the holes, and a balance must be struck between cost and thoroughness of development. If the preliminary surface exploration has shown the lode to be continuous and regular along the strike it is more apt, in a general way, to be regular down the dip than when it is irregular along the strike, and this factor may well be considered in determining the proper spacing of drill holes.

Large, uniformly mineralized deposits of the base metals can sometimes be explored by drilling accurately enough to form a basis for planning regular mine development, without any other exploratory work. On account of the usual erratic occurrence of gold in lodes, information as to grade of ore obtainable by drilling alone is seldom sufficiently accurate to warrant large capital outlay for plant and regular mine development. It is therefore suggested that in the exploration of gold lodes drilling is chiefly valuable as a guide to exploration by sinking, drifting, and crosscutting rather than as a basis upon which to formulate operating plans. In large, well-developed mines, however, where the geology and habit of the ore bodies are well understood, the results of drill exploration can frequently be used with considerable confidence as a basis for planning mine development.

DRIFTING

Once the lode has been opened at one or more horizons below the outcrop, whether by adits, shafts, tunnels, or crosscuts, exploration is carried further by drifting on the lode at these horizons. The drift may take out the full width of the lode if narrow or may follow one or the other of the walls or be kept as nearly as possible in the center if the lode is wide. Usually an effort is made to carry the drifts in the best ore at this stage of exploration. In any event, crosscuts are driven at intervals from the drift to the walls of the lode. Auxiliary exploration may be carried on in the form of raises or winzes to test the lode above and below the level of the drift.

SAMPLING

All openings in the lode should be sampled carefully as the workings advance. It is not within the scope of this paper to discuss in detail methods of sampling gold lodes by drilling or by channeling.

Much has been written on this subject in textbooks, the techincal press, and the transactions of technical societies, and it has been treated in various mining and prospectors' handbooks. However, for the benefit of readers who may have had little experience in sampling ore deposits, a few notes on sampling as it applies to gold-lode deposits are supplied.

The object of sampling in this connection is to determine from the samples, which consist of small portions of the ore, the average grade of the ore, the grade of different sections of the lode, and the limits of the commercial portions thereof. These factors are determined by assaying the samples. Obviously, if the samples do not correctly represent the material sampled the assay results will be misleading.

The occurrence of gold in lodes is usually very erratic, and obtaining representative samples is not a simple matter. Indeed, in some instances [1] it has been found impracticable to obtain representative small samples, and it has been necessary to resort to mill tests on considerable tonnages of ore to determine the average tenor. On the other hand, there are numerous instances where small samples are quite reliable. In most of these, however, the reliability of the sampling is not due so much to the accuracy of individual samples as to the fact that a very large number of samples are taken, in which the individual errors are compensating when averaged.

The method and details of sampling which will give the best results in any given instance will depend upon the characteristics of the lode, the distribution of the gold particles in the gangue, and the occurrence of the pay streaks.

The common methods of sampling are channel sampling, pick sampling, drill sampling, grab sampling, and bulk sampling.

CHANNEL SAMPLING

Channel sampling and pick sampling are probably the most common methods employed in lode-gold deposits and are suited to the greatest number of conditions. Channel samples may be cut either with hammer and moil or with a pick and consist of material cut from channels or grooves of uniform depth and width across the face of the ore. Hammer and moil are usually required in hard rock, while a pick may serve in softer rock. The object in cutting the channel is to obtain an equal amount of material from each unit of length of the sample, which usually amounts to about 1 pound per foot. Before the channel is cut the face should be cleaned of dirt and loose material and preferably washed with water. In many lodes the rock is banded or has a " ribbon " structure, and often the several bands or ribbons vary considerably in hardness and other characteristics. As it is very difficult to obtain equal weights of sample per unit of length from bands of different hardness it is usually best to sample and assay each band separately and measure the length of each sample; then, by multiplying the assay value of each sample by its length, adding the products, and dividing by the sum of all the lengths the average assay value over the width of the face is calculated. The samples in banded material are preferably

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[1] Bradley, P. R., Mining Methods and Costs, Alaska-Juneau Gold Mining Co., Juneau, Alaska: Inf. Circ. 6186, Bureau of Mines, 1929, 18 pp.

taken at right angles to the direction of the banding. Figure 21 shows the method advocated for channel sampling the face of a drift. Samples are usually cut in the faces of drifts after each round is shot, across the back or floor of the drift or both at regular intervals (usually about 5 feet), and sometimes along the side of a drift or crosscut. Often it is the practice to cut no individual samples longer than 5 feet.

PICK SAMPLING

Pick samples, as the term is applied in this paper, consist of bits of rock chipped from the face more or less at random but designed to select amounts of material of each sort represented approximately in proportion to the amount of each sort exposed and from points well scattered over the face. In some types of ore the results by this method have been found to check closely with those from channel

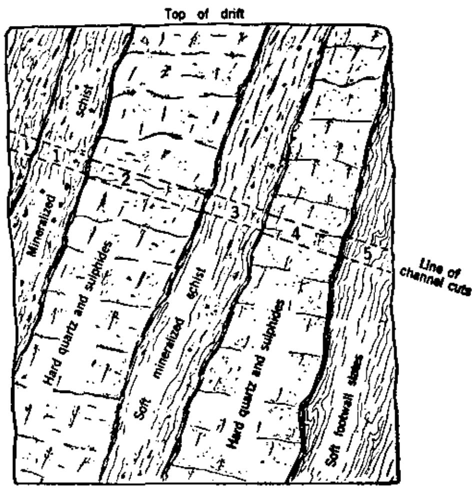

FIGURE 21.—Method of channel sampling drift face in banded vein. Numbers indicate separate samples

samples, but in the majority of instances the channel samples will be found to be more representative. If the ore is banded it is preferable to sample each band separately, giving the assay value of each sample a weight in proportion to the thickness of the band in making calculations of grade of ore as in the case of channel sampling.

DRILL SAMPLING

Drill samples may be taken with a core drill, for deep holes, or by collecting the cuttings from holes drilled with the ordinary rock drill, for shallow holes. Core drilling is usually done by experienced drill operators who understand the proper methods of collecting and recording samples, often under the supervision of a trained engineer or geologist; it is not discussed here.

Samples of cuttings from holes drilled with an ordinary rock drill have in some instances given reliable results, though in gold ores the assays are usually not deemed as reliable as those from channel sampling and are not often used in making careful estimates of the grade of ore reserves. They do, however, often serve to determine the width of commercial material on either side of a drift and thus provide added information for making estimates, and for locating parallel ore bodies in the walls of drifts and stopes.

Several different methods are employed for collecting the cuttings. Sometimes a pail, can, or empty powder box is placed on the floor below the hole, the cuttings flowing from the hole into the receptacle with the drilling water. The cuttings settle out, and the water overflows. It is usually the practice to collect a sample for every 2 or 3 feet of hole drilled. With this method considerable material is apt to be lost by running down the face and not getting into the receptacle at all. A more refined method is to drill a few inches of hole of a large diameter and insert a short piece of pipe with half of the outer end cut away to form a spout. The sample drilling is then done through this pipe, which directs the sludge into a suitable receptacle. When drilling is done dry the cuttings may be collected in a powder box held under the hole or in a sample sack secured below it. If holes are drilled at a steep angle upward the sack may have a hole on one side with a gasket around it through which the drill steel is inserted. In wet drilling the accuracy of the sample is sometimes vitiated by fine light material being carried away with the overflow water. This can only be overcome by providing ample settling capacity in the receptacle.

GRAB SAMPLES

Grab samples may be taken from muck piles or the top of loaded cars. They are usually taken haphazardly and often give very misleading results when so taken. They often consist of handfuls of material selected from several points on the muck pile or of a handful or two from the top of each car. Instances are on record where such samples are 30 to 40 per cent high. On the other hand, with some ores the average of many such samples has given surprisingly accurate results. The writers believe that grab samples, properly taken, can be as reliable as other types of samples.[2] The cost of samples so taken may in some instances be higher than warranted by the value of the information gained.

In grab sampling, the human tendency is to select the best material; or, less often, in an endeavor to counteract this tendency, a disproportionately large amount of poor material is selected. Furthermore. the fine material often carries more gold than the lumps, and if disproportionate amounts of fines and lumps are selected the samples will not be representative. In grab-sampling a muck pile these sampling errors may sometimes be greatly reduced by taking 1 shovelful out of every 10 (or other proportion) as the ore is loaded into the cars and throwing it into the sample box. Another method is to use a small scoop and a rope with knots tied in it at regular intervals. The rope is used to measure off equally spaced points on the pile or on top of the car, one scoopful of material being taken under each knot on the rope. The rope is stretched across the pile along

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[a] Jackson, Charles F., and Knaebel, John B., The Sampling and Estimation of Ore Deposits: Bull. 356, Bureau of Mines, 154 pp.

several equally spaced parallel lines, and thus the sample points are mechanically selected. If the point falls on a lump larger than the capacity of the scoop a piece approximately equal in weight to a scoopful of fine material is broken off with a hammer. Further details of this method are given in the reference cited above.

BULK SAMPLES

Bulk samples are sometimes the only kind that will adequately represent the grade of the ore. These consist of a few to several hundred tons or more of ore shot down from drift backs or stopes, and then put through a mill; the gold recovered, plus the loss in tailings, which can usually be accurately sampled and assayed, gives the total amount of gold in the sample. Dividing by the number of tons in the sample gives the grade of the ore. By taking channel, pick, or drill samples of the same material before the large sample is shot down, it is sometimes possible to determine a fairly accurate error factor which can be applied for the correction of assays of such small samples of a given ore body, based upon the results of the bulk sample.

REFERENCES

Reference is here made to the following articles on sampling:

Peele, Robert. Mining Engineers' Handbook. John Wiley & Sons (Inc.), New York, 2d ed., vol. 2, 1927, pp. 1712–1725.

Woodbridge, T. R. Ore-Sampling Conditions in the West. Tech. Paper 86, Bureau of Mines, 1916, pp. 16–47.

Brunton, David W. Modern Practice of Ore Sampling. Trans. Am. Inst. Min. and Met. Eng., vol. 40, 1909, pp. 553–596.

Rickard, T. A. The Sampling and Estimation of Ore in a Mine. Eng. and Min. Jour., 1903, 222 pp.

Jackson, Charles F., and Knaebel, John B. Sampling and Estimation of Ore Deposits. Bull 356 Bureau of Mines, 1932, 154 pp.

TREATMENT OF SAMPLES

The samples should be crushed and quartered down, one portion saved for future reference, and the other portion assayed. If facilities have been provided at the property for assaying or if there is an assayer near by so that quick returns can be obtained the assays may be used advantageously for keeping ore and waste separate on the dump. It is advisable to post the assay values as received on an assay map. The assays will guide further exploration, and the recording thereof on a good assay map will be of value in presenting the merits of the property if it becomes necessary at any time to acquire additional capital for further equipment and development.

DEVELOPMENT OF LODE-GOLD MINES

In the discussion so far it has been assumed that the material removed from the lode during the course of exploration has been put on the dump and that the work has all been done with new money. If high-grade pockets or shoots of ore are disclosed during exploration some revenue may be derived from hand-sorting the high-grade material, sacking the high-grade ore, and sending it to a smelter or refinery. In this way, the expense of exploration may be partly defrayed by the operations.

It is further assumed that up to this point the main object of the work has been to explore the property and to ascertain its size, average grade, character and distribution of the mineralization, persistence along the strike and down the dip, and that it has not yet entered the productive stage.

The passing of a gold property from the exploratory to the development stage may or may not be a well-defined phase of its history. The term "development" as employed here covers such work as shaft sinking and driving tunnels, adits, drifts, and raises to prepare the ore bodies for extraction of the ore by stoping. The exploratory openings may be employed, in part at least, for this purpose; but depending upon conditions, additional openings may be necessary or desirable for economical operation.

In this connection it may also be desirable to erect a small pilot mill for determining the best method of treating the ore and extracting the gold to help defray operating expenses as exploration and development proceed. This question will be discussed later.

EXPLORATORY WORKINGS

Exploratory workings may or may not have been driven with a view to their utility for extraction of the ore during a later productive period. Thus, when exploration reaches a stage where there are ample ore reserves to justify planning of productive operations it may be found that the shaft is too small to handle ore, timber, men, and supplies under the anticipated scale of operations and to accommodate pipe lines, electric cables, and ladderways. If the anticipated scale of operations justifies motor haulage the existing drifts, crosscuts, and tunnels may be too small and crooked. Existing openings may be improperly located for economical operation either as to their position in the lode or walls thereof or with reference to outside connections with surface transportation and mill site.

It is therefore well during the exploratory period to have in mind the future utility of exploratory workings for ore production; but, remembering that during exploration the chief aim is to find ore, the management should not be criticized too severely if the earlier openings do not fit into a comprehensive mine-development program. In this connection, however, it may be said that exploratory shafts are often sunk with too small a cross-sectional area. It will cost little more to sink the usual 3-compartment, 6 by 15 foot shaft than a 2-compartment shaft, and when finished it can be used for hoisting ore and rock in balance in two compartments, with the third compartment for pipes and ladderway. The same argument, however, does not apply to drifts and crosscuts, which usually have a combined length many times that of the shaft. Thus the additional rock to be handled and hoisted from 6 by 7 foot drifts as compared to 4½ by 6 foot drifts may involve considerable additional cost during the exploration period.

The exploration shaft may have been an incline which perhaps is crooked, and for operating purposes it may be desirable to sink a vertical shaft. The exploration shaft may have been in the lode or in the hanging wall, tying up considerable ore in shaft pillars or so located that stoping operations and probable attendant ground subsidence would endanger it. Again, the exploration may have been done through a shaft; and because of favorable conditions for a tunnel operation, coupled with unfavorable conditions for shaft operation, such as a heavy flow of water, the economies to be effected in haulage and drainage through a tunnel may make it desirable to drive a crosscut tunnel. Another set of conditions might make it desirable to substitute a shaft for tunnel operation.

Figure 22 (p. 54) is a general view of the surface at Goldfield, Nev., showing the development of many small individual properties by separate vertical shafts.

SHAFTS

This bulletin will not go into details of shaft sinking, tunneling, and drifting. For details of shaft-sinking methods and costs and a discussion of sizes and arrangement of compartments the reader is referred to an earlier work.[3] In brief, in planning a shaft the expected daily production of ore, handling of waste, handling of men, materials, and supplies, requirements for handling timber, and room for pipe lines and cables should be anticipated and provided for. Ventilation and fireproofing are other important considerations for permanent shafts.

COSTS OF SHAFT SINKING

The costs of shaft sinking vary widely, depending upon size, depth, nature of the rock, amount of water to be handled, wage rates, cost of power, materials and supplies used, and speed of sinking. A few shaft-sinking costs at gold mines in the United States and Canada follow:

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[3] Gardner, E. D., and Johnson, J. Fred., Shaft Sinking Practices and Costs: Bull, 357, Bureau of Mines, 1932, 110 pp.

Shaft-sinking costs at gold mines in United States and Canada

Mine	Location and year	Size of shaft, feet	Inclination of shaft	Depth sunk, feet	Cost per foot		Total	Remarks
					Labor	Supplies and timber		
Homestake	South Dakota, 1930–31	8.0 by 14.0	Vertical	(1)	[2] $35.00			
Argonaut [3]	California, recent	9.0 by 17.0	70°		[4] 45.00			
North Star [5]	California, 1925–26	6.33 by 15.67	Vertical	2,000	[4] 65.00		$118.38	
Do	do	6.67 by 15.0	26°	1,843	37.50			Plus bonus of $2.50 per foot for over 100 feet per month.
Do	do	7.0 by 15.0	Incline	1,071			[6] 83.73	
Sixteen to One	California, 1930–31	Winze, 7.0 by 18.0	60°	900	45.00		[7] 100.00	Approximate.
Vallecito, Western	California, recent	4.0 by 7.5	Vertical	167			39.50	
Tom Reed (Black Eagle)	Arizona	6.0 by 16.0	do	1,250			100.00	Do.
Teck-Hughes [8]	Ontario, 1928–29	6.33 by 21.33	do	1,484	55.17	44.17	96.34	
Do	do	12.5 by 13.0	do	2,284	70.38	50.37	120.75	
Ajax [3]	Colorado, 1915–16	7.0 by 16.0	do	502	25.77			
Sylvanite	Ontario, 1931	6.0 by 16.0	do	963	34.13	20.14	71.09	Started at 1,000 level.
Vipond	do	6.0 by 15.17	do	252	39.26	21.25	93.71	Started from 1,200 level.
Elkoro	Nevada, 1930	Winzes, 7.0 by 11.0	70°	646	15.44	8.38	23.82	

[1] From 2,000 to 3,100 levels.
[2] Shaft labor and explosives.
[3] Gardner, E. D., and Johnson, J. Fred, Shaft-Sinking Practices and Costs: Bull. 357, Bureau of Mines, 1932, 110 pp.
[4] Contract price, shaft labor only.
[5] Foote, Arthur B., Vertical and Incline Shaft Sinking at North Star Mine: Trans. Am. Inst. Min. and Met. Eng. Year Book, 1930, pp. 87–105.
[6] Includes $24.39 per foot overhead charge. Shaft stations costing $3,000 each not included.
[7] Untimbered. Total cost includes labor, track, signal system, sinking equipment, and power.
[8] Henry, R. J., Mining Methods and Costs at the Teck-Hughes Gold Mines (Ltd.), Kirkland Lake, Ontario: Inf. Circ. 6322, Bureau of Mines, 1930, 11 pp.

Following is a distributed cost sheet showing costs of sinking a 6 by 16 foot vertical shaft in porphyry at the Sylvanite mine, Kirkland Lake, Ontario, in 1931.

1,000 TO 1,250 LEVELS—208 FEET

	Breaking	Hoisting	Timbering	Other costs	Total	Cost per foot
Material	$1,552.46	$145.79	$1,368.58	$584.19	$3,651.02	$17.55
Labor	3,724.65	1,776.90	1,476.76	443.51	7,421.82	35.68
Overhead	1,621.60	1,275.28		701.07	3,597.95	17.30
Total	6,898.71	3,197.97	2,845.34	1,728.77	14,670.79	
Per foot	33.17	15.37	13.68	8.31		70.53

1,250 TO 1,500 LEVELS—255 FEET

	Breaking	Hoisting	Timbering	Other costs	Total	Cost per foot
Material	$1,822.80	$57.83	$2,626.69	$321.93	$4,829.25	$18.94
Labor	3,824.45	1,862.42	1,270.63	205.15	7,162.65	28.09
Overhead	1,115.41	2,344.71		664.18	4,124.30	16.17
Total	6,762.66	4,264.96	3,897.32	1,191.26	16,116.20	
Per foot	26.52	16.73	15.28	4.67		63.20

1,525 TO 2,025 LEVELS—500 FEET

	Breaking	Hoisting	Timbering	Other costs	Total	Cost per foot
Material	$4,219.33	$268.62	$5,216.35	$1,206.12	$10,910.42	$21.82
Labor	9,461.10	4,551.65	3,459.49	810.88	18,283.12	36.57
Overhead	3,449.01	3,249.93		1,775.25	8,474.19	16.95
Total	17,129.44	8,070.20	8,675.84	3,792.25	37,667.73	
Per foot	34.26	16.15	17.35	7.58		75.34

The costs of cutting shaft stations at this shaft were as follows:

1,100 LEVEL STATION—3,350 CUBIC FEET

	Breaking	Hoisting	Timbering	Other costs	Total	Cost per foot
Material	$125	$65	$40		$230	$0.07
Labor	341	138	7	$139	626	.19
Overhead	133	87		59	279	.08
Total	599	290	47	198	1,135	
Per cubic foot	0.18	0.09	0.01	0.06		.34

1,250 LEVEL STATION—8,305 CUBIC FEET

	Breaking	Hoisting	Timbering	Other costs	Total	Cost per foot
Material	$234	$55	$192	$36	$517	$0.06
Labor	813	348	109	145	1,415	.17
Overhead	270	622		164	1,056	.13
Total	1,317	1,025	301	346	2,988	
Per cubic foot	0.16	0.12	0.04	0.04		.36

1,375 LEVEL STATION CROSSCUT—1,703 CUBIC FEET

	Breaking	Hoisting	Timbering	Other costs	Total	Cost per foot
Material	$273	$3		$31	$316	$0.19
Labor	351	130			481	.28
Overhead	119	138		45	302	.18
Total	743	271		76	1,099	
Per cubic foot	0.44	0.16		0.05		.65

1,500 LEVEL STATION—9,000 CUBIC FEET

	Breaking	Hoisting	Timbering	Other costs	Total	Cost per foot
Material	$293	$140	$40	$47	$519	$0.06
Labor	1,086	414	108	8	1,617	.18
Overhead	419	515		242	1,176	.13
Total	1,798	1,069	148	297	3,312	
Per cubic foot	0.20	0.12	0.02	0.03		.37

The following figures cover in detail the cost of deepening a vertical shaft from the 1,200 to the 1,450 foot level at the Vipond mine, Timmins, Ontario, during the period December, 1930, to March, 1931, inclusive. The shaft is 15 feet 2 inches by 6 feet outside the timbers and is timbered with 8 by 8 inch British Columbia fir. Sets are placed on 5 foot 8 inch centers. A 24-hour cycle of operations was employed, and it required 51 sinking cycles to complete 252 feet of shaft.

Cost of deepening Vipond shaft

1. Preparation, installation of hoist on 1,200-foot level, etc.

	Total	Per foot		Total	Per foot
Labor, salaries, and miscellaneous	$1,160.66	$4.61	Tramming	$18.38	$0.07
Explosives	204.33	.81	Hoisting	73.49	.29
General supplies	205.92	.82	Pumping	105.37	.42
Timber	95.86	.38	Shop repairs	591.43	2.34
Rock-drill maintenance	130.78	.52	Miscellaneous	155.85	.62
Steel sharpening	135.77	.54			
Compressed air	258.65	1.03	Total preparation	3,270.27	12.98
Engineering	133.78	.53			

2. Sinking and timbering 252 feet

	Cost per foot				Cost per foot		
	Sinking	Timbering	Total		Sinking	Timbering	Total
Labor, salaries, and bonus	$27.78	$6.87	$34.65	Pumping	$1.51		$1.51
Explosives	6.99		6.99	Shop repairs	.87	$1.76	2.63
General supplies	2.54	.93	3.47	Miscellaneous	.84		.84
Timber		8.78	8.78				
Rock-drill maintenance	2.41		2.41	Total sinking and timbering	62.39	18.34	80.73
Steel sharpening	2.05		2.05				
Compressed air	2.53		2.53	Add preparation			12.98
Engineering and sampling	1.20		1.20				
Tramming	1.98		1.98	Total cost			93.71
Hoisting	11.69		11.69				

The cost of cutting stations at the 1,325 and 1,450 foot levels involving the excavation of 8,196 cubic feet of rock was $2,885.49 or $0.35 per cubic foot.

SHAFT EQUIPMENT

Headframes, hoists, cages, and skips, and other hoisting equipment and accessories essential to shaft operation are not within the scope of this bulletin. Such equipment is described in manufacturers' catalogues and in numerous articles which have appeared in the technical press. The choice of shaft equipment is a matter of importance, and its design to suit a given installation should be in the hands of a mechanical engineer conversant with mine equipment and its operation.

Figure 23 shows a headframe at a vertical shaft in the Oatman district, Arizona, and Figure 24 a headframe at an inclined shaft in the Grass Valley district, California. These are typical of headframes in the western United States.

Among the interesting features of shaft operation are the means employed for loading skips in the shaft. In mines where large tonnages are handled, and especially when hoisting from great depths, means must often be provided to load the skips rapidly without spilling ore over into the shaft. Slow loading lengthens the time of the hoisting cycle and reduces the tonnage that can be hoisted through a given shaft.

Loading pockets of various types and designs have been devised to speed up skip loading with a minimum of labor. A large pocket is usually provided to receive the ore from the mine cars or from an underground crusher. From this pocket the ore is drawn through gates into a measuring pocket which has a capacity equivalent to that of the skip. While the skip is in motion the loader fills one of these measuring pockets; and when the empty skip comes to rest below the pocket he empties the ore into the skip by simply pulling a lever or rope, or turning a wheel, and immediately signals the engineer to hoist.

A very efficient skip-loading station is shown in Figure 25. The ore is drawn from the main pocket through the ball-and-chain gate shown in the illustration. The chains are raised by turning the hand wheel, and the ore runs by gravity into the measuring pocket which is located below the platform and thus does not appear in the illustration. The measuring pocket is quickly filled and upon releasing the hand wheel the chains drop down and cut off the flow of ore.

DRIFTS AND CROSSCUTS

In planning development by drifts and crosscuts the shape, size, dip, and regularity of the lode, the nature of the ore and wall rocks, the stoping method to be employed, and the rate of output desired should all be considered.

LEVEL INTERVAL

One of the first points to be decided is the interval between levels. Here a compromise must be struck between close spacing of levels, which will increase the development cost, and wider spacing, which will reduce the cost but may introduce operating difficulties later if carried to extremes. A common level interval in gold-lode deposits of the tabular, steep-dipping type is 100 to 125 or 150 feet.

In favor of a short level interval may be mentioned: (1) More thorough exploration of the deposit prior to extraction of ore; (2) shorter connecting raises between levels, which, when over 100 feet long, increase rapidly in cost of driving; (3) easier handling of supplies, drills, and steel to and from the stopes; (4) more points of attack by stoping and hence a possibility of a higher rate of production, which is especially advantageous where slow stoping methods such as cut-and-fill and square-set stoping are to be employed.

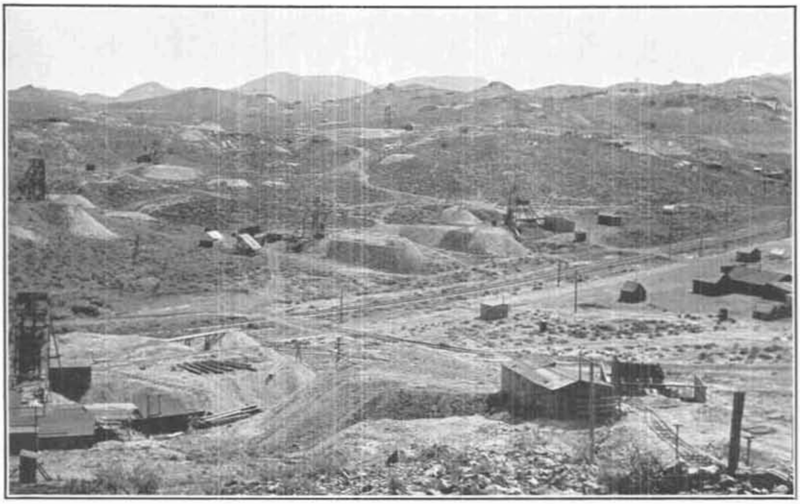

FIGURE 22.—Goldfield, Nev., showing development of small individual properties by separate shafts. (After F. L. Ransome)

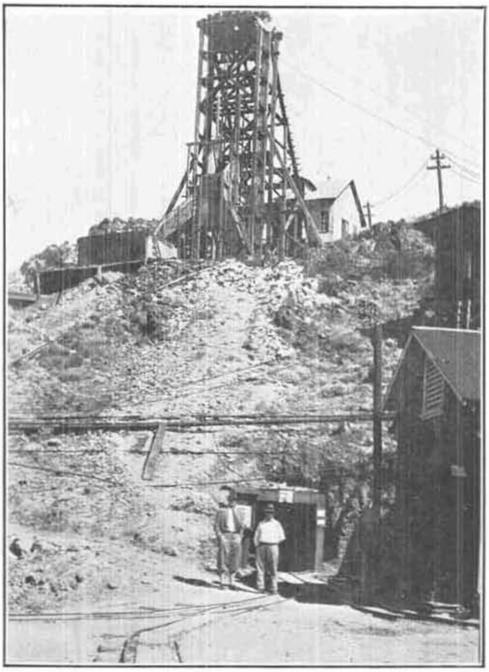

FIGURE 23.—Headframe at a vertical shaft, Oatman, Ariz., showing short adit connecting surface plant with shaft

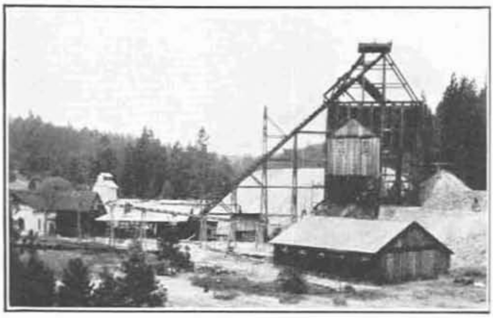

FIGURE 24.—Headframe at an inclined shaft, Grass Valley Calif., typical of small California mines. (After W. D. Johnston, jr.)

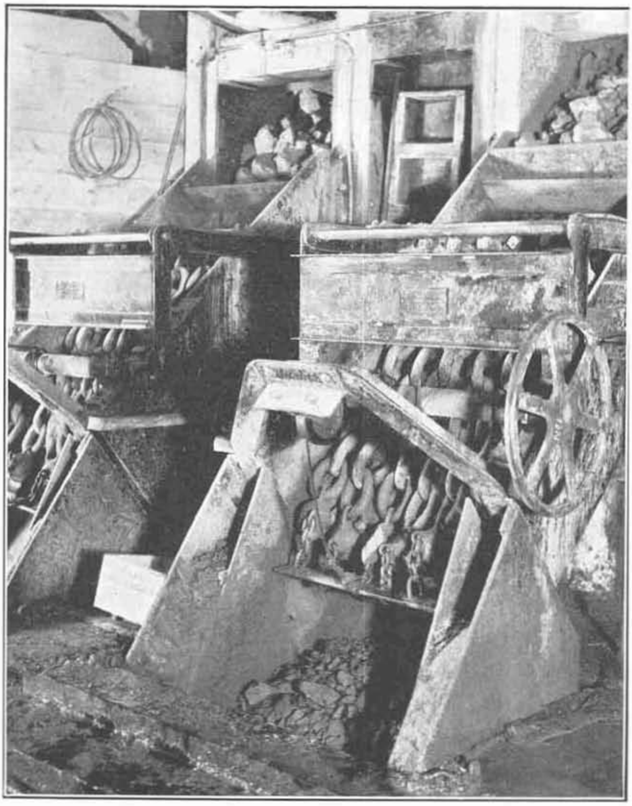

FIGURE 25.—Skip-loading station, 2,200 level, Lake Shore mine, Kirkland Lake, Ontario, showing ball-and-chain gate for filling measuring pocket

The principal disadvantage of the small level interval is the greater cost for cutting and maintaining crosscuts and drifts, with the additional lengths of pipe lines and tracks and the greater number of ore chutes. Also, more ore is usually tied up in level pillars, either temporarily or permanently.

To retain the advantages of the short level interval and at the same time eliminate excessive expenditure for shaft stations, ore pockets, and crosscuts, it is sometimes feasible to employ intermediate sublevels between the main or haulage levels. The latter are connected with the shaft, while the former are developed from raises connecting the main levels. These raises later serve for travel of men, handling of materials and supplies, and, as ore passes, for the transfer of ore from the sublevels to the haulage levels.

Level intervals adopted at a number of gold mines in the United States and Canada are shown in Table 4.

TABLE 4.—*Level intervals at North American gold mines*

Mine	Dip of lode	Inclination of shaft	Mining method	Level interval, feet
Homestake, South Dakota.	70° pitch, 40° to 50° dip.	Vertical	Shrinkage and square setting.	100, down to 1,250 feet; 150, below 1,250 feet.
Alaska-Juneau, Alaska		Tunnel	Modified shrinkage or forced caving.	250.
Argonaut, California	Ave., 63°	70°	Square setting	100 down to 3,000 feet; 150, below 3,000 feet.
Kennedy, California	58°	Vertical to 4,600 level.	do	150 (vertical).
Do	58°	58° underground shaft, 4,600 to 5,900 level.	do	
Central Eureka, California.	50° to 80°	70°	do	100, down to 200 feet (on dip).
Empire, California	11° to 35°, ave. 30°.	11° to 35°, ave., 30°	Open-stulled stopes.	Approximately 400 (on dip).
Idaho-Maryland, California.	70° to 54°	70° to 54°	do	100, down to 170 feet (vertical).
Sixteen to One, California.	35° to 40°	35°	Open stopes	200 (on dip).
Do	35° to 40°	(Winze) 60°	do	200 (on dip).
Vindicator, Colorado	70° to 80°	Vertical	Shrinkage	200.
Portland, Colorado	70° to 80°	do	do	100.
Cresson, Colorado	70° to 80° [1]	do	do	100.
United Eastern, Arizona	70° to 75°	do	Cut-and-fill	100, 150, and 200 feet.
Elkoro, Nevada	60° to 80°	Adits	Shrinkage	100 to 150 feet.
Mogollon, New Mexico	65° to 70°	Vertical and inclined.	do	200 (some 100).
Spring Hill, Montana	60° to 90°	Tunnels	Shrinkage and sublevel stoping.	100.
Hollinger, Ontario	Mostly steep	Vertical	Shrinkage and cut-and-fill.	100, down to 300 feet; 125, 300 to 800 feet; 150, below 800 feet.
McIntyre, Ontario	Steep dip	do	do	100, down to 1,000 feet; 125, below 1,000 feet.
Vipond, Ontario	Over 60°	do	Shrinkage	100, down to 600 feet; 133, 600 to 1,000 feet; 200, below 1,000 feet.
Coniaurum, Ontario	Steep	do	do	300.
Dome, Ontario	Nearly vertical.	do	do	150.
Lake Shore, Ontario	75° to 80°	do	Shrinkage and cut-and-fill.	200, down to 2,200 feet; 125, below 2,200 feet.
Teck-Hughes, Ontario	75°	do	Shrinkage	125.
Do	75°	do	do	125, intermediate; 600, main haulage.
Wright-Hargreaves, Ontario.	Steep	do	Shrinkage and oepn-stulled stopes.	125.
Sylvanite, Ontario	30° to vertical	do	do	125.
Kirkland Lake Gold, Ontario.	Steep	do	Shrinkage	100, down to 1,600 feet; 125, below 1,600 feet.

[1] Some flat dips locally.

If the mining system requires grizzly levels above the haulage levels or if a floor pillar of appreciable thickness must be left under each level or an arch pillar left over the drifts, the effective stoping height is reduced thereby. Therefore, to gain a satisfactory stoping height under either or both of these conditions it is necessary to increase the level interval above that otherwise required.

If the lode dips away from the shaft crosscuts will be longer on successively lower levels, and the flatter the vein the greater the length of crosscuts. Under these conditions the argument for wider spacing of levels gains force.

If the ore bodies are irregular in shape and occurrence, exploration and development become more uncertain as the level interval is increased; under these conditions, short level intervals are advisable.

Heavy ground and expensive level maintenance may be other arguments for a greater level interval. On the other hand, the ground may be so bad that a high stope can not be worked successfully and a short level interval, with drifts in the footwall, may be necessary. Haulage drifts in the footwall, however, entail a large amount of dead work; and, if ground conditions permit, a high level interval will reduce the cost of such dead work.

The various possible combinations of such factors require consideration for each individual case. As will be noted from the preceding table, 125 to 150 feet apparently is a satisfactory average level interval. The present tendency seems to be for greater level intervals than formerly, when 100 feet was a common standard, particularly for the upper levels in a mine. This would seem to be good practice to-day; that is, a 100-foot interval for the first levels until familiarity with the habit of the ore, character of ground, etc., is acquired, when a greater standard interval may be adopted.

SIZE OF DRIFTS AND CROSSCUTS

The size of drifts and crosscuts will be influenced by such considerations as strength of ground and requirements as to timbering for support; width of lode; type of haulage to be used, whether hand or animal tramming, storage-battery locomotives, or trolley locomotives, and size of cars to be used; type of loading chutes or platforms; economical size from the standpoint of the cost of driving; stoping method; and amount of water on the levels, which affects the size of drainage ditches.

If the ground stands well and requires no timbering the drifts need not be driven so large to accommodate given sizes of cars and locomotives with room left at the side for ditch, pipe lines, etc., since the clear opening will not be reduced by timbering. Some ground will stand very well without timbering if the drifts are narrow but will be heavy and require support if driven wide. In this connection, arching the back of the drift where feasible is of assistance. It is obvious that if timbering is required the bigger the timber used the larger must be the drifts to accommodate it.

If the lode is not much wider than the required width of drift it may be desirable to drive the drift the full width of the vein.

The cost of drifting and crosscutting varies with the " drillability " and the breaking characteristics of the ground and the size of the opening. The drillability and breaking qualities often vary with the direction of the banding, jointing, or schistosity with relation to the direction of the heading. The larger the heading, the higher will be the shoveling cost, particularly if hand shoveling is employed. Up to a certain size the cost of drilling and breaking often decreases with an increase in the size, because a better and longer break may be made each round in the larger headings. Beyond this critical size the cost per foot will increase, because more holes and more explosive will be required, as well as because mucking cost will increase. On the other hand, the cost per ton broken will generally be less with the larger heading. Thus, if the heading is in ore in firm ground it may be advisable to drive a large drift to decrease the cost per ton, especially if this will add appreciably to the daily ore tonnage and help to fill the requirements of the mill for ore. At one large gold mine in Canada it has been stated that if drifts are driven wide the cost per ton of ore therefrom is only 1.8 times the cost of the stope ore.[4] Since ore left along the sides of narrow drifts will ultimately have to be removed in robbing prior to abandoning the level or else be lost; and since the removal may be expensive if the levels have taken weight by that time, it is often just as well to drive a wide drift, other factors remaining the same. On the other hand, if the lode is narrow it may be advisable to cut down the width of the drifts to avoid breaking waste.

If hand tramming is to be used the drifts may be narrow, since the cars will be small. If animal haulage is to be employed, somewhat more height will be required than for hand tramming. If trolley locomotives are used a higher drift will be required than for storage-battery locomotives, since they should be high enough to permit carrying the trolley wires well above the height of a man for the sake of safety.

The stoping method will influence the general plan of the levels more than it will the size of the drifts but may affect the latter also. Thus, if shrinkage or cut-and-fill stoping is to be done directly on drift timbers it may be desirable to drive the original drifts high rather than to return and take down the backs later and just prior to stoping. On the other hand, if arch pillars are to be left above the drifts they may be driven low to save shoveling cost and to allow a maximum stoping height between levels. In rock headings and headings in unworkable sections of the lode small openings are advisable, commensurate of course with requirements for passage of haulage equipment and room for water ditch, pipes, and wires.

If a heavy flow of water is expected the headings should be wide enough to accommodate a ditch of ample size.

At ore-loading chutes the drifts must be high and wide enough to facilitate loading the cars and provide enough room to permit safety in loading.

The details of drifting and crosscutting methods and practices are not within the scope of this paper. For data on drilling and blasting practice the reader is referred to Drilling and Blasting in Metal-Mine Drifts and Crosscuts, by E. D. Gardner, published in 1929 as Bulletin 311 of the Bureau of Mines (170 pp.).

Table 5 illustrates the practice as to size of development headings at several gold mines in North America.

[4] Jackson, Charles F., Shrinkage Stoping: Inf. Circ. 6293, Bureau of Mines, 1930, p. 33.

TABLE 5.—*Size of development headings*

Mine and location	Character of ore or rock		Rock dimensions, feet		Type of haulage	Remarks
	Drifts	Crosscuts	Drifts	Crosscuts		
Homestake, South Dakota	Hornblende schist and quartz.	Hard schist and porphyry	6.5 by 7 to 7 by 8	7 by 7.5 to 8 by 8	Compressed-air motors, 1-ton cars.	Usually untimbered.
Alaska-Juneau, Alaska	Slate and metagabbro	Slate and metagabbro	9 by 9	9 by 9	Trolley locomotive, 10-ton cars.	Untimbered.
Argonaut, California	Quartz vein in slate	Slate, greenstone, and schist	9 by 9 to 11 by 10	9 by 9	Hand tramming	Drifts heavily timbered.
Kennedy, California	do	do	9 by 9	8 by 11	do	Do.
Central Eureka, California	do	do	8 by 9		do	Do.
Empire, California	Quartz veins in granodiorite	Granodiorite	5 by 7	5 by 7	Hand and mule tramming. Storage-battery locomotive on 4,600 level.	Untimbered.
Sixteen to One, California	Quartz vein in amphibole schist.	Amphibole schist	6 by 7	6 by 7	Hand and animal haulage	Do.
Cresson, Colorado	Veins in andesitic breccia, fairly hard.	Andesite breccia and basalt dikes, medium soft.	6 by 7 to 7 by 8	6 by 7 to 7 by 8	Hand tramming	Some timbered.
Vindicator, Colorado	do	do	4 to 5 by 6.5		do	
Portland, Colorado	do	do	5 by 7	5 by 7	do	
United Eastern, Arizona	Quartz-calcite veins in andesite.	Andesite	5 by 7.5		do	
Tom Reed, Arizona	do	do	5 by 8	5 by 8	do	
Elkoro, Nevada	Quartz veins in rhyolite	Rhyolite	5 by 7	4.5 by 6	Hand, mule, and locomotive haulage.	Usually untimbered.
Cornucopia, Oregon	Quartz veins in granodiorite and schists.	Hard granodiorite		5 by 7	Storage-battery locomotives	Do.
Hollinger, Ontario	Quartz and mineralized basaltic schist.	Basaltic schist and porphyry	8.5 by 7	8.5 by 7 to 9 by 7.5	Trolley locomotives	
McIntyre, Ontario	do	do	8 by 8	8 by 8	do	
Vipond, Ontario	do	do	6 by 7	6 by 7	Hand	
Lake Shore, Ontario	Very hard silicified porphyry and lamporphyre.	Hard porphyry and lamprophyre.	8 by 8	8 by 10	Storage-battery locomotives, 2-ton cars.	
Teck-Hughes, Ontario	do	do	5.5 by 7.5, later widened to 8.5.		Hand	
Kirkland Lake Gold, Ontario.	do	do	7 by 6	7 by 6	do	
Ankerite, Ontario	Quartz and ankerite	Tough basaltic schist and greenstone.	7 by 6	6.5 by 5	do	
Spring Hill, Montana	Hard and tough	Diorite and hard, altered limestone.	7 by 7	7 by 7	Mostly by locomotive	Do.

COSTS OF DRIFTING AND CROSSCUTTING

Table 6 shows some typical drifting and crosscutting costs at gold mines in North America. These costs are taken from producing mines. It may be noted here that unit costs would average higher at mines doing only development work, since costs of hoisting, pumping, supervision, and office and other overhead expense would have to be borne entirely by the development work, whereas in mines where stoping is going on only part of these costs would have to be borne by the development work. The differences in hardness of rock, size of headings, timbering requirements, etc., are reflected in the wide variations in costs.

TABLE 6.—*Typical drifting and crosscutting costs*

Mine	Character of ore and rock	Size of heading, feet	Cost per foot		Remarks
			Labor	Total	
Homestake, South Dakota.	Hornblende schist	7 by 7	$6 to $9, ave., $8.50.		Contract price, labor, and explosives; untimbered.
Alaska - Juneau, Alaska.	Slate and metagabbro.	7 by 5	$7 to $8		Do.
Do	do	9 by 9	$12.50 to $14.		Do.
Argonaut, California.	Slate, greenstone, and schist; quartz veins.	9 by 9 to 11 by 10.		$15 to $25.	Includes cost of timbering.
Kennedy, California.	do	9 by 9	$14		Plus $10 per set for timbering labor.
Empire, California.	Quartz veins in granodiorite.	5 by 7	$5 to $6		Contract price, labor, and explosives, including tramming up to about 400 feet (untimbered).
Sixteen to One, California.	Amphibole schist; quartz veins.	6 by 7	$6	$15 to $20.	Untimbered.[1]
Cresson, Colorado	Andesitic breccia	6 by 7 to 7 by 8.	$5.75 to $6.50.	$10	Labor cost includes explosives; untimbered.
Portland, Colorado.	Andesitic breccia (medium soft).	5 by 7	$4.19	$6.44	1927 costs; untimbered
Do	Andesitic breccia (hard and ravelly).	5 by 7	$4.82	$6.44	Do.
Tom Reed, Arizona.	Quartz; calcite veins in andesite.	5 by 8	$9 contract	$20	Untimbered.
Elkoro, Nevada	Quartz veins in rhyolite.	5 by 7 to 6 by 8.	$6.28	$11.33	Total includes $1.03 surface and general expense; untimbered.
Vipond, Ontario	Quartz veins in basaltic schist.	6 by 7		$10.97	Untimbered.

[1] Total includes explosives ($2); all labor, including sorting, tramming, and hoisting; compressed air; steel; drill maintenance; and overhead.

PLAN OF LEVELS

Most of the gold lodes mined in the United States and Canada are tabular deposits. These deposits may occur in a single lode or vein in a given mine or may occur as a series of lenses separated by barren country rock. The lenses may be parallel and lie in echelon overlapping one another; they may radiate from a central core or be confined to a narrow zone and be roughly in a straight line. Other forms of deposits are those occurring as a series of many small stringer lodes, separated by thin layers of country rock such as at Juneau, Alaska; the wide type of deposit with irregular outline typified by the Homestake ore body; and the irregular contact metamorphic type.

Each of these types requires a different development plan as to detail, but in general the procedure is to crosscut to the ore bodies from the shaft at each level and then drift on the ore. If it is difficult and expensive to maintain drifts in the ore, drifts may be driven parallel to the lode in the foot wall or hanging wall or both. Typical level-development plans for different types of ore occurrence are shown in Figures 26, 27, 28, and 29.

Figure 26, *B*, shows the development of a lode by drifts in a vein which splits at one end, drifts being driven in each split of the vein with a narrow pillar of waste between.

Figure 27 is a plan of the lower levels of the Argonaut mine, showing crosscuts from the inclined shaft to the main vein with drifts in the vein.

Figure 28 is a plan of part of the 550 level, Hollinger mine, showing the development of a series of veins from a long drift con-

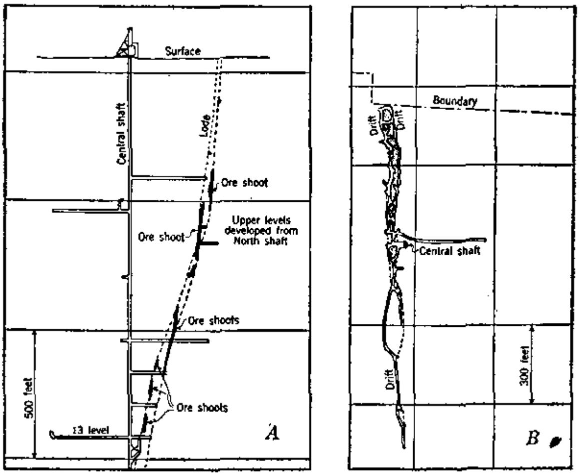

FIGURE 26.—Development of single lode, split at one end forming a hanging-wall ore body and a foot-wall ore body: *A*, Section through central shaft; *B*, plan of 13 level, Teck-Hughes mine

nected to the shafts by crosscuts and paralleling the general strike of the veins, from which a series of crosscuts is driven to cut the several veins. Each vein is then developed by drifting therein.

Figure 29 is a plan of part of a level, showing the method of developing a wide ore body at the Homestake mine by hanging-wall and foot-wall drifts from which crosscuts are driven across the ore body at regular intervals. The hanging-wall and foot-wall drifts are connected to the shaft by a crosscut on each level.

Under the conditions shown in Figure 26 it will be noted that very little crosscutting is necessary, the development work being almost entirely confined to the vein.

Figure 27 shows a condition requiring considerably more crosscutting, and in Figure 28 it will be noted that a large amount of such work is required to develop the large number of ore lenses scattered over the area of the mine. Figure 29 shows the dead work required in blocking out the ore from hanging-wall and foot-wall drifts with crosscuts to the ore body at the Homestake mine. This method of development is due to the great width of the ore body and the method of mining by transverse stopes separated by pillars, each

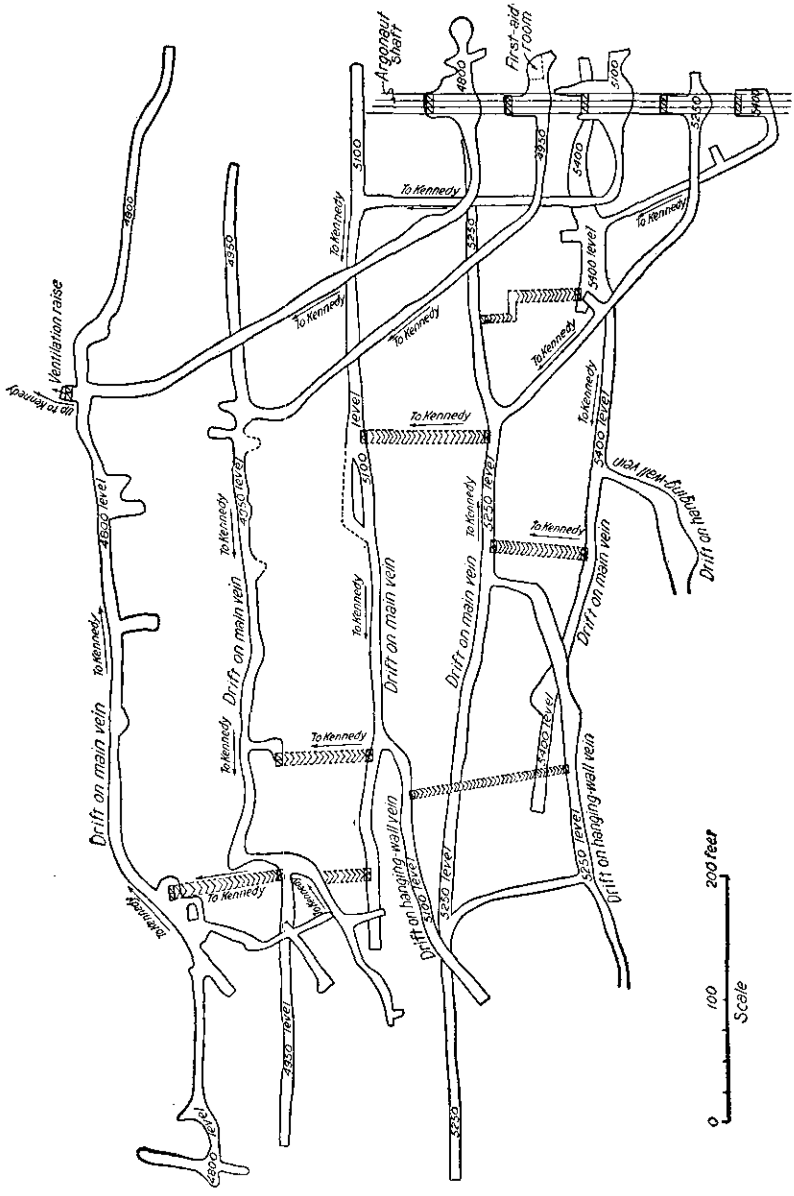

FIGURE 27.—Plan of lower levels, Argonaut mine, showing development headings

stope and pillar requiring a crosscut for its development. The foregoing indicates the variation in the amount of level development work required under different conditions and with different types of ore bodies and methods of mining.

RAISES AND SUBDRIFTS

In addition to the level development by drifts and crosscuts, most mining methods require that some additional development be done before stoping can be started. For longitudinal shrinkage stoping

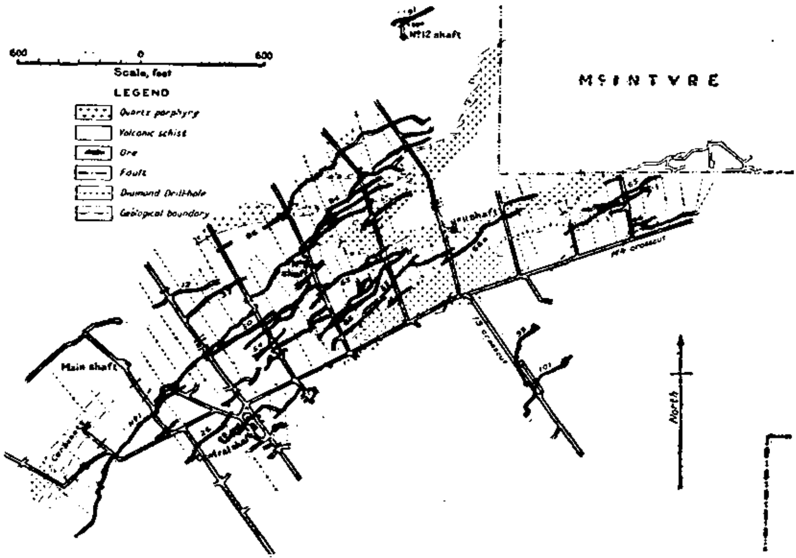

FIGURE 28.—Plan of 550-foot level, Hollinger mine, May, 1924, showing plan of development. (After A. G. Burrows)

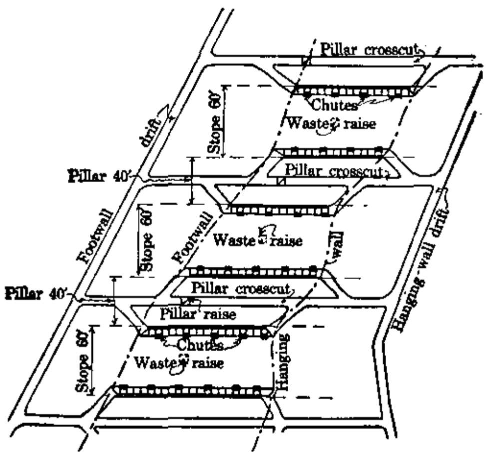

FIGURE 29.—Part of level, Homestake mine, show-showing plan of development

directly on drift timbers this may consist of only an occasional raise to the level above for ventilation, travel, and handling of supplies, drill steel, etc. If shrinkage stopes are to be carried on arch pillars over the drifts a series of short raises is put up to the level at which stoping is to start, and their tops are connected by a subdrift.

If cut-and-fill stoping is to be employed raises must be put up to the next level at regular intervals from which to start the stopes and through which waste filling can be introduced from above. This statement may be qualified by saying that where waste is not available from other sources filling material is sometimes obtained by driving into one of the walls of the stope and mining or caving waste there.

If sublevel stoping is the method employed starting raises must be put up to the next level above for each stope, and sublevels must be driven therefrom. Also, chute and manway raises will be required for drawing off the ore and for access to the stopes.

In square-set stoping, a through raise up to the next level is generally required for each stope for introduction of filling, handling timber and supplies, etc.

For mining by caving or forced caving, as at the Alaska-Juneau mine, considerable preparatory work is required for ore chutes,

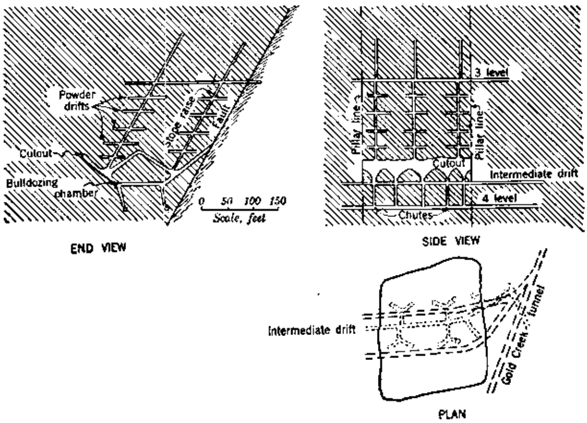

FIGURE 30.—A fully developed stope in North ore body ready for powder drift blasting, Alaska-Juneau mine

grizzly levels and grizzly chambers, stope raises, etc. Figure 30 shows a fully developed stope in the North ore body of this mine.

Figure 31 shows the plan of development for wide ore bodies at the Dome mine, on which may be noted the large number of chute raises (box holes) for drawing off the ore from shrinkage stopes, which are carried on drift pillars.

Raises and subdrifts constitute the principal work of stope development or stope preparation, as it is sometimes called. This work is often carried on the books as a part of the stoping cost, since it is usually done to a large extent concurrently with actual stoping. It constitutes the final stage of development and immediately precedes stoping.

EXPLORATION DURING DEVELOPMENT

During the progress of current mine development it is good practice to obtain as much additional information as possible regarding the deposit by careful geological mapping of the formations, which can best be done while the headings are being driven and the walls are fresh and before they are altered by exposure to air and dampness or covered by a film of rock dust; by careful sampling of all headings, especially where they show signs of mineralization; and by frequently drilling test holes into the walls for determining the presence or absence of ore bodies therein. In drifting on the lode it is often the practice to drill a short test hole into the wall on one or both sides with the drilling of each heading round, the cuttings from which are caught and examined visually and assayed if they are mineralized. These test holes determine the width of the lode and of the ore and detect any parallel ore shoots that may be present.

The necessity and value of such work will vary with different modes of ore occurrence, and the amount to be done will have to be determined for each individual project.

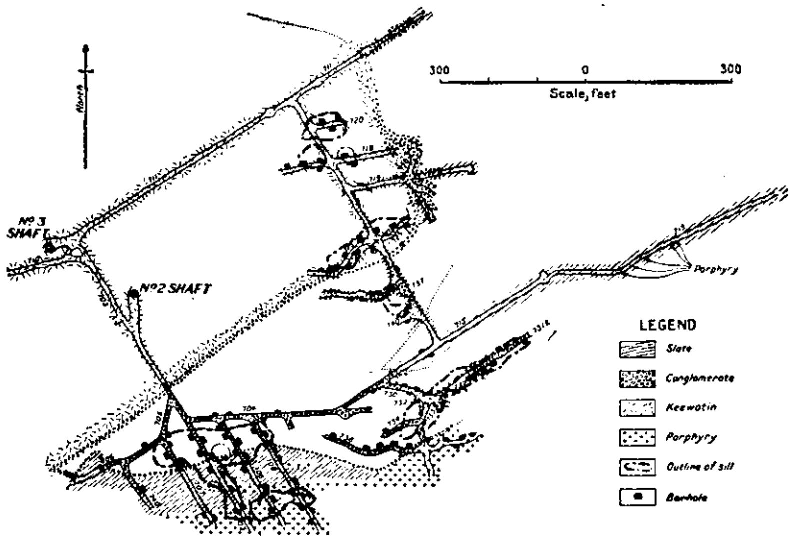

FIGURE 31.—Plan of seventh level, Dome mine, October, 1924, showing level development and box-hole raises to stopes in wide ore bodies. (After A. G. Burrows)

COSTS OF DEVELOPMENT

The cost of mine development varies greatly with the type, size, dip, and regularity of the ore bodies and the interval between lodes or between the ore shoots in each lode. These physical characteristics govern the mode of development, whether by adits, crosscut tunnels, or shafts and the amount of development in waste and in ore. The stoping method employed will also influence the amount of development work required, especially the amount of stope preparation.

Where the lode is regular in strike, dip, and occurrence of the ore shoots from one level to the next, advantage may be taken of this condition in planning development in advance so that it may be kept at a minimum. On the other hand, where there is a series of disconnected lodes or ore shoots, where the lode is displaced frequently by faults, or where strike and dip change erratically or there are other marked irregularities in ore occurrence, the development on any given level can not be planned in detail in advance. Often drifts and crosscuts may have to be driven for purely exploratory purposes, and some of the headings may have little, if any, use for ore-extraction purposes later. Knowledge of the structural geology and of the habit of ore occurrence, assisted by diamond drilling to determine the structure and to locate the ore shoots in advance of and during drifting and crosscutting, are of great benefit in planning economical development and will often save much unnecessary expense. The extent to which the diamond drill may be used to advantage will depend on geological conditions, such as type, size, and dip of the ore bodies.

TONS PER FOOT OF DEVELOPMENT

The number of tons of ore developed per linear foot of development headings may be taken as a measure of the cost of development for different types of ore bodies and ore occurrence. Table 7 shows this relationship at a number of gold mines.

The figures in this table are based principally on current development in producing mines and, except in the instances noted, do not take into account major development, such as shafts and other main openings driven before productive operations were begun.

TABLE 7.—*Relationship between tons of ore mined and linear feet of development work*

Mine or district and year	Character of deposit	How developed	Method of stoping	Approximate tons of ore mined per linear foot of development	
				Stope preparation only	Total development
Homestake, South Dakota, 1929.	Large ore bodies	Vertical shafts	Shrinkage and square sets.	200	
Do	do	do	Shrinkage, square setting, and caving.		60
Alaska-Juneau, Alaska, 1930.	Stringer lodes in wide shear zones.	Tunnels	Modified shrinkage or forced caving.	[1] 460	297
Alaska-Juneau, Alaska, 1914 to 1930 (inc.).	do	do	do	285	177
Argonaut, California, 1929–1930.	Single main vein	Incline and vertical shafts.	Square sets		27
Elkoro, Nevada, 1924	do	Adits and winze.	Shrinkage	40	16
Elkoro, Nevada, 1930	do	do	do	89	8.8
Mogollon, New Mexico, Last Chance and Confidence lodes.	5 main ore shoots	Inclined and vertical shafts.	do		[2] 16
Cripple Creek district, Colorado.	Fissure veins	Vertical shafts	Shrinkage and square sets.		35
Kirkland Lake district, Ontario:					
Mine 1, 1929	Single main vein, steep dip.	do	Shrinkage	75	53
Mine 2, 1929	2 veins, 300 feet apart.	do	do	59	51
Mine 3, 1929	2 main veins, other smaller ones.	do	Shrinkage and open-stulled stopes.	33	10
Mine 4, 1929	Several narrow widely separated veins.	do	Shrinkage and stulled stopes.	50	25
Porcupine district, Ontario:					
Mine 1, 1929	Several veins close together.	do	Shrinkage	222	16
Mine 2, 1929	Series of veins	do	Shrinkage and cut-and-fill.	150	21

[1] Including coyote drifts.
[2] Average for over 9 years.

TABLE 7.—*Relationship between tons of ore mined and linear feet of development work*—Continued

Mine or district and year	Character of deposit	How developed	Method of stoping	Approximate tons of ore mined per linear foot of development: Stope preparation only	Approximate tons of ore mined per linear foot of development: Total development
Hollinger, Ontario, 1930	Series of lenses in basaltic schists.	Vertical shafts	Shrinkage and cut-and-fill, some open-cut.		[3] 23.8
Lake Shore, Ontario, year ended June 30, 1931.	2 principal veins 300 feet apart.	do	Shrinkage and cut-and-fill.	[4] 50.4	[5] 19.7
Teck - Hughes, Ontario, year ended Aug. 31, 1931.	1 main vein	do	Shrinkage	[6] 85.2	[7] 19.8

[3] From published annual report.
[4] Based upon 13,878 feet of raising.
[5] Based upon 11,203 feet of drifting, 4,330 feet of crosscuts, 13,878 feet of raises, 1,080 feet of shaft sinking, 304 feet of shaft raising, 3,783 feet of ore passes, and 795 feet of winze. (From published annual report.)
[6] Based upon 4,649 feet of raising.
[7] Based upon 11,220 feet of drifting, 3,504 feet of crosscuts, 4,649 feet of raising, 298 feet of winze, and 314 feet of shaft sinking. (From published annual report.)

CURRENT MINE-DEVELOPMENT COSTS

Table 8 shows the cost of current mine development at a number of producing gold mines in the United States and Canada, per ton of ore mined. It is obvious that not only the character of the deposit and the methods of mining and development but also the rates of ore extraction will influence the costs when they are expressed in this manner. Thus, during some years there may be abnormally large or small development charges, due respectively to expansion or contraction in development programs, though production may remain fairly constant. Also, with an equal amount of development each year and a fluctuating rate of production the development costs per ton will vary. The sinking of a new shaft or other major project will necessarily increase the development cost appreciably.

The table lists costs at some mines where the development for the period was about normal; at others, the costs include those for shaft sinking and other abnormal development.

TABLE 8.—*Cost of current mine development at some producing gold mines, United States and Canada*

Mine or district and year	Character of deposit	How developed	Method of stoping	Tons of ore produced during period	Cost of development per ton of ore produced
Homestake, South Dakota:					
1929	Large, wide ore bodies.	Vertical shafts	Shrinkage, caving, square setting	1,437,935	$0.130
1930	do	do	do	1,364,456	.181

TABLE 8.—*Cost of current mine development at some producing gold mines, United States and Canada*

Mine or district and year	Character of deposit	How developed	Method of stoping	Tons of ore produced during period	Cost of development per ton of ore produced
Alaska-Juneau, Alaska; 1928.	Stringer lodes in wide shear zones.	Tunnels	Modified shrinkage or forced caving.	3,670,910	.061
Argonaut, California; 1 month, 1930.	Quartz vein in slates and greenstone.	Incline shaft	Square set and fill	7,800	.165
Elkoro, Nevada; 1930	Veins in lava flows	Adits and winzes	Shrinkage and cut-and-fill.	57,539	1.298
United Eastern, Arizona; January, 1917, to May, 1925.	Wide veins in andesite.	Vertical shafts	Cut-and-fill	732,528	.518
Spring Hill, Montana; Aug. 1, 1929, to Apr. 1, 1930.	Contact metamorphic deposit.	Adit levels	Shrinkage and sublevel stoping.	43,323	[1].377
Teck-Hughes, Ontario; year ended Aug. 31, 1931.	1 principal vein in porphyry and lamprophyre.	Vertical shafts	Shrinkage	396,200	[2]1.18
Lake Shore, Ontario; year ended June 30, 1931.	2 main veins 300 feet apart.	do	Shrinkage and cut-and-fill.	698,624	[3]1.403
Sylvanite, Ontario; October, November, December, 1930.	Several narrow veins in porphyry.	do	Shrinkage and open-stulled stopes.		[4]
Wright-Hargreaves, Ontario; 1930.	Several veins in porphyry.	do	do	220,430	1.072
Kirkland Lake Gold, Ontario; 1 month, 1929.	Veins in porphyry and lamprophyre.	do	Shrinkage	4,108	2.87
McIntyre, Ontario; year ended Mar. 31, 1931.	Lenses in sheared basaltic schist.	do	Shrinkage and cut-and-fill.	558,115	[5].424
Vipond, Ontario; year ended July 31, 1930.	do	do	Shrinkage	104,381	.728

[1] Lower than normal cost.
[2] All development and exploration, including shaft sinking.
[3] Includes shaft sinking 1,080 feet, drifting 11,203 feet, crosscutting 4,330 feet, raising 13,878 feet, shaft raising 304 feet, ore passes 3,783 feet, winze 795 feet, and station cutting 121,891 cubic feet.
[4] 1.229 plus 0.930 shaft sinking.
[5] Total development, including proportion of tramming, hoisting and pumping, $0.7822. Exploration and examining prospects, $0.1324.

PILOT MILLING PLANT

As previously stated, it may be desirable to erect and operate a small milling plant or pilot mill during the development period or even during the exploratory period.

Without going into the various methods of treatment at this point in the discussion it may be said that it is generally advisable to have the ore tested first by specialists who have the necessary equipment for determining the most suitable treatment to employ for the particular ore in question. The expense incurred thereby will probably avoid costly mistakes and may often result in the erection of a pilot mill which can later be used as a unit in a larger mill should the scale of operations be expanded at any time.

The optimism commonly attendant upon the discovery and preliminary development of a gold-bearing lode only too often biases the judgment of the owners of the property to the extent that money is expended upon the construction of a mill before it is warranted by the tonnage of ore proved; or, where a small mill is justified, plans are made and carried out for the erection of a much larger mill than justified. The net result in either case has often been that all the available funds are exhausted on mill construction, leaving nothing for operating expenses and further exploration and development of the property.

Here again it is emphasized that sound judgment, backed by experience and a knowledge of ore deposits and of mining and milling practice, is required in deciding when a mill is warranted and of what capacity it should be. Aside from large deposits of low-grade ore, which obviously can only be worked profitably on a large scale, it is best to consider only a small mill at first and to err on the side of too small rather than too large a capacity. The quantity and grade of ore assured by the exploration and to a smaller extent the probabilities of additional ore indicated by the continuity of the ore bodies, together with such considerations as capital available, will be the governing factors. It may be said that it is not advisable in any event to build a mill until the net value of gold in ore on the dump and assured in the mine, after mining, milling, and marketing costs are deducted, will more than pay for its construction. Up to this time it is better to put the available funds into further exploration and development, as it is usually easy to obtain funds for a mill once enough ore is assured.

Generally speaking, a small mill is often warranted on a gold property, however, at an earlier stage of development than at a property producing base metals only. In the treatment of base-metal ores the mill is usually a concentrator, the product of which must be shipped to a smelter for extraction of the valuable metals. Concentration reduces the bulk and weight of material to be shipped; but even then the cost of transportation, with smelter charges and deductions, may take up a large part of the value of the ore. Facilities for hauling the concentrates, such as roads or tramways, may have to be provided, thus requiring further expenditure of capital.

In the case of a gold ore, however, the gold can usually be extracted in the form of bullion of a high fineness by relatively simple processes right at the mine if the ore is not a complex one, and the final product can be shipped at slight expense.

Furthermore, with base ores the value of the metals fluctuates, sometimes widely, and in some instances the value may fall during the construction of the mill to a point where mining and milling will not pay and the plants will have to remain shut down to await a more favorable market. With gold, on the other hand, the product is always marketable at a fixed price per ounce for gold of the fineness produced, and the value of ore in sight can be computed with some degree of confidence. For these reasons it may be stated that construction of a mill at a gold property is often warranted earlier in mine development than at one where base-metal ores are mined. Indeed, it is common practice at small gold properties to erect a small mill at a very early stage; sometimes this enables development operations to be carried on with little, if any, expenditure of new money or at least with relatively slight additional outlay.

STOPING METHODS

The beginning of stoping may be considered to mark the opening of the active productive life of a mine. Considerable ore may have been produced, however, from previous driving of exploratory and development openings, and some financial return may have been received therefrom.

Stoping may begin very early in the development or even in the exploratory stage of a mine and be continued during further exploration and development as rapidly as development work makes ore available. Thus the ore may be stoped at such a rate that never more than a few weeks' or months' supply of ore is blocked out ahead of extraction by stoping. In this event the life of the operation is apt to be precarious, and careful planning for the most economical operation will be difficult. However, where sufficient funds are not available for development much in advance of ore extraction this method of operation is the only alternative. Under these conditions it might often be advisable for the owners of the property to consider disposing of all or of an interest in the property to a well-financed mining company, either for cash or an exchange of shares. In this way, financial disaster to the original owners might often have been averted in the past.

For the most economical operation, then, an appreciable reserve of ore blocked out in advance of stoping operations is decidedly advantageous. Not only will this permit advance planning of methods of ore extraction, with attendant savings in operating costs, but also proper balancing of month-to-month development with stope production and of rate of output with mill capacity and with capital expenditures for mining and milling plants.

It is not proposed that the complete development of a gold-bearing deposit be attempted before production is begun, even were such a procedure possible. Tying up capital in development of ore too far in advance of its extraction, with attendant piling up of interest charges, would obviously be folly, especially at small operations where the yearly drain on ore reserves and the capacity of milling plant are, and can only be, small. With large, low-grade deposits which can only be operated profitably through large-scale operations requiring heavy capital expenditures for plant and equipment, a different picture is presented. To warrant large expenditures there must be an assured reserve of ore large enough to return the capital investment and an attractive profit.

CLASSIFICATION

Stoping methods may be classified on the basis of the method of support during the active life of the stope as follows:

- A. Stopes naturally supported:
  - 1. Open stopes—
    - (a) Open stopes in small ore bodies.
    - (b) Sublevel stoping.
  - 2. Open stopes with pillar support.
- B. Stopes artificially supported:
  - 3. Shrinkage stoping.
  - 4. Cut-and-fill stoping.
  - 5. Square-set stoping.
- C. Caved stopes:
  - 6. Caving—ore broken by caving.
  - 7. Top slicing.
- D. 8. Combinations of supported and caved stopes.

SHRINKAGE STOPING

Of these methods, shrinkage stoping is perhaps the most widely used in the gold mines of the United States and Canada. The method is particularly applicable to the mining of steep-dipping tabular deposits of ore, firm and strong enough to stand without support over widths equal to the width of the lode and having firm walls which will stand well during the mining and emptying of the stope without excessive dilution of the ore with waste during the latter operation. It may also be applied to wide deposits of firm ore

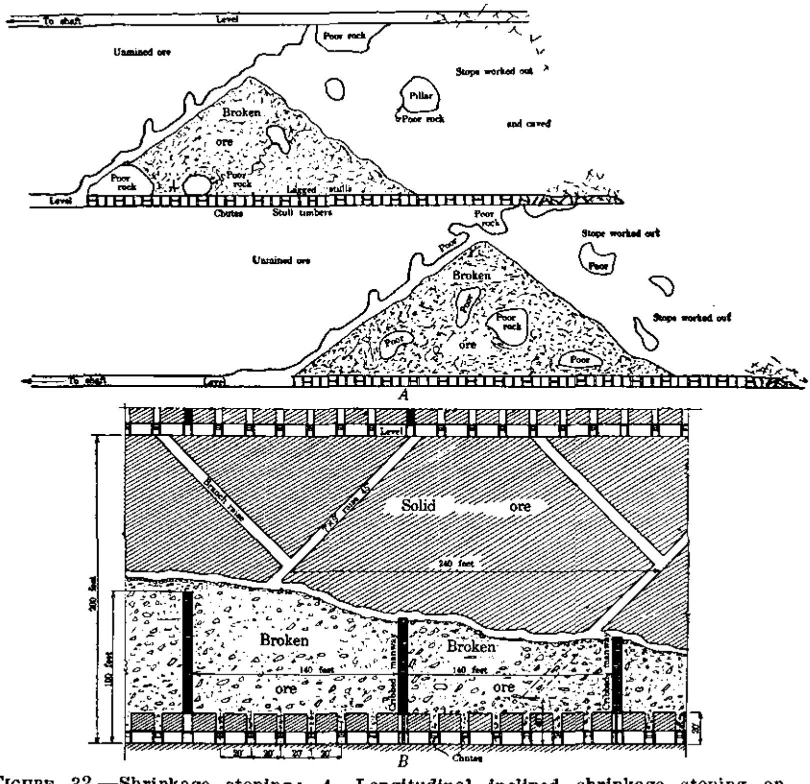

Figure 32.—Shrinkage stoping; *A*, Longitudinal inclined shrinkage stoping on timbers; *B*, shrinkage stoping over box-hole pillars at Kirkland Lake, Ontario

by carrying transverse stopes of a width suited to the nature of the ground, these stopes being separated by pillars of ore, which can be extracted later after the stopes are filled.

In shrinkage stoping (fig. 32) the ore is stoped upward from the level. The miners stand upon ore broken during the preceding cuts, enough ore being drawn off through the chutes after each cut is taken to leave room to work between the top of the pile of broken ore and the unbroken back. The broken ore is supported either upon timbers over the back of the drift (fig. 32, *A*) or upon an arch of solid ore, sometimes termed a "box-hole pillar" (fig. 32, *B*).

With this method, very little stope preparation is required, especially where the lode is narrow or the broken ore is supported on timbers. If the ore is supported on arch pillars left over the drifts short raises must be put up at close intervals and connected over their tops before stoping can be started. With either method, however, stoping can follow closely upon the heels of the level development, and extraction of ore is not long delayed. As a rule only about 40 per cent of the ore can be drawn from the stopes as it is broken; however, the balance being drawn after the stope has been completed to the top of the ore, to the level above, or to the floor pillar left under that level. Thus, considerable capital may be tied up in breaking ore which is not immediately available for extraction. This is not an unmixed evil, however, since the reserve of broken ore may provide a continuous supply to the mill during periods when for any reason the rate of breaking is reduced.

Shrinkage stoping has been discussed by Jackson in considerable detail in an earlier circular,[5] which contained 23 illustrations and covered the application of the method, its advantages and disadvantages, the variations of the method, recovery of pillars, and costs of development and stoping.

The well-known gold mines using shrinkage stoping include the Homestake, Elkoro, and Portland and other mines in the Cripple Creek district; Alaska-Juneau, which uses a modified-shrinkage or forced-caving method; and most of the Ontario gold mines, including the Hollinger, Dome, McIntyre, Vipond, Coniaurum, Teck-Hughes, Lake Shore, Wright-Hargreaves, Sylvanite, and Howey.

In very wide ore bodies the stopes may be carried across the vein. In this instance, it is usual to run a series of stopes separated by pillars of ore, the width of stopes and pillars being proportioned to suit ground conditions. When the stopes are completed they are drawn empty and filled with waste, the pillars then commonly being mined by shrinkage, top-slicing, cut-and-fill, or square-set stoping. (Fig. 33.) The latter is the method employed at the Homestake mine in South Dakota. Figure 34 shows a typical stope in the Homestake mine. The stope has been cut out on the sill floor, chute-line timbers (shown at the extreme right) have been erected, and waste filling is being introduced from a raise at the far end of the stope prior to beginning shrinkage stoping above the level.

CUT-AND-FILL STOPING

Where the walls of the lode are too weak to stand long without support the cut-and-fill method is widely employed. In this method, instead of stoping up from the level on broken ore the ore is drawn out of the stope as it is broken, and the space is filled with waste rock or tailings brought in on the level above and dumped into raises connecting with the stope.

With this method one or more raises are put up to the level above for each stope. Then, starting at a given raise, a cut is taken over the back of the drift, breaking the ore down onto the drift timbers. As in shrinkage stoping, however, the stoping cut may be started some distance above the back of the level, leaving an arch pillar over the drift. (Fig. 36.) This ore is then shoveled or dropped into cars on the level. In horizontal cut-and-fill stoping the cut is

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[5] Jackson, Charles F., Shrinkage Stoping: Inf. Circ. 6293, Bureau of Mines, 1930, 54 pp.

advanced over the back of the drift or over the drift pillar, and the ore is drawn off as it is broken until the cut reaches the end of the ore shoot, the next raise, or a similar cut driven therefrom or the pillar line, if individual stopes are separated by pillars. The excavation is then filled to within about 3 feet of the back with waste and the stoping cuts are repeated. As the stope rises timbered raises are built up ahead of the filling to form chutes through the fill into which the ore may be dumped as it is broken and drawn off into the mine cars on the level below. (Fig. 36.) The filling is dumped or shoveled in to fill the space between and around the chute raises. If the distance between waste raises is considerable the waste may be loaded into cars in the stope, trammed to the edge of the fill, and dumped.

The more modern practice is to use power scrapers for placing the filling material, particularly in wide ore bodies, and thereby elim-

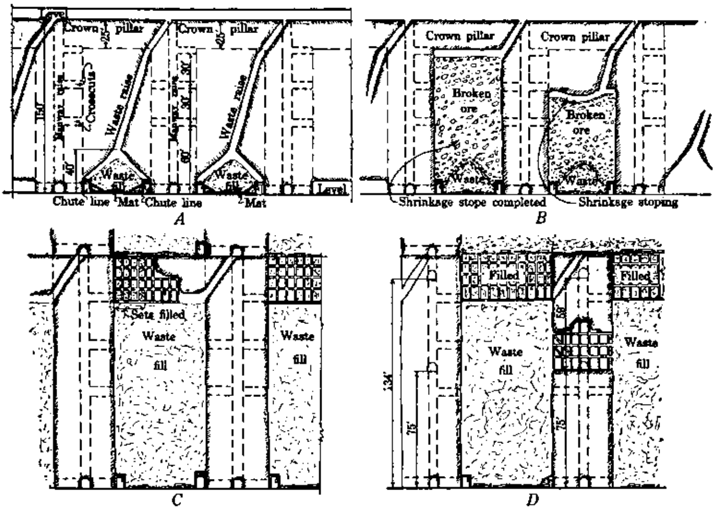

FIGURE 33.—Stoping method at Homestake mine, Lead, S. Dak.: *A*, Longitudinal section showing stopes arched and filled, ready to start shrinkage stoping; *B*, section showing shrinkage stoping; *C*, section showing mining of crown pillar by square sets; *D*, section showing mining of stope pillar by square sets

inating most of the hand work. (Fig. 35.) The same power scrapers may be employed to drag the broken ore into the chutes. Where hand shoveling is employed in the stopes the chutes are usually close together, commonly 20 to 30 feet apart, to obviate the necessity of reshoveling the ore or of loading it into cars. The labor and material costs of constructing these chutes are appreciable items in the cost of stoping. Furthermore, the time required to construct them and to lay tracks for handling waste in cars retards breaking of the ore. With scrapers, however, the chutes may be placed farther apart; and a 100 to 150 foot interval is practicable, particularly after the stope is high enough to provide enough storage capacity in the chutes so that the scraping operation will not be delayed because the chutes are full. Chutes may be installed at 50-foot intervals and carried up a short distance. As the stope rises, every other one may be dropped and covered over, and the chute interval from there up

FIGURE 34.—Stope in Homestake mine, South Dakota, showing filling of the stope with waste prior to beginning shrinkage stoping. (After Eng. and Min. Jour.)

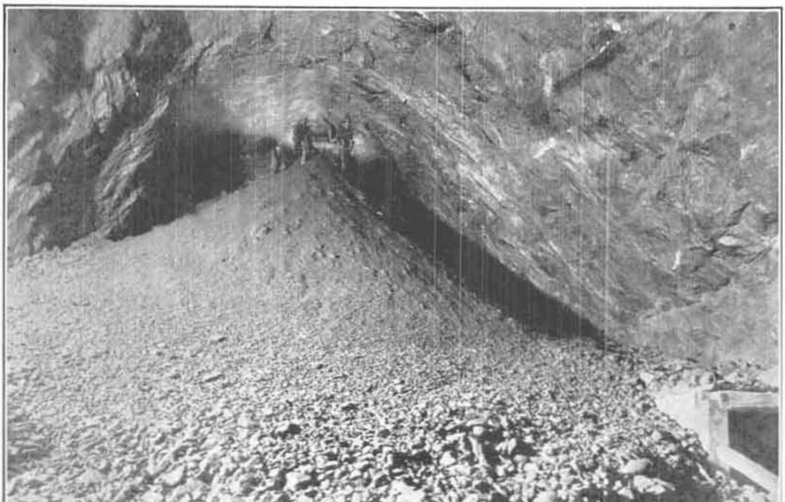

FIGURE 35.—Cut-and-fill stope in Lake Shore mine, Kirkland Lake, Ontario, showing scraper used for handling ore and waste

will be 100 feet. In this manner scrapers may be used to eliminate hand shoveling, speed up the extraction of the ore, and reduce the cost of installing chutes and waste passes.

By carrying the back of the stope at an angle from the horizontal approximately equal to the angle of repose of the filling material,

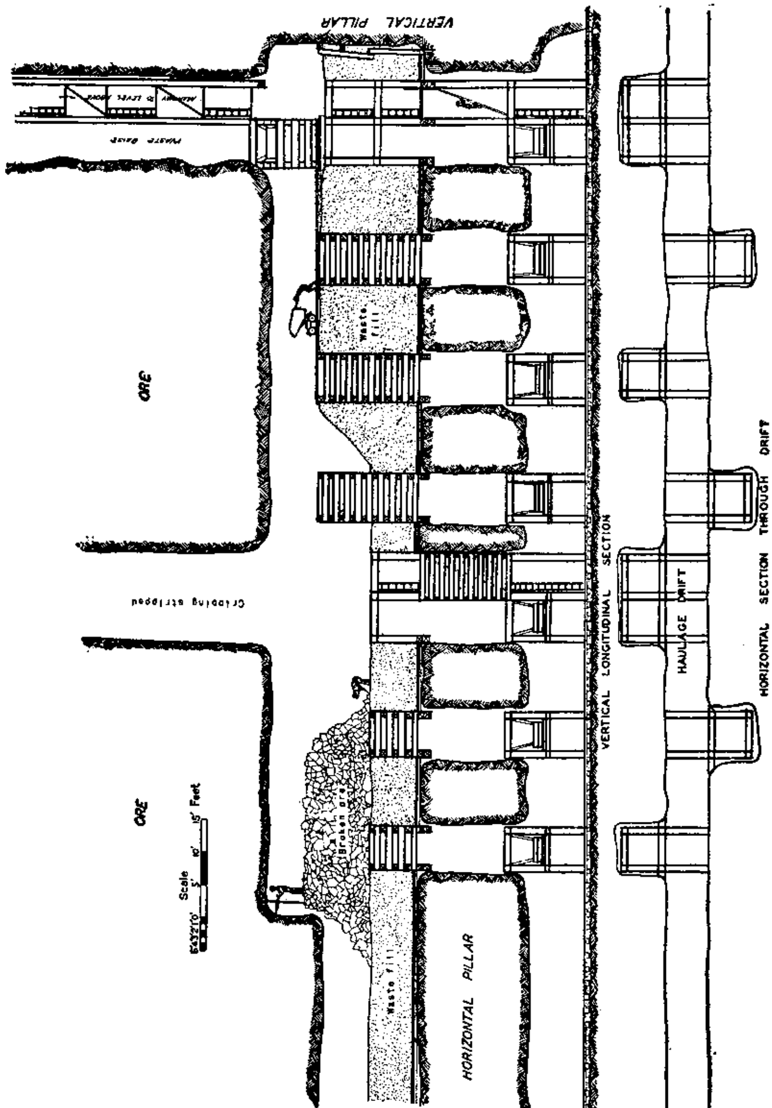

FIGURE 36.—Typical cut-and-fill stope

hand shoveling may be largely eliminated for handling both the ore and the waste (fig. 37), and this method has been employed extensively during the past 20 years. The waste runs into place by gravity from the waste pass, and the ore runs down into the chutes by gravity. The method is a variation of the cut-and-fill method and is commonly termed "inclined cut-and-fill" or "rill stoping." Recently, however, there has been a tendency to return to horizontal cut-and-fill stoping, especially since the advent of powerful, compact, and relatively light power scraping equipment. The reasons for this will appear from observations that follow.

In mining base-metal ores by cut-and-fill stoping, especially if the ores are of low grade, it is sometimes possible to break the ore from the back of the stope directly onto the filling material. With high-

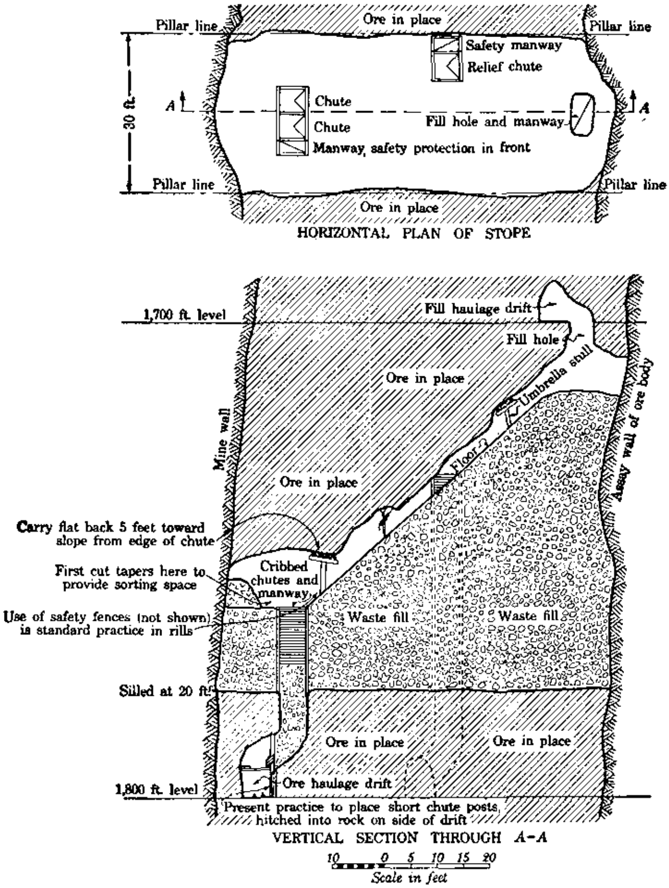

FIGURE 37.—Typical rill section. Inclined cut-and-fill stoping

grade base ores and most gold ores and with the average material used for filling this practice is apt to result in the loss of valuable fine-ore particles because of filtering down into the fill. Also, in shoveling the ore from off the fill, considerable filling material may be shoveled up with the ore. This is particularly apt to be the case where scrapers are employed for handling the ore, which is one disadvantage of scraping.

To keep the ore and waste separate, therefore, and prevent loss of fine ore or contamination of the ore with waste, it is the usual practice to lay a plank floor on top of the fill before the ore is broken down upon it. This practice obviously adds to the cost of labor and material in stoping. If rill stoping is employed the plank has to be handled up and down the sloping stope floor, which adds to the difficulties of the operation, and there is not much available space in which to store the plank in the stope during filling, so that usually it has to be hoisted or lowered from the stope and then, after the stope is filled, lowered or hoisted back to the stope again.

One of the chief advantages of cut-and-fill stoping as compared to shrinkage lies in the opportunity afforded for sorting out chunks of waste in the stope, for following stringers of ore back into the walls, and for attacking parallel ore shoots which may lie in the walls of the main lode, without contaminating the broken ore with waste. This advantage is considerably reduced in rill stoping, since chunks of waste will have to go down to the chutes with the ore, there being no level space upon which to throw them aside.

Another disadvantage of rill stoping is that the miners are always working from a steeply inclined floor where footing is insecure or can only be made secure by "staging up" with timbers; thus they can not work as efficiently as on a horizontal floor.

The cut-and-fill method of stoping is one requiring that a fixed cycle of operations be followed: (1) Drilling and breaking; (2) removal of broken ore from the stope, sometimes with sorting out of waste; and (3) placing of filling. This cycle is then repeated. Such a cycle demands that waste filling be available in the required amounts and at given times or else the stoping operations will be held up. This, in turn, may complicate the haulage problem, in that the tramming of ore and of waste may interfere with each other. Sometimes sufficient waste is not available from regular development work, and a special source of supply must be provided. This may be from surface glory holes or gravel pits, from underground waste stopes which may be mined by caving, or from the immediate walls of the stopes.

Sometimes a considerable volume of waste filling may be obtained by sorting waste from the broken ore and leaving it in the stope. Very narrow and high-grade veins are sometimes mined by "resuing" or "stripping," which consists of breaking the wall rock along the vein and then stripping off the ore. In this system the broken wall rock is left in the stope for filling.

In wide ore bodies transverse stopes with intervening pillars are sometimes employed. The stopes are worked out and filled, and then the pillars are mined out between the filled stopes. This usually requires the sides to be timbered and lagged over as the stopes are mined up, to prevent contamination of the ore with waste when the pillars are mined later. If the filling is somewhat plastic this may not be necessary, however.

As stated above, the principal advantage of cut-and-fill stoping, aside from the support afforded by the filling, is that an opportunity is supplied to do selective mining and to sort out waste in the stopes, which results in raising the grade of the ore sent to the mill, as well as the ability to follow offshoots of ore into the walls and to mine parallel ore bodies in the walls, thus completing the mining of a given section in the first operation.

Among the gold mines employing cut-and-fill stoping may be mentioned: The United Eastern (now worked out); and the Hollinger, McIntyre, and Lake Shore mines in Ontario, where this method has been introduced recently for mining the wider ore bodies, particularly on the lower levels.

SQUARE-SET STOPING

The square-set stoping method is employed for mining regular or irregular ore bodies of appreciable vertical dimension where the ore and walls are too weak to stand without support, except over very small spans and then only for a very limited length of time. On account of the high cost of the method its application is limited to the mining of relatively high-grade ores.

Although it is employed chiefly for stoping upward from the level, it may also be used for stoping laterally from a raise and sometimes for underhand stoping below a level.

In this method the ground is stoped in small rectangular blocks, usually about 5 by 6 feet square and 6 to 8 feet high. These blocks are termed "sets" in common parlance, the term "set," however, referring more properly to the timbers employed for supporting the excavation. As each block of ground is removed by stoping it is timbered by a set of framed timber which serves temporarily to support the surrounding ground. In ground requiring the use of the square-set method it is usually necessary to run in waste filling to form a permanent support shortly after the ore is removed. A complete set of timber consists of 4 posts, 2 caps, and 2 girts framed into each other. (Fig. 38.) Adjoining sets are framed into the initial set, so that for each additional set in line with the first one, 2 posts, 2 caps, and 1 girt or 2 posts, 2 girts, and 1 cap are required, the timbers of the initial set forming one side of the adjoining set. Where two sides are already formed by previous sets only 1 post, 1 cap, and 1 girt are required. Framing details vary with the nature and direction of the ground pressures and the preferences of the management. Two methods are shown at *A* of Figure 38.

As the stope is extended and carried up lines of sets are lagged off at convenient intervals and kept open to serve as ore passes. (See fig. 39, which shows vertical sections of a square-set stope at the Argonaut mine.) Chutes are installed on the level below for loading the broken ore into cars. Wing slides or diagonals may be installed to divert the broken ore from one line of sets to another and thus eliminate hand shoveling as much as possible. Other lines of sets must be kept open to serve as ladderways, to carry pipe lines, and through which to handle timber and supplies. These are often carried alongside chute lines for easy access to the chute for repairs and for opening up the chute if the ore hangs up.

Square-set stoping may be accommodated to ore bodies of irregular outline and to bodies containing irregular blocks of lean material or waste; it is well suited to selective mining. With this method blocks of waste can be left in place and the ore stoped from around them. Waste broken with the ore can be sorted out in the stopes and thrown into the gob, thus raising the grade of the ore sent to the mill and saving the cost of loading, tramming, hoisting, and treating worthless rock. By following the extraction of the ore closely with

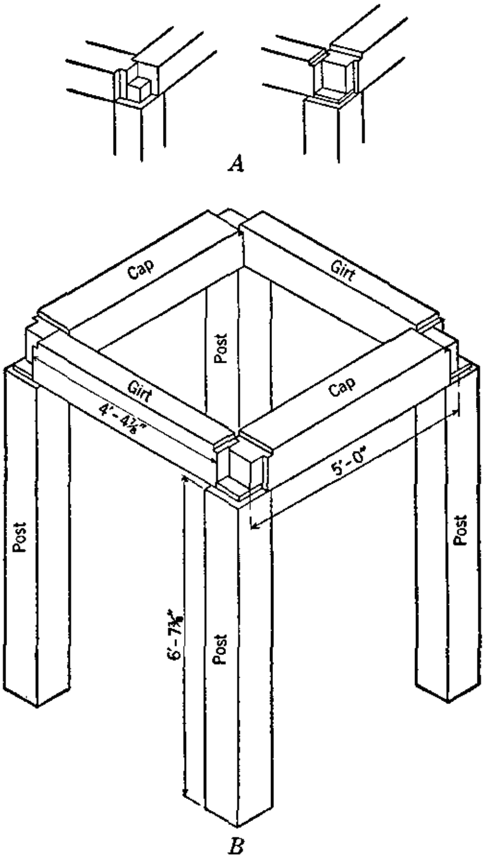

FIGURE 38.—Square-set timbering: *A*, Details of two methods of framing; *B*, complete 4-post square set

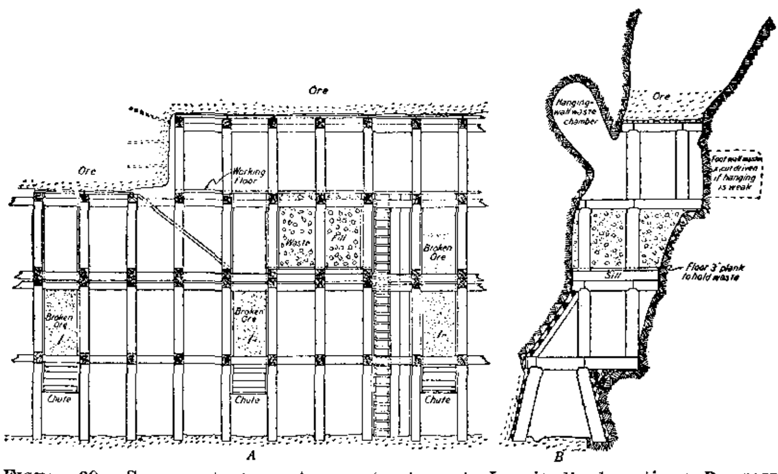

FIGURE 39.—Square-set stope, Argonaut mine: *A*, Longitudinal section; *B*, cross section

filling of the sets, only a small amount of ground need be kept open at a time. The disadvantages of the method are: (1) High cost per ton mined; (2) slow rate of ore extraction and low tonnage per man-shift; (3) large amount of timber required with attendant high cost of timber and of handling, framing, and placing timber; (4) based on reliable accident statistics, the highest accident-frequency rate of any of the common stoping methods; and (5) fire hazard due to the large amount of timber employed.

The method is used as the principal mining method at the gold mines of the Mother lode, California, and is said to have originated on the Comstock lode in Nevada. It was formerly the principal method in use at Cripple Creek, but here it has largely been abolished in favor of shrinkage stoping. At the Homestake mine, where the ore is mined by shrinkage in transverse stopes 60 feet wide separated by pillars 40 feet wide, the completed shrinkage stopes are drawn empty and filled with waste, and the pillars are then mined by square-

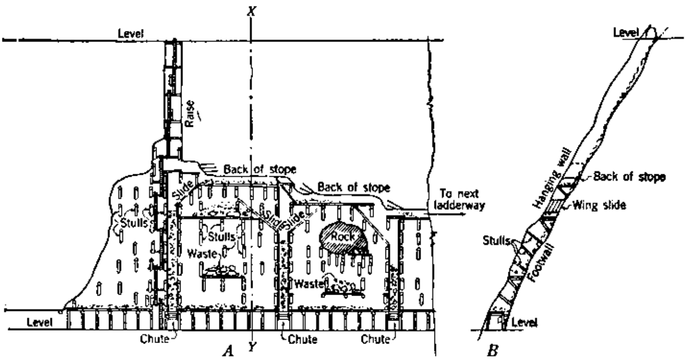

FIGURE 40.—Overhand open-stull stope: *A*, Longitudinal projection; *B*, cross section on line *X–Y*

set stoping. (Fig. 33.) At the Hollinger and McIntyre mines in the Porcupine district the method is used occasionally for mining wide stopes in badly broken, faulted, and heavy ground and for mining some of the floor pillars.

OPEN STOPES

Open stopes with temporary stull support are employed for mining narrow veins of firm ore with firm walls. In wide ore bodies of considerable vertical height the sublevel system of open stopes has been employed successfully in gold mines. Firm ore and wall rocks are essential in either case. Open stopes may be carried upward from the level, laterally from starting raises, or may be mined underhand from the level downward.

Figure 40 shows an overhand open-stulled stope in a narrow vein at a gold mine in Ontario. Depending upon the width of the vein and the nature of the ground the level may be timbered with regular drift sets or with stull timbers. The back of the drift is taken down to a height of about 12 to 15 feet and then timbered. Then, working on top of the timbers, the ore is removed by overhand stoping. Stulls are placed as required for support of the walls and for staging on which to work. As the stope rises, regular lines of stulls are set, and the sides are planked over to form ore passes. Temporary wing slides are put in as required to divert broken ore into the ore passes, thus avoiding hand shoveling as much as possible.

To avoid breaking the stulls and lagging, care must be taken not to put too much burden on the holes or to blast too heavily.

Slabs of waste inadvertently broken from the walls can often be left in the stope by planking over a few stulls as shown in the illustration, and areas of lean ore and waste may be left unbroken. Ladderways are maintained from level to level at intervals of about 100 feet, and air and water lines are carried therein.

Underhand open stopes may be employed in narrow veins having a steep dip. Raises are driven through from level to level; and, starting at the top, the ore is mined out by benching downward. Stulls are placed as needed for support of the hanging wall. This method is sometimes considerably cheaper than overhand stoping under favorable conditions. It is essential that the walls be very

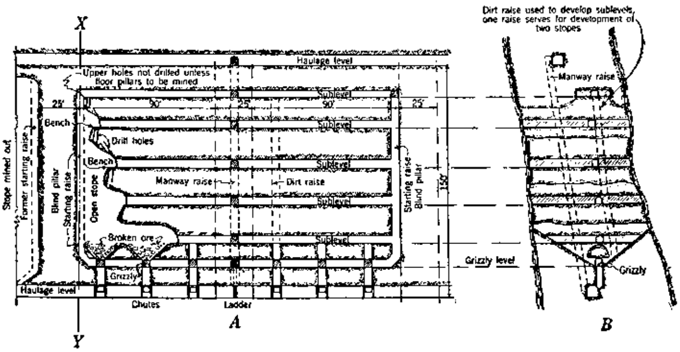

FIGURE 41.—Sublevel stoping in moderately firm and hard ore: *A*, Longitudinal vertical section; *B*, section *X–Y*, looking toward stope face

firm, however, since the miners are working some distance below the exposed hanging wall. In overhand stoping the miners are working close to the back of the stope and hanging wall so that the ground above them can be closely inspected and tested at all times, and any loose material can be taken down at once.

In wide ore bodies on a steep dip (55° to 60° or more) with firm ore and wall rocks, the sublevel stoping method may be used. It is applicable to the same conditions of ore and wall rocks as is shrinkage stoping but is not economical for mining ore bodies less than about 20 to 25 feet wide on account of the high cost of stope preparation (driving of sublevels) per ton of ore. For greater widths, however, the method is usually cheaper than shrinkage and is to be preferred from the standpoint of safety. This method, with its principal variations, has been described and discussed in some detail by Jackson.[6] In this circular the advantages and disadvantages of the method as compared to shrinkage and other stoping methods are discussed, and costs are given at a number of mines employing it.

Very briefly, the most common practice in applying this method is as follows. (See fig. 41.) The ore body is developed by haulage

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[6] Jackson, C. F., Mining Ore in Open Stopes, Eastern and Central United States: Inf. Circ. 6193, Bureau of Mines, 1929 (revised 1931), 36 pp.

levels at 125 to 200 foot vertical intervals, and raises are put through from level to level at intervals of 75 to 150 or 200 feet along the strike. One of these raises is at the end of the ore body or of a stope section and serves as a starting raise for the stope. These raises are connected by sublevels at an established sublevel interval, usually about 20 to 30 feet vertically. Meanwhile, short raises are put up 25 to 40 feet apart to serve as ore chutes or mill holes for drawing off the ore as broken in stoping. Sometimes the bottom sublevel is used as a grizzly level, as shown in Figure 41. The tops of the raises are belled out to form funnels.

Stoping begins at the starting raise and on the bottom sub (usually), the ore being blasted into the funnels by drilling up and down holes. Benches are cut around the face of the stope on the upper subs, and these are blasted into the funnels or mill holes by drilling up and down holes in the benches, the work for each sub being kept in advance of that on the sub above. Thus the face of the stope is in the form of a series of slightly overhanging benches, which are tied into the walls on either side. The miners are working at all times under solid, undisturbed ground and close to the back. Entrance to and retreat from the stope face are through the sublevel drifts connecting with manway raises in solid ground. The method therefore commends itself from the standpoint of safety.

Modifications of the standard method are: (1) Carrying the stope face vertical or even slightly sloping instead of overhanging in soft ore too weak to hold a bench; (2) mining in a series of transverse stopes separated by pillars to be extracted later, in very wide ore bodies (the above brief description applied to longitudinal stoping); and (3) increasing the sublevel interval and mining in several benches between subs by down holes only (underhand stoping), applicable to very firm ground.

The principal advantages of sublevel stoping as compared to shrinkage are: (1) When once developed rapid stoping is possible, with removal of ore as fast as broken if so desired, and with a high production rate from each stope; (2) lower breaking cost; (3) greater safety to the miners; (4) possible extraction of all the ore during the first working since the walls are exposed in the open stope; and (5) wider spacing of chutes or mill holes. As in shrinkage stoping, some broken ore may, however, be accumulated in the open stopes if it is desirable to maintain a reserve of broken ore.

Sublevel stoping has not been used extensively in gold mines, probably because the method was not widely known until recently and because the majority of gold-bearing veins have been too narrow. It was recently substituted for shrinkage stoping in the wider ore lenses at the Spring Hill mine in Montana.[7] (See fig. 7.)

It was used successfully at the Paymaster mine in Ontario until the ore became too low grade to work profitably.[8]

Open-stulled stopes are employed at the Empire-Star, Idaho-Maryland, and other mines in the Grass Valley (Calif.) district in both flat and steeply dipping veins. At the Empire mine the dip is in general flatter than the angle of repose for the broken ore, and

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[7] Pierce, A. L., Mining Methods and Costs at the Spring Hill Mine, Montana Mines Corporation, Helena, Mont.: Inf. Circ. 6402, Bureau of Mines, 1931, 11 pp.

[8] Hubbell, A. H., Pioneer Effort to Mine Low-Grade Ore in Ontario: Eng. and Min. Jour., vol. 126, Nov. 10 and Nov. 17, 1928, pp. 740–745, 785–788.

FIGURE 43.—Tom Reed mill and built-up tailings pond, Oatman, Ariz., October, 1931

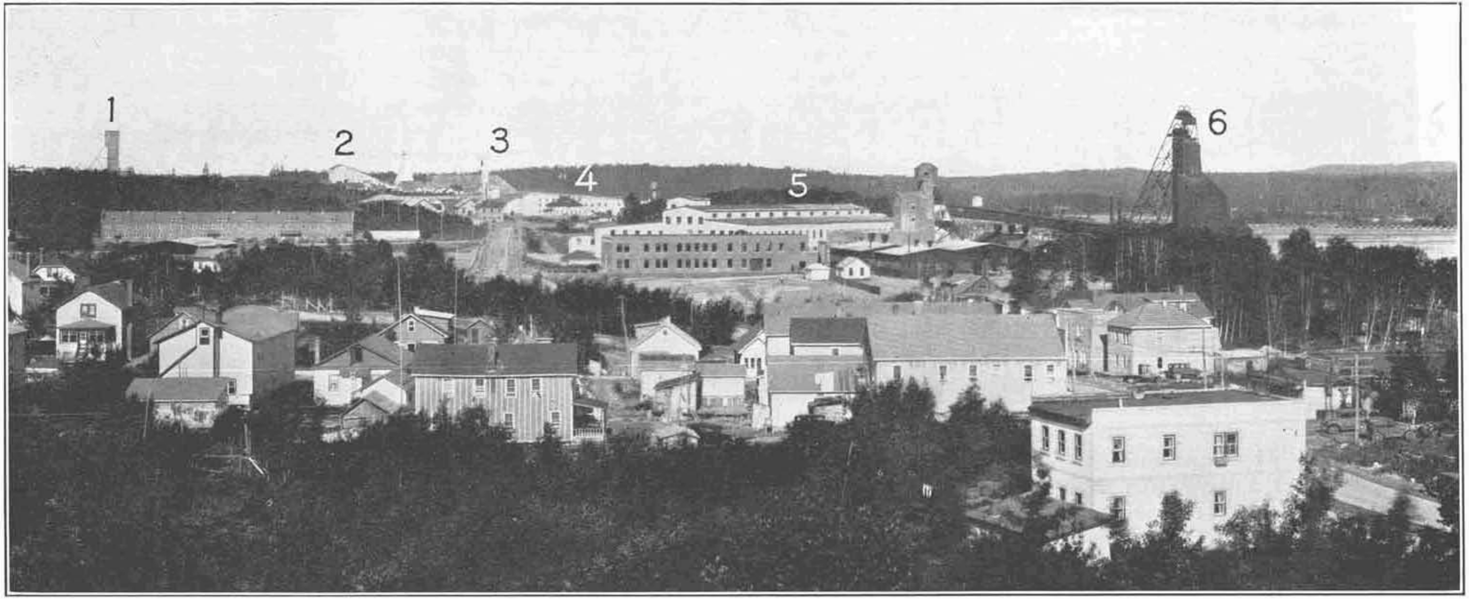

FIGURE 44.—Kirkland Lake, Ontario, showing Lake Shore mine plant in center, Teck-Hughes and Kirkland Lake Gold Mines shaft houses in background to the west, and tailings disposal in Kirkland Lake at extreme right: *1*, Teck-Hughes south shaft; *2*, Kirkland Lake Gold Mines (Ltd.); *3*, Teck-Hughes central shaft; *4*, Teck-Hughes mill; *5*, Lake Shore mill; *6*, No. 3 shaft, Lake Shore mine

the ore is transferred from the stope to the level by go-devils (two small cars, the upper ends of which are connected by a wire rope passing over a triple block equipped with a brake, the loaded car going down pulling the empty one up).

At the Sixteen to One mine in the Alleghany district, California, the same method of mining is employed in a flat vein, power scrapers being utilized instead of go-devils to transfer the broken ore to the levels.

For drift mining in buried placer deposits open-stulled stopes are employed. These deposits are flat, and the ore is shoveled directly into cars at the stope face. Bowlders are often abundant in the gravel; and these are thrown back or built into walls, in which event they may furnish some support to the back. Room-and-pillar or longwall-face mining is usually employed with temporary support in the form of props or stulls.

CAVING METHODS

So far as known, none of the typical sublevel or block caving methods of mining have been employed in gold mines of North America. At the Alaska-Juneau mine, however, a wholesale non-selective method of mining is followed which has sometimes been classified as a caving method.[9] The authors prefer to class this as a modified-shrinkage or forced-caving method. With this method the stopes are kept partly filled with broken ore; but, instead of drilling and blasting the back from the top of the broken ore, the ore is broken by large charges of explosive, averaging 4,000 pounds in weight, placed in powder drifts driven from a series of stope raises. (See fig. 42.)

At the Homestake mine a large tonnage of ore has been mined by caving the upper portions of the ore body, which had been mined by open stopes and square-set stopes in the early days, leaving much ore behind in pillars.[10] The caving is not a systematic operation here, but raises are driven up from lower levels to come under the old pillars. From the tops of these raises the ground is weakened by enlargement around the raise until the ground starts to cave. As it caves it breaks up, falls into the raises, and is drawn off below into cars. Usually a grizzly chamber is built at each raise a short distance above the level where the chunks are broken up and put through a grizzly before passing to the chute.

STOPING COSTS

Table 9 gives some typical stoping costs per ton at a number of gold mines in the United States and Canada. The wide variations in costs are due principally to the varying characteristics of the ore bodies, the nature of the ground, the size of the ore shoots and the interval between them, the grade of the ore, the method of stoping which depends on these charactertistics, and the rate of output. The location of the mines and costs of supplies and power, wage scale, etc., also affect the costs. These costs are taken from Bureau of Mines information circulars, annual reports of mining companies, the technical press, and special communications.

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[9] Bradley, P. R., Mining Methods and Costs, Alaska-Juneau Gold Mining Co., Juneau, Alaska: Inf. Circ. 6186, 1929, 18 pp.

[10] Jackson, Charles F., Shrinkage Stoping: Inf. Circ. 6293, Bureau of Mines, 1930, pp. 19–22.

Although the costs are given on the per ton of ore basis it is important to remember that a low cost per ton of ore may not give a low cost per ounce of gold. This may be due to a low grade of ore, which is beyond the control of the operator, or to the mining method employed. Thus a low cost per ton method may result in high dilution with waste due to sloughing of the walls, or the method may be one which does not permit the leaving in place of waste inclusions in the ore body. Again, a low-cost method may be wasteful in itself or, if carelessly practiced, may result in leaving considerable valuable ore behind. Thus the cost per ton is not in itself a measure of the efficiency of an operation but must be considered with the character of the deposit, the nature of the walls, the per cent of recovery of the total valuable ore in the deposit, and the effect of dilution, which increases costs of handling and of milling.

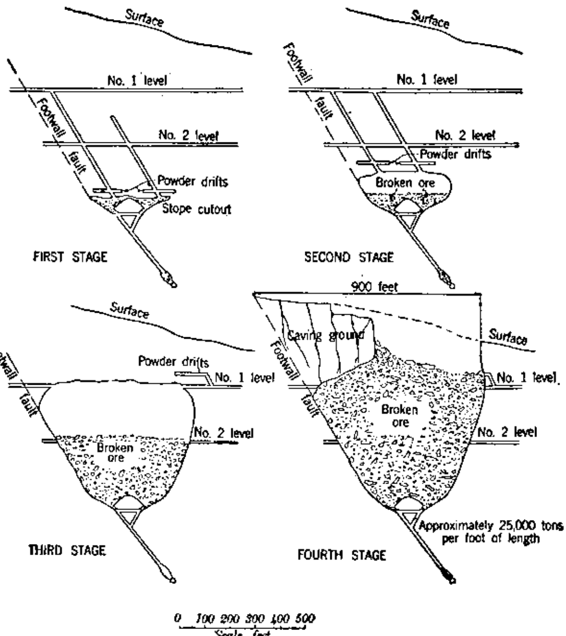

FIGURE 42.—Method of stoping, Alaska-Juneau mine

TABLE 9.—*Typical stoping costs at gold mines in the United States and Canada* [1]

Mine and period covered	Stoping method	Average daily tonnage	Cost per ton for stoping only
Alaska-Juneau, Alaska; 1928	Modified shrinkage or forced caving	[2] 10,000	$0.122
Homestake, South Dakota; 1929	Shrinkage, caving, and square-setting	4,100	1.143
Argonaut, California; 1 month, 1929	Square-set stoping	260	2.439
Elkoro, Nevada; 1930	Shrinkage	230	1.610
Spring Hill, Montana; Aug. 1, 1929, to Apr. 30, 1930	Sublevel stoping and shrinkage	175	1.072
Teck-Hughes, Ontario; 1 month, 1929	Shrinkage	900	1.710
Vipond, Ontario; year ended July 31, 1931	do	315	1.275
Sylvanite, Ontario; 3 months, 1930	Shrinkage and open-stulled stopes	250	2.318
Kirkland Lake Gold, Ontario; 1 month, 1930	Shrinkage	140	1.620

[1] Do not include haulage, hoisting, and surface costs. [2] Approximate.

Table 10 gives some costs per ton in units of labor (man-hours per ton), explosives, and timber for stoping at gold mines.

TABLE 10.—*Stoping costs per ton in units of labor, timber, and supplies*

Mine and period covered	Stoping method	Average daily tonnage	Stoping labor, man-hours per ton	Explosive consumption, pounds per ton for stoping	Timber per ton for stoping
Alaska-Juneau, Alaska; 1928	Modified shrinkage or forced caving.	10,000	0.086	0.340	
Homestake, South Dakota; 6 months, 1929.	Shrinkage, caving, and square-setting.	4,100	.675	.503	4.44 bd. ft.
Argonaut, California; 1 month, 1929.	Square-set stoping	260	2.401	-------	
Elkoro, Nevada; 1930	Shrinkage	150	1.373	1.239	0.852 lin. ft.; 1.355 bd. ft.
Spring Hill, Montana; Aug. 1, 1929, to Apr. 30, 1930.	Sublevel stoping and shrinkage.	175	1.081	.875	0.385 bd. ft.
Vipond, Ontario; year ended July 31, 1930.	Shrinkage	315	1.176	1.050	
Teck-Hughes, Ontario; 1 month, 1929.	do	900	1.517	2.100	3.87 bd. ft.
Lake Shore, Ontario; 1929	do	1,400	.668	.830	1.67 bd. ft.[1]; 0.59 lin. ft. lagging.
Sylvanite, Ontario; 1929	Shrinkage and open-stull stopes.	250	1.700	2.270	0.864.[1]
Kirkland Lake Gold, Ontario; 1 month, 1930.	Shrinkage	140	1.450	2.540	1.75 bd. ft.

[1] Includes development.

COST OF MINING LODE-GOLD ORES

As outlined in the preceding pages, a number of factors influence the cost of exploration, development, stoping, handling, and transport of ore and other items that comprise the total mining cost.

Since mining is pursued to make a profit, that balance between capital investment, operating cost (including milling), per cent extraction of total ore in the deposit, and rate of working out the deposit which will return the greatest profit from the enterprise is the aim sought. Upon these considerations the policy of management should be based. Generally speaking, a rapid rate of ore extraction will usually result in the greatest over-all profit. A high production rate can be used to write off the capital investment and interest charges thereon quickly and will result in low operating costs. It may result, however, in losses of ore and the jeopardizing of the mine by crushing and subsidence, with attendant high mining and maintenance costs, if carried to extremes.

MANAGEMENT POLICY

The policy of a given management may be to develop and mine out the deposit rapidly, maintaining the grade of the ore extracted at an approximate predetermined minimum, leaving material in the ground on which a doubtful or only small profit can be expected, and employing a plant capacity that will require a minimum capital outlay for the purpose of obtaining a maximum quick profit. In some instances this would be justified. In other instances the material of marginal value under a small output program may be in sufficient quantity that a larger plant and production rate, with attendant lower operating and overhead costs, would yield a good rate of profit and result in greater over-all profit for the project.

In determining the policy for a gold-mining enterprise the factors differ somewhat from those in the mining of base-metal ores. In the latter instance the factors are more variable. Thus, material which is of marginal value during the planning of the project may, with increasing prices for the metals, yield a good profit by the time ore extraction and treatment begin; or, conversely, what was originally estimated as ore (material which can be mined and treated at a profit) may, due to a fall in prices, become marginal material or definitely of too low grade to work. The grade of ore which it is profitable to mine may thus fluctuate from time to time during the life of the base-metal mine. Operating costs may either even off, to some extent, the peaks and valleys of these fluctuations or enhance them, depending on conditions. Due to direct reduction in costs of labor and material and increased efficiency of labor during periods of low prices the reduction in prices received for the mine product may be partly or entirely offset by reduced operating costs. On the other hand, increases in costs due to curtailment in production rate may, at such times, more than offset these savings and the fluctuations in the marginal grade of ore become more pronounced. Improvements in mining practices, ore dressing, and metallurgical methods make lower grades of ore profitable.

EFFECT OF ECONOMIC CONDITIONS

Gold mining, as previously pointed out, occupies a somewhat different and unique situation, since the final product—gold—has a fixed dollar value and is relatively stable, although its purchasing power varies. A notable exception was during and following the World War, when, due to increased prices of labor and commodities, the purchasing power of gold fell so rapidly that many gold mines were forced to close down or operate at little, if any, profit. The reverse situation exists at present, when depreciation of some currencies, as those of Great Britain and Canada, has caused the value of gold as measured by these currencies to be greater than normal, thus decreasing costs and increasing the amount of currency received for the product and increasing the profits.

The effect of an economic depression on the cost of gold mining is probably not as great as it is popularly supposed to be. During the present period (1930–31) wages at gold mines have not been reduced in the United States and Canada, except that, as pointed out above in the case of Canada, the value of the Canadian dollar has depreciated in terms of gold. It is true, however, that the efficiency of mine labor is increased, because labor is more plentiful. What this increased efficiency amounts to in per cent of total costs would be difficult to estimate. On the other hand, materials and supplies used in mining account for, roughly, only about 25 per cent of the total production cost—mining, milling, taxes, and overhead. Even a 25 per cent reduction in the average cost of materials and supplies, which would be large, would then only reduce the total production cost 6¼ per cent.

OTHER CONSIDERATIONS

Altruism and patriotism seldom enter into calculations in conducting a mining enterprise. However, it may be argued that the deliberate mining of gold ore of marginal value is warranted on these grounds, particularly during periods of economic depression, when the mining of such ore may supply employment and at the same time increase the amount of gold in the country, so adding its bit to the return of prosperity. This policy may, in the case of gold mines, be followed in some instances without any real loss, although there may be no direct profit therefrom.

The foregoing discussion is offered to show that costs per ton and per ounce of gold are affected not only by physical characteristics of the ore deposits, the grade of the ore, and the mining and milling methods and practices but also by the management policy and economic conditions.

OPERATING FACTORS

In considering the costs per ton of mining as presented in Table 11 it is obvious that so many factors are involved that a comparison of costs as between different mines is illogical. Furthermore, differences in accounting methods employed by the several mines make it impossible to set up the costs on an exactly comparable basis. The table merely indicates how and between what limits the mining costs may vary under different conditions of ore occurrence, rate of development, and production and management policy.

DEVELOPMENT COSTS

Especial note should be made of the development costs. It is obvious that the size of the ore bodies, their mode of occurrence, and the distance between the various ore shoots will have a marked effect upon development costs. It is also clear that temporary cessation of development would, in some of the mines, greatly decrease the mining cost for a time, though if continued it would result in depletion of the ore reserves and their ultimate exhaustion. On the other hand, a vigorous development campaign carried on at the expense of ore extraction may cause development charges to be abnormally high for a time. At some time in their history most gold mines face the necessity of such campaigns to build up ore reserves and prolong the life of the operations. An ideal balance between development and ore-extraction rate is seldom easy to achieve, but it should be the aim to strike a suitable balance between ore extraction, ore reserves, and the financial position of the company. If development is allowed to lag the company may suddenly find itself without ore reserves and with a depleted treasury at the same time. Assuming that there are good possibilities of finding further ore a company in this situation may nevertheless be unable to prosecute exploration. If, however, the ore possibilities are exhausted it is well to find this out far enough ahead of time to permit ultimate abandonment of the mine with the least possible loss.

The variations in development costs in Table 11 are thus due largely to the type of ore occurrence and size of the ore bodies but in some instances may be ascribed to the position of the mine as regards ore reserves during the period for which the costs are given.

STOPING COSTS

Stoping costs depend principally upon the size and dip of the ore bodies and the strength and hardness of the ore and wall rocks, which, in turn, determine the stoping method employed. Thus, where these conditions demand the use of square-set stoping, costs are bound to be high. On the Mother lode in California the immediate walls are very heavy and slough off upon exposure to the atmosphere, necessitating immediate support with heavy timbers following excavation of even a small volume of ore. In the quartz veins of this district, which exhibit ribbon structure, the central portion is often composed of friable quartz, with hard banded quartz on the sides and soft slate, much contorted and shattered, forming the immediate walls.

At the opposite extreme is the Alaska-Juneau type of deposit, where nonselective mining on a large scale by means of modified shrinkage and forced caving of large blocks permits low stoping costs.

Where it is necessary to sort out much waste in the stopes the cost is greatly increased. On the other hand, overbreaking the vein in mining, by which considerable wall rock is included with the ore, may lower the cost per ton mined, but it also lowers the grade of the ore and increases the cost per ounce of gold and thus the cost per dollar of product. Therefore, the mine having a high stoping cost per ton of ore may have a lower stoping cost per ounce of gold than one with a lower cost per ton of ore, even though the grade of the ore in the veins may be the same. In some of the veins in the Grass Valley district, California, the pay streak is narrow; but in breaking it down in the stopes barren or very low-grade material often breaks away to the hanging wall, the thickness of this material sometimes being several times that of the pay streak.

In thin, flat-lying ore bodies it is often necessary to employ scrapers or go-devils to move the ore from the stopes to the haulage levels. In some instances this may add considerably to the stoping costs, although either of these methods would be cheaper than repeated shoveling by hand.

In shrinkage stoping the rate of breaking may either exceed considerably or be much lower than the rate of drawing and hoisting, thus correspondingly increasing or decreasing the costs per ton of ore hoisted, for a time, as compared to the average costs.

The characteristics of the deposit are often such that there is little if any choice as to the stoping method to be employed. In other instances there may be a choice between two or more methods, and the selection of that best suited to the deposit is a matter of the utmost importance, requiring thorough knowledge of ground support and of all the factors involved. Once the general method has been selected the details thereof and the skill and efficiency with which it is applied will have an important bearing on the costs.

RATE OF PRODUCTION

The rate of production will influence the development and stoping costs per ton to a small extent and the haulage and hoisting costs to a somewhat greater extent, but will reflect principally upon the fixed overhead charges per ton. In Table 11 such costs as pumping, surveying, sampling and assaying, miscellaneous materials and supplies, and prorated office and supervision charges are included in the column headed General.

COST DATA

The data in Table 11 are not as voluminous as might be desired; but some companies, especially those in which the stock is closely held, do not publish their costs. The figures given, however, indicate the general range of mining costs per ton at producing lode-gold mines in the United States and Canada, including the lowest known costs of which any record is available.

The costs at the Big Indian mine[11] are out of line with those for the other mines listed, both as to date and mining method, but they are given as a matter of general interest.

With the exception of the data on the United Eastern mine the figures cover comparatively short periods, and may include some times of abnormal or of subnormal expenditures for exploration and mine development or those during which the rate of breaking and the rate of drawing were considerably different. The data for the United Eastern mine have exceptional interest in that they cover the entire life of the mine. These data have been taken from Moore,[12] who has given the costs at this property in much greater detail.

Table 12 gives some mining costs in units of labor, materials, and power.

TABLE 11.—*Underground costs per ton of ore at lode-gold mines*

Mine and period covered	Mining method	Direct mining cost per ton of ore milled				
		Development and exploration	Stoping	Haulage and hoisting	General	Total
Alaska-Juneau,[a] Alaska:						
1928	Modified shrinkage or forced caving.	$0.0611	$0.1219	$0.1136		$0.2966
1930	do					.2869
Homestake, South Dakota:						
1929	Shrinkage, caving, and square-setting.	.1299	1.1428	.3201	$0.1911	1.7839
1930	Square-setting, caving, some shrinkage.	.1808	1.4537	.4776	.2136	2.3257
Argonaut, California; 1 month, 1929.	Square-set stoping	.165	2.439	.519	.868	3.991
Elkoro, Nevada; 1930	Shrinkage and cut-and-fill	1.298	1.610	.416	.701	4.025
United Eastern, Arizona; January, 1917, to May, 1925.	Cut-and-fill	.518	3.073	.275	.537	4.403
Spring Hill, Montana; Aug. 1, 1929, to Aug. 1, 1930.	Sublevel stoping and shrinkage.	.377	1.072	.131	.365	1.945
Hollinger, Ontario; 1930	Shrinkage and cut-and-fill					2.983
McIntyre, Ontario; year ended Mar. 31, 1931.	do	[b].4239	1.9349	.8335	[c].0231	3.2154
Vipond, Ontario; year ended July 31, 1930.	Shrinkage	.728	1.275	.629	.148	2.78

[a] Figures for Alaska-Juneau based upon tons of raw ore. At this mine only about one-half the tonnage is actually milled, the balance being sorted out as waste before going to the mill proper.
[b] Total development and exploration cost $0.9146, including prorated transportation and pumping which in the table are included under Haulage and hoisting and General.
[c] Pumping.

[11] McIntosh, Colin, Big Indian Mine: Min. and Sci. Press, vol. 87, 1903, pp. 236–237.
[12] Moore, Roy W., Mining Methods and Records at the United Eastern Mine: Trans. Am. Inst. Min. and Met. Eng., vol. 76, 1928, pp. 56–92.

TABLE 11.—*Underground cost per ton of ore at lode-gold mines*—Continued

Mine and period covered	Mining method	Direct mining cost per ton of ore milled				
		Development and exploration	Stoping	Haulage and hoisting	General	Total
Lake Shore, Ontario; year ended June 30, 1931.	Shrinkage and cut-and-fill	[d] 1.403	[e] 2.448			3.851
Teck-Hughes, Ontario; year ended Aug. 31, 1931.	Shrinkage	[d] 1.18	[e] 2.42			3.60
Sylvanite, Ontario; October, November, December, 1930.	Shrinkage and open-stulled stopes.	[f] 1.229	2.318	.641	.397	[g] 4.585
Dome, Ontario; 9 months, 1929	Shrinkage	[d] 1.34	[e] 1.47			2.81
Wright-Hargreaves, Ontario; 1930	Shrinkage and open-stulled stopes.	[h] 1.072	1.926	1.023		4.021
Kirkland Lake Gold, Ontario; 1 month, 1929.	Shrinkage	2.87	1.62	1.61	.66	6.76
Big Indian, Montana; 1903	Opencut					.208

[d] Probably includes a prorated charge for haulage and hoisting.
[e] Includes haulage and hoisting and general.
[f] Plus $0.930 shaft sinking.
[g] $5.515, including shaft.
[h] After deducting $0.975 advance development (excess tons developed over tons mined).

TABLE 12.—*Underground costs in units of labor, material, and power*

Mine and period	Mining method	Costs in units of labor, materials, and power per ton					
		Man-hours			Materials and power		
		Development	Stoping	Total underground labor	Explosives, pounds	Timber, feet or board feet	Power, kw.-h.
Alaska-Juneau, Alaska; 1928	Modified shrinkage or forced caving.	[1] 0.030 [2] .048	[1] 0.065 [2] .086	[1] 0.159 [2] .266	0.400	0	1.61
Homestake, South Dakota; 6 months, 1929.	Shrinkage, caving, and square-setting.	.0489	.675	1.161	.753	[3] 4.87	10.00
Argonaut, California; 1 month, 1929.	Square-setting and filling	.296	2.401	4.168	1.010	[4] 1.08 [3] .705	33.08
Elkoro, Nevada; 1930	Shrinkage and some cut-and-fill.	1.179	1.373	3.299	2.229	[4] .99 [3] 1.69	10.92
Spring Hill, Montana; Aug. 1, 1929, to Apr. 30, 1930.	Sublevel stoping and shrinkage.	.250	1.081	1.904	1.262	[3] .385	10.43
Central Eureka, California; September, 1930.	Square-setting and filling			4.538	1.010	[3] 25.78	86.00
Kirkland Lake district, Ontario; mine No. 1, 1929.	Shrinkage	.151	1.312	2.480	1.750	[3] 3.52	19.56
Do., mine No. 2; 1929	Shrinkage and cut-and-fill	.845	.668	1.858	1.623	[3] 1.67 [4] .59	
Do., mine No. 3; 1929	Shrinkage and open-stulled stopes.			3.359	2.860	[4] 2.56	
Kirkland Lake Gold; June, 1930.	Shrinkage	2.60	1.45	6.83	4.720	[3] 4.23	84.15
Vipond, Ontario; year ended July 31, 1930.	do	.424	1.176	2.826	1.78	[3] .42 [4] .70	28.00
Porcupine district, Ontario, mine No. 2; 1929.	Shrinkage and cut-and-fill	.320	1.630	2.700	1.277	[4] 1.39	20.81

[1] Underground crew only.
[2] All labor charged to mining.
[3] Board feet.
[4] Linear feet.

Part 3.—GOLD MILLING

Comprehensive treatment of the subject of gold mining requires some discussion of the milling of the ores, since the mining and the milling operations are nearly always mutually interdependent, and at most properties efficient and economic operation of both mine and mill is essential to the success of the undertaking as a whole. Gold mining is, in one sense, unique among metal-mining enterprises in that a more or less refined product is usually produced at the mine.[1] Long shipments of ore or concentrates between successive steps in processing are the exception rather than the rule, and the entire cycle of operations is generally conducted under the same management. Some gold ores are still shipped to custom mills for treatment; but these usually come from small mines, leasing operations, or mines in the development stage. The heyday of the custom gold mill has practically passed. A small amount of gold is also produced from ores which are shipped direct to smelters, where they sometimes command a premium because of their beneficial character in the charge as siliceous fluxing agents.

As previously stated, the fact that a product of high value and small bulk can be produced at the mine, often by a relatively simple milling process, is largely responsible for the successful operation of gold mines in remote and inhospitable regions, where costs of transportation and fuel would be too great to permit exploitation of base-metal ores of equal dollar value.

Only the broader aspects of gold milling will be touched upon herein. The discussion will be as brief as clarity will permit and will deal chiefly with application of the principal methods and the factors that affect operations. Detailed treatment of the subject from the standpoint of the physics and chemistry involved, equipment used, and operating technique, as well as descriptions of individual operations, may be found in the references.

TYPES OF ORES

From the milling standpoint gold ores may be grouped as follows:

1. High-grade ores suitable for shipment to smelters or for treatment by direct refinery methods without intermediate concentration.
2. Free-milling or amalgamating ores in which most of the gold may be amalgamated with mercury, after suitable grinding and without roasting, leaching, or other auxiliary treatment.
3. Nonamalgamating ores, which, because of intimate association of the gold with pyrite, antimony, or arsenic, combination with tellurium, rusty oxide coatings, or the presence of graphite or greasy substances, must be subjected to treatment by other methods than amalgamation.
4. Ores that will yield part of their gold content by amalgamation but have to undergo additional treatment by other methods to give a satisfactory overall recovery. Most ores belong to this class, although the distinction between ores of this character and those of the preceding and following groups is more often based on economic rather than physical considerations.

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[1] This statement applies with equal force to the quicksilver industry, however.

5. Ores that contain small amounts of amalgamable gold, with most of the gold in other forms, carrying, in addition, secondary amounts of base metals which are worth the expense of recovery.

6. Highly siliceous ores, often of low grade, which are salable under certain conditions to lead or copper smelters for use as flux. They may command a premium for their fluxing value over and above the value of their contained gold.

7. Base-metal ores in which the gold has minor or secondary value. These contribute heavily to total production, but since they are milled in plants designed primarily to recover other metals there is no need to consider them further here.

8. Placer gold. Recovery of metal from this type of material has been touched upon in an earlier section. Amalgamation is generally resorted to for treating placer material, the gold having been concentrated by natural agencies.

BRIEF OUTLINE OF HISTORY AND PRINCIPLES OF GOLD MILLING

Many processes for the reduction of gold from its ores have been conceived, but few have achieved commercial success. The property possessed by mercury of readily amalgamating with gold has long been known and was utilized by the ancients in recovering gold from placers and lode ores. Until the latter part of the nineteenth century amalgamation, chlorination, and direct smelting were the usual practices; but panning or sluicing, followed by laborious picking of the nuggets and colors from the heavy concentrates by hand has always been employed to some extent in crude operations.

Various processes have been utilized for the extraction of gold by converting it to the chloride. Of these, the Plattner process, now nearly obsolete, had quite a vogue in the eighties and nineties in California, where it was introduced in 1857, and at the Alaska-Treadwell mine. The ore was roasted, charged to vats, and treated with chlorine gas or a solution of chlorine in water. The gold chloride formed was then washed into other vats, and the solution was treated with ferrous sulphate to precipitate the gold. The barrel process, similar in principle to the Plattner method, involved treatment in revolving cylinders. It was popular before the nineties in Australia, Colorado, and South Dakota, but gave way to the cyanide process early in the nineties.

The volatilization process involved roasting of the ore with salt (sodium chloride), the gold and silver being volatilized and recovered in scrubbing towers or burlap filters sprayed with water or in bag houses. It was tried in Arizona in 1906 but was soon abandoned because of difficulties in recovering the fume. Later experimental work by the Bureau of Mines [2] has shown that gold can be rather completely volatilized and recovered in Cottrell fume treaters. Tests of Black Hills gold ores and complex ore from Nevada gave encouraging results.

The cyanide process is based on the work of MacArthur and the Forrests, begun in 1886. It involves leaching of gold and silver from alkaline pulps with dilute solutions of sodium or potassium cyanide. The gold is dissolved as the double cyanide of sodium or potassium and is precipitated from the clear pregnant solution by means of zinc dust or shavings. Increased knowledge of the chemistry of the

[2] Varley, Thomas, Barrett, E. P., Stevenson, C. C., and Bradford, Robert H., The Chloride Volatilization Process of Ore Treatment: Bull. 211, Bureau of Mines, 1923, 99 pp.

process, supplemented by great advances in the physical and mechanical methods of treatment, have so greatly enhanced its utility that cyanidation is now widely employed for extracting gold from its ores. In fact, cyanidation and amalgamation are now recognized as the wet methods of greatest importance and with smelting account for practically all the bullion recovered from lode ores.

Concentration by gravity methods or, more recently, by flotation, is often done before the material is subjected to final smelting or cyanide treatment to recover the gold as bullion.

FACTORS AFFECTING CHOICE AND OPERATION OF MILLING METHODS

In devising a flow sheet for the treatment of a particular ore many factors must be considered and evaluated in their proper proportions. Outstanding among these are the character of the ore, the probable daily tonnage to be handled, the estimated life of the enterprise, the amount of capital which can be counted on, and the available supply of water. Other factors which demand consideration and may or may not be of major importance, depending on local conditions, are: Geographic location, altitude, and climate; transportation facilities and costs; cost of supplies; availability, kind, and cost of fuel and power; abundance, efficiency, and honesty of labor; political conditions; Government regulations, and commercial affiliations; cost of construction; royalties on the use of certain processes or equipment; and general economic conditions.

The various factors which influence the choice and operation of milling methods are discussed below. Their relative importance in any specific instance must, of course, be determined in the light of whatever combination of conditions prevails at the property in question.

CHARACTER OF ORE

The character of the ore must be investigated with regard to its grade or richness, the uniformity of its tenor, the coarseness of the gold, and its mineral associations.

GRADE

The grade of ore has a marked influence on the design of milling plants. Ores of very low value per ton can usually be successfully exploited only when treated on a large scale in plants of large capacity. The Alaska-Juneau project is the classic example of successful milling of extremely low-grade ores.[3] Heavy capital expenditures are necessary for the construction and operation of large plants. Higher-grade ores require smaller plants and less initial capital per dollar of output, except where the ore is of a refractory nature and a complex, expensive treatment is unavoidable. Extremely rich material, such as the "specimen" gold mined from pockets or bonanzas in some veins, is usually treated in the mill refinery or shipped to smelters.[4] In general, the higher the grade of

[3] Bradley, P. R., Milling Practice at the Alaska-Juneau Concentrator: Inf. Circ. 6236, Bureau of Mines, 1930, 16 pp.

[4] Young, G. J., Gold-Ore Mining and Milling: Eng. and Min. Jour., vol. 132, Sept. 14, 1931, pp. 195–199.

ore treated the wider the margin of profit per ton for mills of approximately equal metallurgical efficiency handling ores of otherwise similar characteristics.

Conversely, then, with higher-grade ore, milling may show a profit even though simpler, cheaper methods are used which yield less complete extraction. This fact has been, and in some instances still is, responsible for the loss of much gold in the tailings. In former years, such losses were often unavoidable because of the imperfections in the art of gold milling. At present, however, gold metallurgy is far more efficient than it was 25 or 50 years ago, and poor recoveries in the treatment of high-grade ore may generally be ascribed to a lack of funds sufficient to install an adequate and proper plant or to an attitude of indifference to the high tailings assays as long as good dividends are being paid. In many such instances the investment in an additional plant to recover part of the gold lost to the tailings would yield a good return.

UNIFORMITY OF TENOR

Uniformity in the grade of ore is important. Mills treating ores of reasonably constant characteristics are the simplest to design and operate and give the most efficient results. Erratic assays in the heads usually result in erratic mill performance. Sudden increases in value may cause high tailings losses before changes in the mill routine can be made to offset them. In cyanide plants the whole procedure is thrown out of gear by such fluctuations. Higher-grade ores containing coarse gold require longer periods of agitation. Amalgamation, where applicable, is best suited to cope with variable grades of the ore. Large mines are able to control the grade of ore milled to some extent, or at least to prevent sharp fluctuations from day to day by proportioning the amounts stoped from different parts of the mine, and gradual changes in the nature of the ore mined can be paralleled by gradual modification of milling practice. At small properties, however, these fluctuations may be serious factors in the mill operation or management policy. In one rather extreme instance—the Alleghany district of California—the values are so erratic within the vein that the company sometimes operates at a loss for several months between periods when good profits are made.[5] Under such conditions ample financial reserves to provide for continuous operation during periods when the grade of ore is low are essential to over-all profitable operation.

SIZE OF PARTICLES

The size of the gold particles in the ore often greatly influences the methods of treatment used. If coarse free gold is present the method must be adapted to recover the gold in its coarse condition, since grinding will not greatly reduce such a soft and malleable metal. In the best practice, amalgamation plates or stationary tables with blankets or riffles are frequently employed to catch coarse gold; if the material is cyanided without prior amalgamation longer leaching periods (in sand leaching) or a greater amount of agitation (in slime treatment) are necessary for complete solution of the gold. In

[5] Young, G. J., work cited.

either case, the result is decreased daily capacity or increased plant installation, with higher operating costs. Prolonged grinding may offset these effects somewhat by flattening the gold particles and thus increasing the area exposed to solvent action. In some instances grinding in cyanide solution and passing the pulp over amalgamation plates before the usual cyanide treatment period have proved beneficial, but care must be used to keep the plates well covered with mercury to prevent the cyanide from attacking the copper in them. On the other hand, many operators have found that the use of amalgamation ahead of cyanidation precludes the possibility of grinding in cyanide solution, which is a disadvantage, in that a longer leaching period is then required or more agitation capacity must be provided. In general, where appreciable amounts of coarse free gold are present in the ore it is good practice to amalgamate this gold before further treatment by other methods.

If the gold occurs in very small particles the ore must be finely ground to unlock the values. The limits to which fine grinding should proceed are governed by a comparison betwen the additional recovery and the increased cost. For cyanidation, it is often necessary to expose only part of the gold, since solutions attacking the exposed portions will dissolve their way into the inclosing gangue. Thus, for ores in which the fine gold is inclosed in quartz and must be largely freed by grinding before it is recoverable on amalgamating plates or by flotation, cyaniding may give a satisfactory extraction from less finely ground middlings and in such cases may prove to be the best method from the standpoint of grinding costs alone. If, however, the fine gold is closely intergrown with sulphides flotation treatment may be desirable for yielding a good recovery with much lower grinding expense than direct cyanidation. If the gold is mostly in a finely divided state in the ore, so that sliming (very fine grinding) is necessary to liberate it, cyanidation is doubtless the best method of treatment, provided the ore does not carry substances which inhibit dissolution of the gold or cause fouling of solutions and destruction of cyanide. In some ores of this kind, however, the finely divided gold is largely contained in sulphides which can be concentrated at relatively coarse sizes by flotation, thus permitting the discard of a large part of the gangue before finer grinding and further treatment of the concentrate by cyanidation are undertaken.

MINERAL ASSOCIATIONS

The nature of mineral associations in gold ores is perhaps the most important factor in the selection of treatment methods. Certain substances which are sometimes present may, by causing poor extraction or high costs, rule out of consideration a treatment process which would otherwise be attractive or compel the use of certain expensive steps which ordinarily are not necessary. The effects of the most common minerals that have a modifying influence on methods or on results are discussed below.

PYRITE

Pyrite is generally present in unoxidized gold ores and nearly always contains some gold values in intimate mechanical mixture.

Complete liberation of the gold from this mineral would nearly always entail very fine grinding and consequent high costs, and even then the gold particles would sometimes retain a surface film of sulphide sufficient to prevent amalgamation of all the gold. It is a safe generalization that ores containing much vein pyrite will not yield a satisfactory percentage of their gold by amalgamation alone but must be treated further by gravity concentration or flotation or by cyanidation. In many ores the sulphides carry so great a proportion of the total gold that the best all-round results are obtained by straight cyanidation or, in some recent plants, by flotation. Where flotation is employed it may be followed by cyanidation of the concentrates, of the tailings, or of both, or by smelting of the concentrates.

OTHER SULPHIDES

Other sulphides of base metals may have the same effects as pyrite, inasmuch as gold may be locked up in them in an intimate mechanical association. They all exhibit a slight tendency to foul mercury, but in most gold ores the sulphides are not sufficiently abundant to cause much trouble of this kind, except, in some instances, those of antimony, arsenic, and to some extent bismuth. Base-metal sulphides may be partly oxidized before being mined or may undergo more or less oxidation while lying broken in stopes and bins. As a result, the mill feed may contain enough partly oxidized material to seriously interfere with operations; that is, in addition to the acids liberated by the breakdown of the original sulphides, the ore will contain soluble sulphides, as well as ferrous and other soluble sulphates which are deleterious to cyanide solutions. The mineral pyrrhotite (Fe_nS_{n+1}) is very slowly dissolved by cyanide solutions, with resultant consumption of cyanide, and is also injurious owing to its power of absorbing oxygen. Sulphide minerals of the base metals are not often abundant enough in gold ores to be worth saving. Some sulphide ores which carry rather high values in gold, as at Rouyn, Quebec, are mined and milled principally for their content of copper or other base metals and in normal times yield gold as a by-product of secondary value. In this bulletin these ores are not classed as gold ores and will not be considered further. Occasionally, however, the presence of base metals in the ore has modified the method of treatment simply because these metals, although small in amount, are worth saving as a source of some additional revenue. Arsenic was recovered from gold ores at the Nickel Plate and at other mines. At the Alaska-Juneau mill part of the gold is associated with lead in the concentrates; here, the lead is present in the ore as mined to the extent of only a fraction of 1 per cent but is deemed worth saving in a concentrate, which is shipped for smelting rather than treated locally to produce bullion with loss of the lead. At the Spanish mine copper, lead, zinc, iron sulphide, and barite are all recovered as separate products; the gold is saved chiefly in the copper, but in the other sulphide concentrates as well and is recovered in smelting.[6]

[6] Bradley, James, Mining and Milling at the Spanish Mine: Min. and Met., vol. 12, October, 1931, pp. 435–439.

ARSENIC AND ANTIMONY

Arsenic and antimony cause serious trouble if present in appreciable amounts. Many sulpharsenides and sulphantimonides are known to occur in precious-metal ores, especially ores that are high in silver, but stibnite (Sb_2S_3) and arsenopyrite (FeAsS) are the commonest minerals of these elements. They induce "sickening" of the mercury in amalgamation, chiefly by coating it with a black film which causes the quicksilver to separate into small globules and prevents the amalgamation of gold and silver. Partly decomposed arsenic and antimony minerals, or the oxides realgar and orpiment, are particularly bad offenders in this way. Arsenic and antimony in some forms are soluble in cyanide solutions and cause high consumption of the reagent; they also tend to deoxidize and foul the solutions to such an extent that gold and silver can not be dissolved and may form reducing compounds which precipitate the gold from solution.

BISMUTH

Bismuth sulphide sickens mercury somewhat,[7] but usually is not sufficiently abundant to be a source of trouble.

TELLURIDES

Tellurides, important gold and silver minerals in a few districts, will not amalgamate directly with quicksilver and cause sickening of the mercury. Sodium amalgam will amalgamate them, however. They are not directly amenable to ordinary cyanidation but are dissolved by solutions of bromocyanide. Very fine grinding, with abundant aeration, sometimes promotes the breakdown of tellurides.

OXIDIZED MINERALS

Certain metallic oxides, carbonates, hydrates, sulphates, and arsenates, as well as soluble sulphides, form a group known collectively as "cyanicides." These substances react with cyanide, forming double cyanides which only very slowly dissolve gold and silver, or they may decompose the cyanide and thus destroy its value. Soluble sulphides are a further source of trouble, in that they tend to reprecipitate values from solution. Soluble copper salts are perhaps the most troublesome cyanicides, for in addition to reacting with cyanide they deoxidize the solutions. Along with ferrous sulphide, sulphate, and other salts, their effects can not be counteracted by the addition of lime to the pulp. Copper salts can be partly removed by a preliminary acid or ammonia wash. The cyanide used in the dissolution of copper may be mostly (80 per cent) regenerated by the acid-sulphide precipitation method.[8] Soluble sulphides may be rendered more or less innocuous by charging a little lead acetate or lead oxide into the solutions. In flotation partly oxidized sulphides are sometimes difficult to recover and are usually floated by means of special reagents, such as sodium sulphide, which plate the particles with a

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[7] Rose, T. K., The Metallurgy of Gold: C. Griffin & Co. (Ltd.), London, 1915, 601 pp.

[8] Leaver, Edmund S., and Woolf, Jesse A., Copper and Zinc in Cyanidation; Sulphide-Acid Precipitation; Tech. Paper 494, Bureau of Mines, 1931, 66 pp.

sulphide film. Oxidation of some minerals may affect amalgamation recovery by causing a thin film of foreign matter to adhere to the gold, rendering it "rusty" and difficult to catch on the plates. Manganese oxides tend to sicken mercury.

CARBONACEOUS MATTER

Carbonaceous matter in an ore will cause increased consumption of cyanide and reprecipitation of gold from solution.[9, 10] Certain ores, notably some of those mined along the California Mother lode, have in the past yielded low recoveries because the carbonaceous slimes which contain an appreciable proportion of the gold could not be cyanided. These slime portions are amenable to flotation.[11] Carbonaceous materials may also, in some cases, sicken the mercury on amalgamation plates.

GREASE AND TALC

Many greasy, colloidal, or talcose substances are present in the ore fed to gold mills and in noticeable quantity may have serious effects. Oil and grease from the mine or crushing machinery are very deleterious to amalgamation, causing sickening. Talcose and clayey ores may have much the same effect,[12] and talc or other hydrous silicate minerals of magnesium and aluminum may cause a slimy froth to coat part of the gold and render it unrecoverable on the plates. Colloidal, greasy, or clayey minerals like kaolin give trouble in flotation by diluting the concentrates, since usually they are not depressed or removed by preliminary treatment without high tailings losses or excessive cost, although starch material is now sometimes used with success to depress talc and kaolin. Colloidal and clayey materials are a common source of trouble in filtration, and they increase the difficulties of gravity concentration or cyanidation by fine sliming.

STRONG ALKALIES OR ACIDS

If strong alkalies or acids are present in the ore or mill water they often act on the copper of the amalgamating plates, causing discoloration and subsequent scouring by the formation of copper oxides, carbonates, or sulphates and sometimes mercury compounds. Such conditions in the water usually increase the cost for reagents in cyanidation or flotation. Calcium and magnesium bicarbonates in the water at the Homestake have prevented the otherwise desirable use of an alkaline milling circuit.

INFLUENCE OF TONNAGE TO BE HANDLED

Two interrelated and very important factors that should be given studied consideration in devising a milling method are the probable life of the enterprise and the most economic daily tonnage that should be handled. Low operating costs per ton of ore usually

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[9] Peele, Robert, Mining Engineer's Handbook: 2d ed., John Wiley & Sons, 1927, p. 1933.

[10] Leaver, E. S., and Woolf, J. A., Re-Treatment of Mother Lode Carbonaceous Slime Tails: Rept. of Investigations 2998, Bureau of Mines, 1930, 6 pp.

[11] See footnote 10.

[12] MacFarren, H. W., Practical Stamp Milling and Amalgamation: 3d ed., Min. and Sci. Press, 1914, 225 pp.

(although not invariably) depend on a fairly large scale of operations. On the other hand, large plants involve the outlay of considerable capital and require an assured supply of ore for a length of time sufficient to retire the investment with interest plus a reasonable profit. Thus it is never safe to build and equip a large mill until several years' supply of ore has been proved in the mine and sufficient knowledge of the characteristics of the local ore occurrence has been gained to indicate that the deposit will persist far enough beyond the known reserves to supply additional future ore. Where only small tonnages have been blocked out but where the mine conditions are promising a modest mill is often justified, especially when the ore is rich enough to return some profit from small-tonnage milling operations and thus assist in paying for further mine development. A plant of this kind also has great value as a pilot mill. By providing facilities for large-scale testing under actual operating conditions, it assists in correctly designing the larger permanent plant and minimizes the chances of costly mistakes and necessary alterations later. An interesting example of an undertaking of this kind in the Porcupine district of Ontario has been described recently.[13]

In selecting the tonnage capacity of the initial mill at a gold property it is wise to err on the side of conservatism rather than to proceed at once with over-ambitious plans. The ore needs of the mill should be kept well within the capacity of the mine to supply without forcing production at the expense of development and thus depleting the reserves. Expansion can be undertaken when the mine has been developed to a point which justifies the additional investment in plant. By that time, moreover, considerable experience will have been gained as to the milling characteristics of the ore, and the new unit can be designed to better advantage.

It is often possible to devise a mill flow sheet in which the ore is treated in one or more similar and independent units arranged side by side or "in parallel." By proper construction of the mill buildings with this consideration in mind expansion of capacity involves simply the installation of additional units alongside those already in use. This system possesses several outstanding advantages, chief of which are simplicity in layout, standardization of equipment and operations, and the ability to shut down one unit for necessary repairs without complete mill shutdown. Furthermore, if for any reason it becomes necessary or desirable to curtail output one or more units can be stopped and the rest run at full efficiency.

The tonnage to be handled and the probable life of the mine may influence the milling practice in other ways than those just discussed. For example, a certain ore may be amenable to direct cyanidation at low cost and with good extraction if milled in tonnages great enough to justify the plant investment. The reserves, however, may be small and the financial resources of the management meager. In such cases an immediate return on the capital already put into the enterprise is often imperative to avert failure and loss, and it may thus be good business to install a small stamp mill to treat the ore by amalgamation and table concentration, shipping the concentrates

[13] Vary, Ronald A., Amalagamation Practice at Porcupine United Gold Mines (Ltd.), Timmins, Ontario: Inf. Circ. 6433, Bureau of Mines, 1931, 5 pp.

to a custom cyanide plant or smelter. Even though a poor recovery is made the mill may show a profit if the ore is of good grade, and a shutdown and possible financial failure are avoided. Thus, where large sustained tonnages of ore are not available a small amalgamating or concentrating plant, even if metallurgically inefficient, may be the only sound method of meeting the situation.

AMOUNT OF MONEY AVAILABLE

The importance of the amount of money available has been discussed in part in the preceding paragraphs. It has a vital bearing on the selection of a milling method where funds available are so limited that only the simplest and least expensive installations can be considered. When larger sums are at hand the amount of capital that must be invested becomes secondary in importance to other factors that bear more directly on the cost of producing gold or concentrates in the mill and thus largely determine the ultimate profit.

WATER SUPPLY

Water supply is of vital importance in milling operations. All commercially successful methods of treating gold ores require more or less water, in amounts which commonly range from 1 ton to 5 or 6 tons of water per ton of ore treated. In desert regions the cost of bringing water to the mill site may spell the difference between profit and loss. The purity of the water is important in cyanidation and flotation. If it is excessively acid, as are some mine waters, the sequel may be a high cost for lime to render it slightly alkaline; or if it contains certain soluble salts, these may have to be removed or rendered innocuous by chemical treatment at some additional expense. Certain impurities in water may cause discoloration of amalgamating plates or fouling of the mercury. Usually, however, the character of the mill water does not greatly affect amalgamation and rarely has any influence on the results of gravity concentration. In planning a mill, therefore, the water supply should be carefully studied. The cost of ditches, flumes, pipe lines, or wells, pumping, purification, or chemical treatment, and the possibilities of recovering water from the tailings for reuse are aspects of the question which should be scrutinized and interpreted in the light of their effect on costs. In northern climates heating may be necessary, which increases the difficulties of the problem.

COMMERCIAL AFFILIATIONS

Commercial affiliations may occasionally modify the milling practice adopted. Large companies operating their own smelters often need siliceous flux for balancing furnace charges. Thus, gold ores may command a premium in regions where low-silica base-metal ores are smelted, and may be salable at a fair profit, despite the fact that they are too low in gold value to be successfully milled for the gold alone. Enterprises having favorable smelting connections may find it to their advantage to smelt their concentrates, where independent operators would realize a greater return by subjecting the concentrates to cyanide treatment. Thus, one mine may adopt certain methods or steps in the treatment of its ores which would be uneconomical for another management that has advantages through affiliation with transportation, power, or smelting companies, even though the physical conditions at the mine and the nature of the ore were closely similar in the two instances. Like all the other factors involved in selecting a milling method, this one boils down to a comparison of costs.

ROYALTIES

The use of certain processes, equipment, or reagents is sometimes controlled by patents. It occasionally happens that a process or part of a process may be adopted, or equipment or reagents used, which are less efficient technically than others covered by active patents, simply because the over-all results obtained with the less efficient arrangement yield a better profit than could be attained by the technically superior method with its contingent royalties. This factor presents a constantly changing picture as old patents expire and new ones are issued and must be considered in each specific case.

LOCATION

The location of a gold-mining property may have a marked influence on the choice of a milling method and on the degree of success attending its operation. It has already been pointed out in preceding pages that gold mining is affected less adversely because of situation in remote or difficultly accessible regions than usually is true of base-metal exploitations; nevertheless, a great variety of operating conditions are more or less definitely modified by the location of the deposit (either local or geographic), and together they may result in such high costs that profitable operation is impossible. Some of the conditions which may be regarded as dependent on location are discussed below.

ALTITUDE AND CLIMATE

Rigorous climatic conditions are the sequel of great altitudes or location in high latitudes. Long periods of freezing weather must be contended with and are reflected on the cost sheet by the expense of heating the mill and dwellings and sometimes of heating mill water and solutions. Pipe lines must be rather deeply covered and structures built to withstand heavy snow loads. Roads around the plant must be kept open. Frequent interruptions in power and transportation service are probable. If the climate is very dry water may be scarce and costly or of poor quality and evaporation losses great. In tropical climates labor efficiency is low, and added costs are likely to be incurred in combating malaria and other diseases. Usually climate has no direct effect on the relative technical applicability of different milling methods (except as it may determine water supply); but it may, and often does, modify labor conditions and the like. Labor is often scant in the far north or lazy and inefficient in tropical countries; in any region of extreme climate staff men and skilled white labor command high pay and are difficult to keep on the job. This question is one that has troubled many managers in remote districts.

TRANSPORTATION

Transportation into the district is important to the gold-mill operator chiefly when it is slow, unreliable, and excessively expensive. Its importance is a function of location. It may affect the choice of a flow sheet to some extent, in that cyanide plants require more equipment to be installed than small amalgamation and concentrating mills. The cyanide and flotation processes depend on a regular (though usually not large) supply of chemical reagents, whereas a small amalgamation mill may be operated almost indefinitely with a relatively small supply of quicksilver, lubricating oil, and small supplies. In remote regions of rugged topography, pack animals are usually relied upon for transportation. In the far north, men and supplies are transported by boats and canoes in summer and by sleds in winter, or, in recent years, by airplanes. This sort of transport is not cheap and increases the cost of supplies tremendously. Long and serious delays are frequent in winter in northern climates or at great altitudes, and the mill should either be equipped and supplied to run without shutdown during interruptions from this source, or operations should be confined to favorable seasons.

From a purely local standpoint the question of transporting the ore from mine to mill must be dealt with, and in some cases may have a prominent part in determining the mill location. In rugged country the management may be confronted with a choice between two or more possible locations, each of which would involve a different transportation problem. As a rather common example we may assume that the mine shaft or main tunnel is located on a steep mountain side, 3 or 4 miles from the nearest good road, railroad, or waterway, with favorable hillside locations available both at the mine and adjacent to the road or waterway. It becomes a question whether to put the mill at the mine and build a road to it for bringing in supplies and water or to install the mill at the lower point, where water is available, and provide an aerial tramway for transporting the ore. For large mills the latter alternative would be more attractive than for smaller operations, since construction of the plant would cost less at the lower site and sufficient tonnage would be handled to make the tramway an economical investment. Sometimes a road has been built to the mine before a mill was even contemplated, but for small gold mines a bad trail is often the only means of access. These are matters upon which it is impossible to generalize to any extent, and they can only be judged in the light of prevailing local conditions.

COST OF SUPPLIES

Supplies account for an appreciable portion of the cost of milling gold ores. Amalgamation is a cheaper method of treatment per ton of ore than flotation or cyanidation, from this as well as from other standpoints. Considerable saving could be effected in some mills by careful supervision and attendance to maintenance work, and in remote regions where the cost of supplies is excessive every effort should be made to reduce repairs by constant attention to lubrication, machine adjustments, and the like. Supply costs at a few North American gold mills are included in Tables 13 and 14, and in the examples of practice.

POWER AND FUEL

Most mills in established districts purchase electricity from power companies, and all machinery is electrically driven. In new districts or remote regions not served by transmission lines the question of power supply becomes especially important. If the mill is to be of moderate or large size a central generating plant is often justified; and in such cases water power, steam, or Diesel engines may be employed as prime movers. The choice between water and engine driven plants will depend on the availability and continuity of a supply of water under sufficient head and the relative costs for dams, ditches, flumes, penstocks, and first cost of equipment on the one hand, as against the cost of fuel and installed plant on the other. Periodic droughts or severe winters are serious difficulties with water-power projects. If a steam or Diesel plant is chosen the choice between them will be determined by the character of boiler water available, the cost of coal or fuel oil, the quality of the oil, and other factors. Small plants may use gasoline or Diesel power with line-shaft transmission, Pelton wheels with line shafts, or electric power obtained from small generating plants driven in any of these ways or by steam—depending, of course, on local conditions.

This subject is too involved for detailed discussion in this paper, but it is one which should not be minimized in importance.[14] Some power costs are shown in Tables 13 and 14 and in the examples of practice.

Fuel for heating may be required for operating in cold climates. The relative cost of different milling methods may be somewhat affected by this item, since a flow sheet using large volumes of solutions, especially if they are pumped through many feet of iron pipe, is more apt to give trouble from freezing than a scheme employing less water. The chief use of fuel around most mills is for keeping them warm enough to work in.

At some cyanide plants treating arsenical, antimonial, or telluride ores a preliminary roast is necessary to prepare the ore for obtaining satisfactory solution of the gold. In such instances the fuel bill may be quite appreciable—often enough to make flotation a more attractive process than cyanidation. Fuel is also used for drying concentrates after filtration at some gravity or flotation concentrating mills.

TAILINGS DISPOSAL

In selecting a mill site the problem of disposing of the tailings or mill waste is nearly always one of considerable importance. If the ultimate tonnage to be treated during the life of the mine is large adequate facilities for handling the tailings at minimum cost should be provided at the outset. The cheapest method of transporting such material is by means of gravity, which in turn requires the use of water. Thus, if other factors permit, it is desirable to locate the mill at a point from which tailings can be piped or laundered down grade to a suitable disposal area. In remote districts where water is abundant it may be most economical merely to discharge the waste material into a stream bed. On the other hand, if water is scarce and

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[14] Taggart, Arthur F. Handbook of Ore Dressing: John Wiley & Sons, New York, 1927, pp. 1302–1317. Peele, Robert, work cited, pp. 1315–1346.

must be recovered for reuse or if the tailings are high enough in gold content to justify the thought that they might be profitably re-treated in the future by the aid of improved processes, it is advisable to provide for storage. This is usually accomplished by impounding the tailings pulp behind a dam; as the pond fills up with settled material the dam is raised. Figure 43 illustrates the tailings built up in this manner at the Tom Reed property in Arizona.

Natural lakes are sometimes utilized for the storage of mill tailings, as shown in Figure 44, which illustrates the way in which this material from three mines at Kirkland Lake, Ontario, is being impounded in Kirkland Lake. In this case the lake provided the cheapest means of disposing of the tailings, but recent developments indicate that much of the material will be re-treated by flotation.

As mentioned on a succeeding page, legal requirements in some localities require the impounding of tailings to prevent stream pollution.

In some instances land near the plant may have considerable value, and the location of the tailings-disposal area with respect to the mill, as well as the method of handling the pulp, may be considerably modified by this factor. It may thus be cheaper in some cases to elevate or pump the tailings than to launder them by gravity to a disposal site purchased at high cost.

LABOR SUPPLY

Since the labor cost of gold milling is usually the greatest single item on the cost sheet, constituting 30 to 50 per cent of the total operating cost, the labor question has utmost importance. In established gold-mining districts a supply of experienced millmen is at hand, and these men are usually honest and fairly efficient. It is, however, unfortunately true that antiquated flow sheets are still in use in some places because the customs of the region have become so firmly implanted that mill foremen and even managers often condemn without fair trial any improvement in methods or equipment as being new fangled and impractical. This attitude has in some instances helped the stamp mill to live beyond its proper time and has hindered the introduction of flotation methods for the milling of ores which are amenable to such treatment at an improved profit.

In tropical regions labor is rather inefficient, so that costs are little if any reduced by its cheapness. White men command high wages and will not remain long in unhealthy climates. In the far north men are hard to get, and at very high altitudes the labor turnover is excessive. In most gold mills, however, the force is not large; the thing of greatest concern, especially in amalgamation, is to find millmen who are efficient, experienced, and honest. Amalgamation results are in a measure proportional to the skill of the men who dress the plates; and losses from theft are sometimes serious, although in the United States and Canada much less frequently so than one would casually suspect. "High-grading" is more commonly practiced by miners than by millmen, for some unknown reason.

Tables 13 and 14 give the labor cost per ton milled at several gold mines. It is interesting to note that although the absolute costs vary widely, due to the use of different methods or to different grades of ore and scales of operating, nevertheless labor accounts for about 40 per cent of the total operating cost, with the exception of one very small and one very large plant. The lowest labor costs per ton of ore are usually obtained in those mills which are laid out to operate in units. In this way each man in the plant is given a more limited and specialized class of work to perform and by becoming proficient at it is able to do his work with greater efficiency and dispatch, thus reducing the size of the crew needed. Attention to the type of equipment used and the manner of fitting it into the general scheme, with its relation to routine labor operation always in mind, will pay ample dividends in reduced labor account.

GOVERNMENT REGULATIONS

In building a mill it is necessary only to comply with the land laws governing location of mill sites (if on the public domain) and in some cases to make provision for impounding tailings to prevent stream pollution. The California débris regulations are an example of the latter. Laws for the protection of health and promotion of safety are not ordinarily unjust or burdensome. Accident-compensation laws are in general force and in some States or Provinces are high; increased costs are the consequence. These factors must be considered for each case according to local regulations.

COST OF CONSTRUCTION

Local construction costs affect the unit capital charge against the ore milled and may greatly influence the type of mill buildings used. Thus, in remote regions where structural steel is expensive local timber is generally employed for small or moderate-size mills. In recent years the use of insulation materials for buildings in cold climates has increased. High costs for steel construction may indirectly result in great losses, inasmuch as they are often responsible for timber being chosen to reduce capital expenditures; in consequence, the plant may be destroyed by fire, as evidenced by the disastrous experience of some mining companies. In this manner, the original saving in construction costs is wiped out many times over, and even should no fire occur the difference in cost is partly offset by higher insurance rates. This is but another of many reasons why it is generally wise to delay installation of a large permanent mill until ample funds are at hand to permit erection of an adequate and sound plant. Too great economy at the outset can be as great a mistake as overlavish expenditure.

GENERAL ECONOMIC CONDITIONS

The bearing of general economic conditions on gold mining has been mentioned in part 1. As pointed out, the price of gold is fixed, so that variations in general commodity price levels and wage rates affect the profitableness of gold mining without any influence from the added factor of fluctuating prices for the product, as in base-metal exploitation. That inflation or deflation in the value of gold resulting from these causes greatly affects the status of gold-producing enterprises is well known; during the war period many mines were hard hit [15] by inflation of labor-supply costs, and the present (1931) vast interest in gold mining is largely due to inflation in the value of gold produced by general depression. However, popular opinion greatly overestimates the effect of commodity price variations on the gold-mining industry. Labor wage rates are not so fluctuant as supply prices and often are not reduced until periods of depression have persisted for many months. In the mining of the ore, where about 60 per cent of the total direct cost is for labor, this means that lowered commodity prices are only partly reflected in reduced costs. In the milling operation labor accounts for an average of about 40 per cent of the operating cost, and variations in supply prices are thus paralleled more closely by variations in the cost of milling than is true for mining. The chief effect of business depression on gold-ore milling is that slightly higher profits are obtainable from ores of a given grade; or, conversely, lower-grade ores may be treated to yield the same profit.

MILLING METHODS

OUTLINE OF CHIEF PROCESSES

In describing the principal methods of gold milling no attempt is made to present a complete or exhaustive treatment of the details of mill operation or of the equipment employed, for to do so would entail useless duplication of material already available in several able and complete published works.[16] Furthermore, such an attempt would contribute nothing new and would tend to confuse and obscure the object sought in this bulletin—to compare the applicability of various methods under different conditions and the results which are obtained or might be expected by their use.

The principal methods used in the milling of gold ores in present-day practice are hand sorting, amalgamation, gravity and flotation concentration, cyanidation, and various combinations of two or more of these methods. They will be considered in the order named. After each method has been briefly described all will be compared and discussed from the standpoint of applicability.

HAND SORTING

An important step in the treatment of many gold ores is hand picking or sorting before fine crushing. Sorting is applicable to material in which more or less barren or very lean rock is mixed with the ore as mined and depends on pronounced visible differences in appearance. By removing a portion of the waste at an early stage the tonnage subjected to fine crushing, grinding, and subsequent treatment is reduced, with consequent lowering in over-all costs. At many properties the mining method and character of the

[15] Report of a Joint Committee Appointed from the Bureau of Mines and the U. S. Geological Survey by the Secretary of the Interior to Study the Gold Situation: Bull. 144, Bureau of Mines, 1918, 84 pp.

[16] Taggart, Arthur F., Handbook of Ore Dressing: John Wiley & Sons, 1927, 1679 pp. Rose, T. K., The Metallurgy of Gold: C. Griffin & Co. (Ltd.), London, 6th ed., 1915, 601 pp. MacFarren, H. W., Practical Stamp Milling and Amalgamation: Min. and Sci. Press, 3d ed., 1914, 225 pp. Clennell, J. E., The Cyanide Handbook: McGraw-Hill Co., New York, 2d ed., 1915, 601 pp. Peele, Robert, Mining Engineers' Handbook: John Wiley & Sons, 2d ed., 1927, pp. 1797–1959. Gaudin, A. M., Flotation: McGraw-Hill Book Co. (Inc.), New York and London, 1st ed., 1932, pp. 321–342.

ore permit underground sorting; at others, the entire tonnage must be hoisted. In the latter instance it often pays to screen and wash the product of the primary crushers before it is passed over a picking belt.

The degree to which it may be economical to carry sorting must be determined by comparing the saving in fine milling expense with the cost of sorting and the value of the recoverable gold lost in the sorted waste. At the Alaska-Juneau [17] mine nearly half the tonnage mined is discarded by screening and hand sorting at a cost of \$0.13 to \$0.15 per ton rejected, and the balance is milled at a cost of \$0.31 to \$0.33 per ton milled. The sorted waste carries \$0.19 per ton in gold and the sorted ore assays \$1.92 per ton, from a mine run assaying \$1.10. The advantages of sorting in this case are readily apparent. The desirability of sorting usually is not so pronounced as at Juneau, but it is a matter worthy of careful investigation. At some small high-grade mines, sorting is the principal or only method of preparing the ore before shipment.

AMALGAMATION

Amalgamation of gold with mercury was the principal means of recovering gold from ores until the successful introduction of the cyanide process; and it is still of great importance, especially in small mills. The method may be divided into two main subdivisions: Plate amalgamation and barrel amalgamation.

PLATE AMALGAMATION

In plate amalgamation the ore is finely crushed in stamp mills or ground in ball or tube mills and the pulp passed over copper plates (often silver plated) which are covered (dressed) with a coating of amalgam and mercury. The free gold liberated by crushing is absorbed by the quicksilver and combines or alloys with it to form gold amalgam, which is retained on the plates.

Where stamps are used they are usually of the gravity type, weigh 900 to 1,500 pounds each, and are operated in batteries of 5. After preliminary crushing in jaw breakers and gyratory or cone crushers, the ore is fed to the mortars of the battery. Stamp feed usually is 2 to 3½ inches in maximum size. Water is added in suitable quantities, depending chiefly on the particle size to which stamping is carried, and the ground pulp is discharged from the mortar box through a screen. The screen openings determine the maximum size of particles fed to the outside plates; screens are made either of steel plate with slotted punched openings or of steel, brass, or bronze wire. Screen openings range from 12 to perhaps 40 mesh in ordinary service, depending on the height of mortar discharge.

Mercury is sometimes added to the battery feed, especially where much coarse gold is present, to give a longer period of contact and thus effect more complete amalgamation. The amalgam formed in this way is largely trapped between the dies and at the inside base of the mortar box. Care must be used to prevent "flouring" (churning up of mercury in the battery until it takes the form of a mass of small globules which sometimes float on water and do not readily

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[17] Bradley, P. R., work cited.

coalesce with each other). "Sickening" of the mercury, a term used to describe its condition when coated with thin films of base-metal compounds derived from the ore or from impurities in the mercury itself, may cause much trouble. Arsenic and antimony are the worst promoters of sickening and cause a black film to coat the mercury so that it will not amalgamate gold nor coalesce. Grease, graphite, talc, clay, and similar materials cause a form of sickening by protectively covering the mercury surface and inducing mechanical separation; they also cause a loss by coating of the gold particles. Most base-metal sulphide minerals will sicken mercury if partly oxidized. The oxidation of sulphide minerals that form acid in the water is often counteracted by adding lime to the ore feed. The use of some sodium amalgam on the plates increases recovery from some troublesome ores, as it is a powerful amalgamator and will take up many substances that cause sickening, although it does not help very much on antimonial ores. Where sickening is particularly active, occasional cleaning with weak cyanide or other solution is advisable to remove the impurities and brighten the plates.

Inside amalgamation, or the use of small plates inside the stamp mortar, is a practice formerly popular but not now much employed, since the use of heavier stamps and lower mortar discharge to increase capacity has necessitated the use of mortars too small to accommodate the plates; furthermore, the larger capacity batteries produce coarser, faster flowing, and consequently more abrasive pulps, which scour the plates badly. Excessive flouring has likewise resulted from this practice.

Outside plates are 4 to 5 feet wide and 6 or more feet long. The plates are usually set in series with about a 4-inch drop between each plate. They are set on firm bases with 2-inch sides and have slopes of 1½ to 3 inches per foot, according to the rate of discharge and the fineness of the gold. It is desirable to have sufficient slope to keep the particles in the pulp in free motion and prevent banking; on the other hand, with too much slope, some fine gold will be washed over the plates without coming into contact with the quicksilver and thus escape amalgamation. To insure contact of all the gold with the plates, it is desirable that the pulp shall pass over them in a series of waves. This action may be induced by proper, even distribution at the head of the plates, and is increased by a series of drops of a few inches between successive plate sections. Pulp densities of 10 to 25 per cent solids have been found best on outside plates. The use of swinging amalgamated plates suspended in the pulp stream in launders below the plates assists in recovering fine gold. Stationary blanket tables and riffles are sometimes used instead of plates. They are discussed under Concentration.

Plates are dressed at intervals of a few hours, according to the grade of ore, amount of sickening, and degree of scouring due to abrasion or discoloration of the plates by chemical action. Dressing usually consists of brushing from the bottom upward with a whisk broom or sometimes with a rubber scraper, removing excess amalgam and sprinkling fresh mercury through a muslin or chamois bottle stopper where needed and working the mercury until the surface shows the proper amalgam condition. At intervals, usually monthly or semimonthly, a thorough clean-up is made. At this time, the plates are scraped (but not to the bottom of the amalgam) and carefully redressed with fresh quicksilver. Rubbing the new "quick" into the amalgam with a cloth and liberal doses of "elbow grease" is a common and commendable practice. At some mills the plates are removed for cleaning-up and immediately replaced with freshly dressed plates. Careful attention to the condition of the plate surfaces is essential to satisfactory amalgamation results. Between surprisingly wide limits, the recovery obtained is proportional to the skill and application bestowed upon the tack of dressing the plates.

The amalgam cleaned up is squeezed within a chamois to remove excess mercury, placed in a cast-iron retort, and slowly brought to bright red heat. The mercury is driven off in the form of vapor and recovered (for further use) in a condenser. The gold residue or sponge is melted in graphite crucibles with suitable fluxes, such as borax, and then cast in bricks or doré bars.

The stamp battery as a crushing machine has been heartily condemned in recent years by many engineers concerned with the metallurgy of gold and stoutly defended by numerous practical millmen and operators accustomed to its use. Many sound arguments for and against the battery have been advanced, but nevertheless thousands of stamps are still dropping in numerous mills throughout the world. It is significant, however, that new mills of the larger size generally employ other methods of grinding and that some large plants are replacing the batteries, as they wear out, with ball mills. It is now pretty generally agreed that stamps are inefficient as compared to ball or tube mills when the tonnage to be handled is large enough to keep a medium or large size grinding mill constantly running at full load. The manifest advantage of the stamp battery is its small unit capacity, which permits great flexibility by the simple expedient of shutting down one or more batteries when the volume of heads is reduced and running the rest at full efficiency. This characteristic is so desirable in small mills that, combined with the fact that stamps compare favorably with the smaller sizes of revolving mills from the standpoint of operating cost, it often justifies the installation of stamps in plants treating hard ores and having capacities of 200 tons or less.

An exception is the Homestake mill where stamps were installed in the new mill ahead of ball mills for a particular coarse crushing problem.

Where amalgamation is employed with ball or tube mills the plates follow the last mill, unless part of the gold in coarser sizes is abundant enough to justify the use of plates between grinding stages. The best practice favors recovery of the gold by a series of plates, the recovery to begin as early as practicable. Shaking amalgamation plates are occasionally used after plate amalgamation or concentration. They are effective, but flouring is excessive and mercury and amalgam losses are apt to be high.

Traps are always provided below the plates to catch any loose mercury and amalgam which escape from them. Various designs of traps are used; usually they are merely modified launders having overflow lips high enough above the bottom of the trough to prevent heavy particles from flowing over. In some mills sumps serve as traps and are cleaned out at relatively long intervals.

BARREL AMALGAMATION

In barrel amalgamation the feed usually consists of heavy concentrates from corduroy tables or similar equipment placed after the stamps or grinding mills, of rich picked ore, or various secondary products. The barrel, a steel cylinder carrying balls or one or more pieces of shafting as the grinding medium, is charged with feed and mercury and revolved for several hours. The material is then removed from the barrel, the amalgam is panned out and treated in retorts, and the tailings are either re-treated on the next clean-up, or further ground and cyanided.

PAN AMALGAMATION

Pan amalgamation, in which the ore or concentrate is ground with mercury in cast-iron tubs or pans equipped with rotating dies or grinding shoes, was formerly practiced to some extent but is now practically obsolete in gold milling. The Berdan pan is a modification used for amalgamating high-grade ore and rich table sands at some mills. It consists of a tilted revolving pan, and uses steel balls for grinding.

ARRASTRES

Arrastres were important in Mexico and elsewhere in early days. They consisted of a circular pavement of stone surrounded by a low wall and having a central post on which were pivoted radial arms attached to a sweep. Large blocks of stone were suspended from the arms and were dragged around the pavement by means of a horse or burro hitched to the sweep. The ore was mixed with quicksilver and ground for several hours, water being added in sufficient quantities to form a rather dilute pulp. When all the mercury and amalgam had settled to the bottom, as indicated by barren pannings of the pulp, the surplus water was drained and the amalgam recovered by panning. Good extraction is possible in arrastres, but the process is slow and laborious. It still has occasional application for preliminary work on prospects in remote regions, when the available capital is very small.

CONCENTRATION

Concentration of gold ores from run-of-mine material into higher-grade products of less bulk is necessary for the commercial success of many gold-mining ventures. The methods employed are based either on differences between the specific gravity of the common gangue minerals and metallic sulphides and gold, or on the surface-tension characteristics of the various minerals which permit application of the flotation process. Until quite recently, gravity concentration was the only method used in treating gold ores. It is still the most widely used, but in comparatively recent years greater understanding of the basic principles and notable technical advances in the art have so greatly widened the scope of the flotation method and reduced its cost that it is now quite widely utilized in the treatment of gold ores. It is perhaps most commonly applied as an auxiliary to other methods, but several gold mills are using it as the principal or only method, and many others would probably find it better adapted to the ores treated than the older methods now in use. Unquestionably the flotation process will be seriously considered in designing the flow sheets of new mills to treat sulphide-bearing ores.

GRAVITY CONCENTRATION

A large number of gravity-concentrating devices have been devised and used in gold-ore mills, but only a few are of outstanding importance in present-day practice. Corduroy blankets, coco matting, or similar material on tables, shaking tables, and vanners will be considered briefly.

STATIONARY TABLES

Blanket tables, consisting of flat, sloping surfaces covered with strips of corduroy blanket cloth, coco matting, or coarse-woven canvas, are used in many plants. The thin pulp is spread over the head of the table; and the heavy minerals, including gold, sink to the bottom and are caught in the riffles of the corduroy or between the fibers of the matting or other material that may be used. Riffled tables and launders are sometimes used in the same way as blankets for catching coarse gold. Concentrates, washed from the blankets or riffles at short intervals, are amalgamated in barrels (p. 108) or pans, or are cyanided, or sent directly to smelters. The tailings usually receive further treatment on shaking tables or by cyanidation or flotation, except in a few small mills.

Blankets, matting, and riffles have two rather distinct fields of use. In the earlier practice blankets were chiefly employed as scavengers or guards to catch any gold not retained on the plates or saved on shaking tables, and they are still recognized as excellent devices for this service. Later, they became competitors of plate amalgamation and have directly replaced the plates at some mills, notably on the Rand. Although they usually do not save as much free gold as plates they cost considerably less to operate, and where the pulp is subjected to further treatment by other methods they frequently yield improved economic results. At the 70-ton mill of Granada Gold Mines (Ltd.), Rouyn, Quebec, blanket tables were substituted successfully for plates because of the deleterious effects on the mercury of organic matter in the muskeg lake water used in milling (fig. 45).

Riffle tables are employed satisfactorily at the Sixteen to One mill, Alleghany, Calif., where the values are chiefly in the form of coarse gold.[18]

If precautions are taken thefts are not as easy to perpetrate with blanket tables or riffles as when plates are used.

SHAKING TABLES

Shaking tables are of several types, but they all operate on the same basic principle. A flat sloping surface, fitted with thin riffle boards parallel or diagonal to the long dimension (for sand feed) or covered with canvas or similar material (for slime feed) is rapidly oscillated in a direction parallel or diagonal to the long dimension by an eccentric mechanism. The slope is both parallel and at right

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[18] Young, G. J., work cited.

angles to the long dimension and is adjustable to suit the needs of the particular material treated. The ground ore, more or less classified or sized according to conditions, is fed to the table from a small trough along the upper long edge of the deck (table surface) near the head end. Additional water as needed is distributed at desired points on the upper long edge from another trough. In operation, the heavy particles settle into the riffles or rest on the deck surface and are carried along by the slope of the table and the jerking motion, while the lighter material is more mobile and is washed sidewise by the stream of water. Concentrates are taken off into

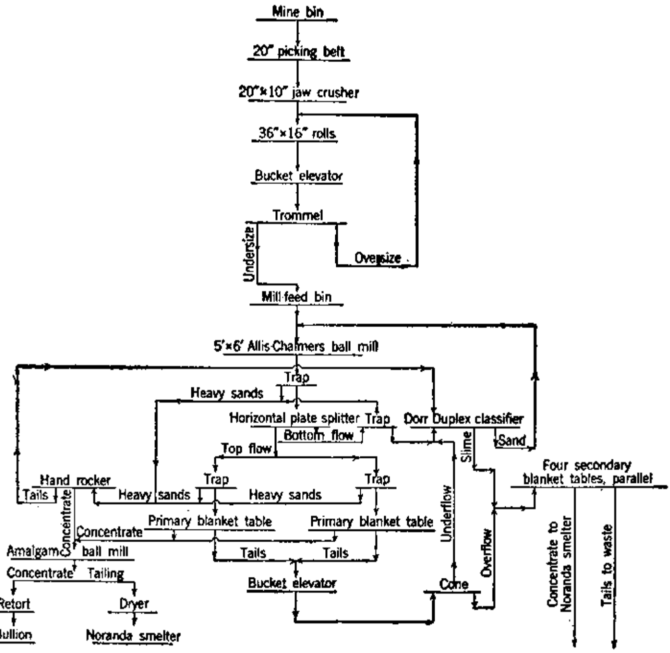

FIGURE 45.—Flow sheet of 70-ton mill, Granada Gold Mines (Ltd.)

launders at the end of the table and middlings and/or tailings into separate launders along the lower side. Slopes, sizes, speed, pulp density, and amount of water fed vary with different ores.

VANNERS

The Frue type, which is the best known of the vanners, consists essentially of a broad belt with flanged edges run over large head-and-tail pulleys at slow speed. A moderate slope is maintained on the top of the belt from the head pulley toward the lower end, and a rapid oscillating motion, either transverse (as in the Frue type) or longitudinal, is induced by suitable mechanical devices. The pulp is fed in a thin layer near the upper end, and fine jets of water are directed against the upward moving belt from a pipe between the pulp feed and the head pulley. Fine or light material is washed down and off the lower end, while the heavier grains hug the belt and are discharged over the head pulley. The vibrating motion, aided by the jets of water, keeps the light particles in motion and prevents banking. As in the case of tables, the degree of concentration is controlled by varying the slope, feed, and amount of water.

CLASSIFICATION

Sizing and classification for table and vanner concentration are usually done in classifiers of the mechanical type (rake, drag, or screw machines) for the coarser sands, and in spitzlutten, spigot and cone classifiers, settling tanks, or other types of hydraulic classifiers for the finer sizes. Theoretically, gravity methods of concentration are most effective when conducted on well-sized feed, but practically a balance must be struck between the cost of different degrees of classification and the added recovery obtained in each case. In many mills a concentrate and a middling are made on sand tables; the middlings are reground and classified and further treated on slime tables or vanners or by flotation or cyanidation. Stage grinding of this nature nearly always pays where a fair proportion of the values are liberated and may be cut out at the first stage, thus reducing the tonnage necessary to subject to expensive finer grinding.

TREATMENT OF CONCENTRATES

Table and vanner concentrates are partly dewatered in thickeners of various types, filtered (in North America most filters are of the continuous vacuum variety), and then subjected to barrel amalgamation, cyanidation, or smelting, depending on the nature of the particular concentrate. Those which contain much coarse free gold are usually treated in a barrel, the residue being cyanided. If the gold is chiefly locked up in sulphides, direct cyanide treatment or smelting is preferable.

FLOTATION

The flotation method of concentrating gold ores is rapidly growing in importance. It is applicable to ores carrying quantities of sulphide minerals sufficient to stiffen or stabilize the froth; at present very lean or nonsulphide ores are not usually amenable to this treatment.

Briefly, the process comprises preliminary crushing, followed by grinding to 48 mesh or finer, conditioning (usually in agitation tanks) with proper flotation reagents as determined by test or experience, and separation of the gangue from the ore minerals in the flotation cells. The metallic and sulphide minerals are wetted by the small amount of reagents in the pulp, and through the action of certain surface physical phenomena which are not entirely understood the particles of ore adhere to fine bubbles coated with films of oil or other frothing agents. Many minerals of nonmetallic luster can be floated with suitable reagents, but in gold mills they are not important in flotation practice. The froth is produced in the cell by mechanical agitation, by the introduction of air under pressure, or both; it rises to the surface and is skimmed off into launders, while the gangue flows out at the end of the cell.

Rough concentrates with clean tailings, rich concentrates with middlings or high tailings, and various other combinations of results are obtainable by varying the amount and character of reagents, pulp density, degree of agitation, speed of treatment, and so forth. Stage grinding and classification are effective, as in gravity methods. In fact, the two methods are analogous in most respects, except that the one is based on differences of specific gravity while the other depends on the surface tension and other properties of the minerals.

Flotation concentrates are thickened and filtered in the same way as gravity concentrates. They are usually cyanided, roasted and cyanided, or smelted, depending on the nature of the ore.

Where flotation is used on tailings or as an auxiliary to other methods certain modifications are made, but the general treatment is similar in all cases. The subject is too involved for more than the briefest treatment here; many useful data are available in textbooks and in numerous articles in the technical press.

CYANIDATION

The cyanide process, the history of which has been very briefly mentioned on page 90, based on the solubility of the precious metals in dilute solutions of the alkaline cyanides. Much has been written about the chemistry and mechanics of the process. Detailed discussion of this broad subject is unnecessary in this paper, because of the wealth of published information already available. For the purpose of pointing out the applicability of the process and the results obtained by its use a brief outline of the principal features is presented.

CHEMISTRY

Sodium cyanide (NaCN) or less frequently potassium cyanide (KCN) is used in weak solution as the dissolving agent. Strengths of reagents are quoted in terms of KCN. Sodium cyanide is the more efficient per unit weight, since it has 32.65 per cent more available cyanogen (CN) than potassium cyanide.

Gold reacts in cyanide solutions to form the soluble double alkali cyanide according to the following basic equation, known as Elsner's equation:

$$2Au+4NaCN+O+H_2O=2AuNa(CN)_2+2NaOH.$$

From this statement it will be seen that oxygen is essential to the reaction; the need for supplying it in adequate quantities has been chiefly responsible for the remarkable development of the mechanical equipment employed in present-day practice.

Precipitation of the values from solution is usually accomplished, after clarifying and deaërating the pregnant solution, by contact with zinc dust or shavings, according to the equation:

$$KAu(CN)_2+2KCN+Zn+H_2O=K_2Zn(CN)_4+Au+H+KOH.$$

Excess zinc being present, more $K_2Zn(CN)_4$ is formed by reaction with cyanide than would be necessary for complete precipitation of the gold, and indirectly by various other reactions. As a result, zinc salts tend to build up; and if the solutions are continuously reused they become fouled, with resultant weakening of dissolving capacity. Consequently, large excess amounts of zinc should not be used.

Various methods for regenerating cyanide from fouled solutions have been devised; of these, the acid of the Mills-Crowe process, in which the solution is acidified with sulphurous acid and passed in towers against air under low pressure to remove the cyanogen as HCN and thence through more towers containing alkaline solutions to absorb the HCN as NaCN or $Ca(CN)_2$, is practical for some ores. A method using sulphur dioxide is successful at several mills.

Aluminum dust is sometimes used in precipitation for unusual conditions and has the advantage of not fouling solutions but can not be used in lime-bearing solutions. Charcoal has also been used but has no commercial importance.

APPLICABILITY AND LIMITATIONS

Nearly all gold ores are amenable to cyanidation, although some of them possess characteristics that cause excessive consumption of reagents, fouling of solutions, and resultant high costs. A few are so refractory that the method can not be directly applied with commercial success and must be modified either by the introduction of more or less costly preliminary steps or replaced by another treatment method, such as flotation.

The chief obstacles to cyanidation are substances known as cyanicides, which consume cyanide and foul the solutions. The commonest cyanicides are soluble sulphides, sulphates, arsenates, and antimonates, iron and copper salts, free acid, and carbonaceous matter. Soluble sulphides not only consume cyanide but may, by various reactions, cause high oxygen consumption and even reprecipitate silver and gold. The last effect may be partly counteracted by keeping the solutions up to a certain strength, dependent on the ore being leached. Free acid decomposes cyanide and causes loss of strength by evolution of HCN gas; this effect is prevented by maintaining a protective alkalinity with lime. Arsenic and antimony minerals react with cyanide and destroy its usefulness, and they sometimes form compounds which display a strong reducing action. Roasting usually helps this situation, if properly done. These elements, if present in appreciable amounts, may deoxidize and foul the solutions to such an extent that gold will not be dissolved, and to prevent such an eventuality roasting is nearly always necessary. If the roast is carried out too long or at too high a temperature (as 600° C. or more), certain arsenates and antimonates are formed which lock up the gold and preclude its dissolution. A special low-temperature roast (at about 450° C.), followed by treatment with lime before cyanidation, has been shown by Leaver and Woolf [19] to yield good results on refractory ores possessed of these characteristics.

Oxidized copper and iron minerals readily form double alkali cyanides, using up reagents and increasing costs; furthermore, ferrous iron and copper salts are not rendered innocuous by the presence of lime, and losses due to them are difficult to prevent. One method is to employ an acid wash prior to cyanidation to remove the oxidized minerals.

Another method, which consists in treating the pregnant solution with sodium sulphide and then with sulphuric acid (or sometimes sulphur dioxide gas) in air-tight apparatus, is described by Leaver

[19] Leaver, Edmund S., and Woolf, Jesse A., Cyanide Extraction of Gold and Silver Associated with Arsenic and Antimony in Ores; Tech. Paper 423, Bureau of Mines, 1928, 52 pp.

and Woolf.[20] In this method, the sulphides of silver, mercury, and zinc are precipitated by agitation with sodium sulphide, and copper sulphide is thrown down when the solution is acidified. The copper can thus be separated from the silver, if desired. Hydrocyanic acid (HCN) gas is liberated on acidification but is converted to safe usable form when the solution is made alkaline with lime. Less than 10 per cent of the gold is precipitated by the sulphide, the balance being obtained in the ordinary way by the aid of zinc dust after the silver and base sulphides are filtered out and the solution made alkaline. By this method, ores containing cyanide-soluble copper and zinc may be treated without excessive cyanide loss, and in some instances a commercial by-product of copper is obtainable. The cyanide that combines in simple compounds with copper and zinc is more than 80 per cent regenerated; the more complex cyanide compounds, cyanates and sulphocyanates, are not regenerated.

Carbonaceous matter in an ore consumes cyanide and causes premature precipitation of values. When it is abundant, as in the slimes of Mother lode ores, it carries into the tailing considerable gold, which can be recovered economically only by flotation. If amalgamation is employed ahead of cyanidation some mercury may enter the vats. If the mercury is oxidized or combined as sulphide it is dissolved by the cyanide, but metallic mercury is not soluble in cyanide solution. In small amounts this element is helpful, in that it tends to precipitate soluble sulphides in the pulp and also aids the zinc precipitation of gold from the pregnant solution, but excess quantities cause high consumption of zinc.

Telluride ores are not directly amenable to ordinary cyanide treatment but must be roasted to release the gold in soluble form. Another method involves the use of bromocyanogen (BrCN) in the lixiviant; tellurides are readily dissolved in this way, but costs for reagents are rather high. Small amounts of tellurides in the Kirkland Lake ores are successfully handled by very fine grinding, long treatment (36 to 40 hours) with frequent aeration and changes of solution, and the use of sodium peroxide to break down tellurides.

Some refractory ores, unless they carry rather large amounts of arsenic, antimony, tellurium, or carbon, can be cyanided successfully by fine grinding (sliming) until 75 per cent or more of the pulp will pass a 200-mesh screen. Even greater degrees of comminution are in use at some plants, and the tendency to grind finer and finer is still noticeable.

METHODS OF TREATMENT

Cyanide treatment consists of four steps—preparation of the ore (crushing, grinding, classification, etc.); leaching or dissolving the gold; separation of the pregnant solution from the leached pulp (thickening, filtration, and clarification); and precipitation of the values by zinc or occasionally by aluminum or other methods.

Preparation of the ore involves crushing in jaw or gyratory machines, secondary reduction in disk or cone crushers, rolls, or stamp mills, and final grinding to the desired size in ball and/or tube mills, accompanied by suitable classification. Grinding is often carried out

[20] Leaver, Edmund S., and Woolf, Jesse A., Copper and Zinc in Cyanidation; Sulphide-Acid Precipitation: Tech. Paper 494, Bureau of Mines, 1931, 63 pp.

in cyanide solution, thus decreasing the subsequent agitation necessary.

The leaching operation is done in different ways, according to the character of the material under treatment. Thus, the entire tonnage may be finely ground and treated by the agitation or all-slime process; the material may be separated into sand and slime portions and each cyanided separately; or concentrates may be cyanided by one method and tailings by another.

Sand leaching.—Sand leaching is done in tanks fitted with filter bottoms. The pulp is charged by hand or by means of mechanical devices, drained, and subjected to successive leaches with cyanide solution, each leach usually being weaker than the one preceding. Solutions are applied either at the top or under slight pressure at the bottom of the tank. Draining and leaching in successive stages give better results than continuous passage of solutions, since the charge, being better aerated in this way, receives an ample supply of oxygen. Sometimes air is forced through the sands between leachings. For efficient leaching the charge should be well classified and free from slimes. The latter, if present, are likely to pack and blind the charge, resulting in "channeling" and consequent poor recovery. Various modifications are practiced, but at most plants three cyanide solutions and one water wash are used.

All-slime leaching.—All-slime or agitation leaching depends on the constant movement of the ore particles in well-aerated cyanide solutions. The ore is ground by means of ball or tube mills, with suitable classifying equipment, to a degree of fineness that will produce the most satisfactory over-all results. Nearly always, grinding must proceed to at least 75 per cent minus 200 mesh, and often much finer comminution is practiced; in some instances, the ore is reduced to minus 300 mesh, if the values are very finely interlocked with the gangue or if some tellurides are present. In the latter instance, the very fine grinding promotes better contact between the telluride minerals and the oxygen in the solutions and thus facilitates the breakdown of the tellurides and dissolution of the gold contained in them.

Grinding in cyanide solution is desirable with agitation leaching. The time of treatment in the tanks is thus reduced materially, since the agitation and aeration given the pulp while it is in the grinding mills, classifiers, and launders are very effective in promoting the dissolution of gold. Solution grinding is commonly practiced, except at some mills where amalgamation is included in the flow sheet (the undesirable effect of cyanide on the plates has already been mentioned); where preliminary acid washes are used; or where strong solutions are employed and cyanide loss would be high, due to the necessary discard of excess solution.

After grinding and classification have reduced the ore to the desired size the pulp is thickened and fed to the agitation tanks, where it is brought to the desired consistency and cyanide strength by the addition of stock cyanide solution. The tanks most commonly used are the Dorr type and the Pachuca or Brown type. The latter depends on compressed air for agitating the pulp. It consists of a steel tank 30 to 55 feet high by 7 to 22 feet in diameter, with a conical bottom. A central pipe, supported so that the open bottom end is about 18 inches above the tank bottom, extends nearly to the top of the tank and acts as an air lift. Pulp charged to the tank slowly settles to the bottom and is drawn into the central pipe by a rising current created by the introduction of air into the lower end of the pipe under a pressure of about 30 pounds. In rising through the pipe the pulp is thoroughly mixed and agitated with solution and air. It overflows at the top of the pipe back into the tank. Disadvantages of the Pachuca tank are its height, weight, and rather high power requirements. Some ores produce sticky pulps which tend to adhere to the tank walls; periodic cleaning is necessary in such cases.

The Dorr-type agitator, now widely used, combines pneumatic and mechanical methods of agitation in a single machine. Tanks, fitted with radial plow arms near the bottom, a hollow central shaft (with air inlets near the bottom), and radial revolving feed launders above the discharge level at the top, comprise the equipment. The settled pulp is moved by the plows toward the center of the tank, where it is raised by the central air lift and redistributed by the revolving launders over the surface of the charge.

The time of agitation necessary varies widely for different ores and must be determined by test and experience in each case.

Agitation leaching by batch methods can be rather closely controlled, but costs are higher than with the more recent continuous treatment, which is now almost universally practiced in slime plants. To insure adequate leaching time for each particle in the pulp and prevent premature discharge by short-circuiting, continuous agitation is always done with several tanks in series. At least three should be employed; and more are better, especially in large plants. An extra tank, installed as a stand-by, permits cutting out any one of the others for cleaning, repairs, and so forth.

After leaving the last agitation tank the leached pulp may be thickened and washed successively several times and finally filtered, or it may be subjected to continuous countercurrent decantation, as in some modern plants. In the latter method three, four, or more thickeners are arranged in series. The leached pulp and rich solution enter the first thickener from the last agitator; most of the pregnant solution overflows and goes to clarification, deaeration, and precipitation (either directly or via the grinding circuit), while the slimes, plus some rich solution, are discharged to the next thickener. In this tank the pulp is mixed with the much lower grade overflow from the third tank. After mixing in the second tank most of the dissolved gold which came in with the pulp is dispersed throughout the solution and raises its value. The clear solution goes to the first tank, while the thickened pulp discharges to the third, carrying with it some solution of lower grade than that which it brought from the first thickener. The process is repeated in each tank, the pulp becoming successively leaner as regards the value of the solution carried along with it, while the main body of solution is successively enriched at each step. At the last tank fresh water is added sufficient to offset the amount of liquid leaving the circuit with the leached slime plus spillage and other losses, and the slime goes to the tailings pond or to filtration with further washing.

Cyanidation of concentrates.—Concentrates present a special problem in cyanidation because of their higher value, greater weight, and greater tendency to form fine sticky slimes than ordinary siliceous ores. Sand leaching can not be applied to them as a rule because of difficulty in maintaining protective alkalinity, blinding or packing of the charge with consequent poor aeration and uneven percolation, and long time of leaching for satisfactory extraction. In agitation treatment concentrates must be finely ground, and they generally require much stronger cyanide solutions than do ores. Solutions used on different concentrates may vary in strength between such wide limits as 1 and 22 pounds of KCN per ton. Greater cyanide consumption and more fouling of solution are generally experienced in treating concentrates than with lower-grade materials.

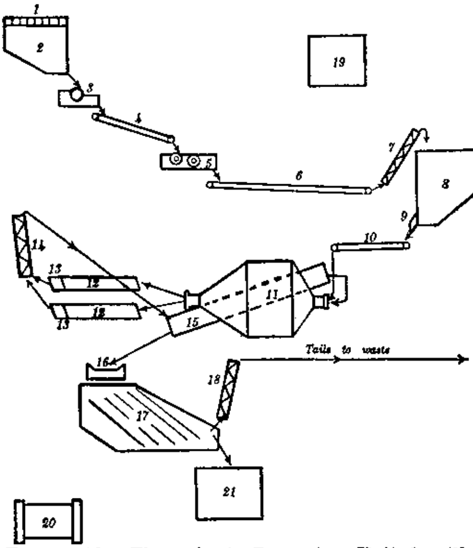

FIGURE 46.—Flow sheet, Porcupine United mill: *1*, Grizzly; *2*, crude-ore bin; *3*, jaw crusher; *4*, belt conveyor; *5*, rolls; *6*, belt conveyor; *7*, bucket elevator; *8*, mill ore bin; *9*, chute; *10*, ball-mill feed conveyor; *11*, ball mill; *12*, amalgamation plates; *13*, blankets; *14*, bucket elevator; *15*, Dorr classifier; *16*, Gibson amalgamator; *17*, James tables; *18*, tailings elevator; *19*, water tank; *20*, amalgamation barrel; *21*, concentrates bin

One variation of the usual agitation practice that seems well adapted to flotation concentrates involves fine sliming in tube mills and bowl classifiers, followed by alternate agitation and filtration in three successive stages, the filtrate being washed with solution twice and with barren water at the last filter.

EXAMPLES OF CURRENT PRACTICE

Some typical examples of present practice in the application of the milling methods discussed in the foregoing pages are presented below. Flow sheets are given for most of them, with a brief summary of the principal operating conditions, milling costs, and consumption of power, supplies, and reagents.

PORCUPINE UNITED GOLD MINES (LTD.), TIMMINS, ONTARIO [21]

Flow sheet.—Figure 46.

Ore.—Quartz and mineralized schist, with pyrite and visible gold; 75 per cent free milling.

Capacity.—25 tons per day.

Power.—Electric, purchased; consumption, 40 kilowatt-hours per ton.

Water supply.—Mine.

Assays.—

	Per ton
Mill heads	$11.00
Amalgamation tailings (table heads)	2.80
Table tailings	1.80
Table concentrates	40.00

Extraction.—Seventy-five per cent of gold in feed, by amalgamation (40 per cent on plates, 35 per cent in barrel). Ten per cent of gold in feed, in concentrates. Eighty-five per cent of gold in feed, total mill recovery.

Operating cost.—$1.851 per ton of ore.

Labor.—

	Man-hours per ton
Crushing	0.32
Amalgamation	.64
Plant foreman	.32
Total	1.28

Supply consumption.—

		Per ton
Grinding balls	pounds	1.50
Ball-mill liners	do	.48
Mercury	ounce	.05

Summary.—Amalgamation and gravity concentration. Plates are dressed every three hours and cleaned up daily. Gold and amalgam caught in sumps, launders, and traps and on blankets are treated periodically in the amalgamation barrel. The Gibson amalgamator catches fine gold and mercury which escapes plates and blankets. Table concentrates are sent to a near-by cyanide plant for treatment. Mill operates on three shifts with a foreman (who takes care of amalgamation) and a crusher man on day shift, and one amalgamator on each of the other shifts.

ORIGINAL SIXTEEN TO ONE MINE (INC.), ALLEGHANY, CALIF. [22]

Ore.—Clean quartz carrying free gold, mostly coarse, in erratic distribution. Most of the production is from rich shoots. Finely disseminated gold occurs in one part of the mine. Pyrite and nonmetallic gangue other than quartz are present in minor amounts.

Tonnage.—40 tons per day. A small tonnage hand-sorted in the mine accounts for about two-thirds of the gold produced.

Power.—Electric, purchased.

Assays.—Chief production is from erratic high grade, sorted underground. Sixteen to One mill heads variable, usually ranging between $1.60 and $9.55 per ton; Tightner heads are similar in grade; concentrates (Sixteen to One mill), variable with feed; concentrates (Tightner mill), $80 per ton (average); Berdan-pan residues (from mill clean-ups), $80 per ton average; Berdan-pan residues (from high-grade treatment), $300 per ton average.

Tailings (Sixteen to One mill).—Variable with feed, average $0.40 per ton minimum.

Extraction.—High grade: Nearly 100 per cent in amalgam and Berdan residue. Sixteen to One mill: Variable, increasing with higher-grade ore. Ranges from 70 to 92 per cent.

Summary.—High-grade ore supplies two-thirds of the gold. This material is sacked underground, crushed in the clean-up room, and amalgamated in Berdan pans. Amalgam is retorted without squeezing, and the gold is shipped. Rich residue sands are sent to the smelter. The mills are used to recover any gold missed in sorting as well as the values in lower-grade development rock.

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[21] Vary, Ronald A., Amalgamation Practice at Porcupine United Gold Mines (Ltd.), Timmins, Ont.: Inf. Circ. 6433, Bureau of Mines, 1931. 5 pp.

[22] Young, G. J., Gold-Ore Mining and Milling (deals with the Sixteen to One enterprise): Eng. and Min. Jour., vol. 132, Sept. 14, 1931, pp. 195–199.

The Sixteen to One plant treats the ore containing coarser gold. After crushing, the ore is ground in a ball mill to minus 0.29 inch and passed through a 10-foot sluice equipped with Hungarian riffles; 70 per cent of the contained gold is caught in this sluice. The sluice discharges into a Dorr simplex classifier which returns sands to the ball mill and discharges an overflow containing 33⅓ per cent minus 100-mesh material to four 3 by 8 foot stationary canvas tables fitted with ⅜-inch riffles; these tables catch 12 per cent of the gold. The pulp then passes over three 5 by 12 foot riffled tables in series, which save 7 to 8 per cent of the gold. These riffle tables have replaced the amalga-

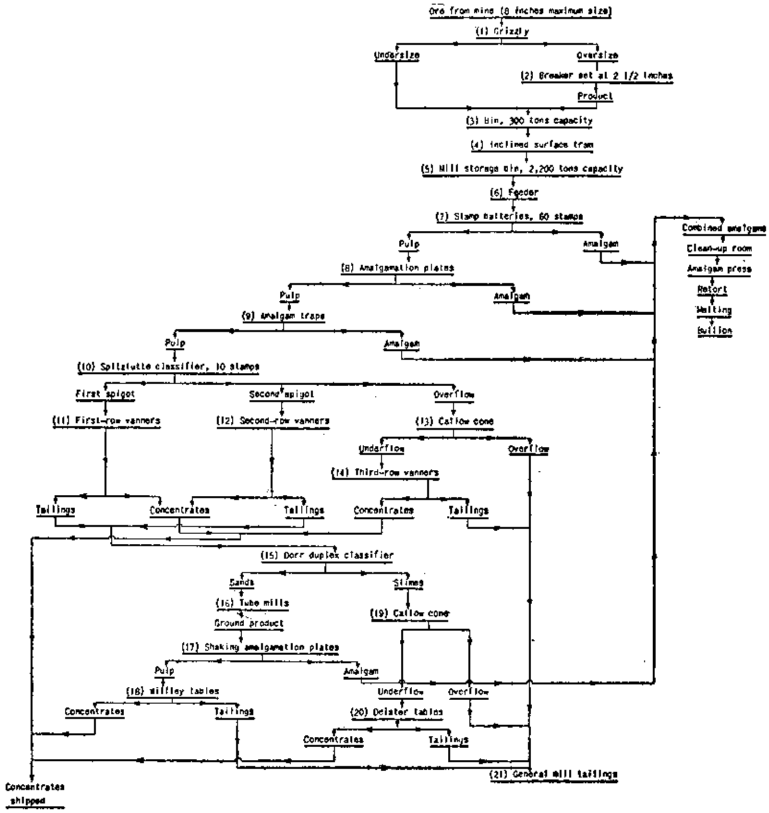

FIGURE 47.—Flow sheet of Argonaut mill

mating plates formerly used in the Sixteen to One mill. The first tables are cleaned up daily and the second set at intervals of 2 to 4 weeks, the heavy concentrate being amalgamated in Berdan pans.

The Tightner mill, operated on ore containing rather finely divided gold, involves fine crushing by means of two 5-stamp batteries of 1,250-pound stamps fitted with 30-mesh screens and amalgamation on lip plates and 5 by 6 foot outside plates. Riffled tables (3 by 6 feet) and traps follow the plates, and the pulp passing the traps is treated on a Wilfley table. A Berdan pan is used for the clean-up.

ARGONAUT MILL, JACKSON, CALIF.[23]

Flow sheet.—Figure 47.

Ore.—Quartz, some greenstone, and slate, carrying coarse and fine free gold with auriferous pyrite and some galena, sphalerite, chalcopyrite, tetrahedrite,

[23] Woodworth, Selim E., Milling Methods and Costs at the Argonaut Mill, Jackson, Calif.: Inf. Circ. 6466, Bureau of Mines, 1931, 12 pp.

pyrrhotite, and arsenopyrite. Fine grinding is not necessary; 6 or 7 per cent of the gold is lost in carbonaceous or graphitic slimes.

Tonnage.—Capacity, 300 tons per 24 hours; average treated, 242 tons per 24 hours.

Power.—Electric, purchased at $0.008744 per kilowatt-hour.

Water supply.—Purchased at $0.20 per miner's inch delivered to reservoir, which supplies mill by a 3,000-foot inverted siphon.

Assays (1929).—	Per ton
Mill heads	$6.07
Concentration heads	2.25
Vanner concentrates	64.87
Tailings	.68

Extraction (1929).—	Per cent
By amalgamation	62.93
By concentration	25.86
Total mill extraction	88.79

Cost (1929).—$0.96 per ton (includes $0.38 for treatment of concentrates).

Labor (1929).—18.9 tons per man-shift; 0.423 man-hour per ton.

Supplies.—		Per ton milled
Mercury (1929)	troy oz	0.113
Stamp dies (ave., 1915–1929)	pound	.282
Stamp shoes (ave., 1915–1929)	do	.422
Water (1929)	net gallons	1,425
Power (1929)	kilowatt-hour	19.50

Summary.—Stamp milling, amalgamation, and gravity concentration. Concentrates and tailings cyanided in another plant. Stamps crush to minus 24 mesh, inside and outside amalgamation being practiced. Plate tailings classified in spitzlutten and concentrated on vanners. Vanner tails classified in Dorr classifier and sands reground in tube mills followed by shaking amalgam plates and Wilfley tables. Spitzlutten overflow thickened and concentrated on vanners; Dorr classifier overflow thickened and concentrated on Deister slime tables. Tables only used when tailings are high enough to warrant expense; only vanners used in concentration since 1928.

CENTRAL EUREKA MILL, SUTTER CREEK, CALIF.[24]

Ore.—Free gold in quartz, auriferous pyrite and minor arsenopyrite and galena; also mineralized greenstone ("gray ore") and gouge. Gangue contains slate, greenstone, and gouge minerals.

Tonnage.—160 tons per day.

Power.—Electric, purchased. Approximately 0.9 cent per kilowatt-hour.

Assays.—	Per ton
Mill heads	$4 to $20
Concentrates	70 to 90

Extraction.—	Per cent of total recovery
Amalgamation	85
Concentration	15

Summary.—Amalgamation and gravity concentration. Forty-stamp mill treats ore by amalgamation in the batteries and on outside plates. Plate tails are concentrated on Frue vanners. Concentrates are cyanided locally under contract.

HOMESTAKE MINING CO., LEAD, S. DAK.[25]

Flow sheet.—Figure 48 (South mill).

Ore.—Altered dolomitic limestone with cummingtonite, chlorite, and stringer quartz, pyrite, aresenopyrite, and pyrrhotite. Gold is 75 per cent in native state, mostly finely disseminated. Gangue contains considerable ferrous iron.

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[24] Spiers, James, Mining Methods and Costs at the Central Eureka Mine, Amador County, Calif.: Inf. Circ. 6512, Bureau of Mines, 1931, pp. 1–2.

[25] Clark, Allan J., Milling Methods and Costs at the Homestake Mine, Lead, S. Dak.: Inf. Circ. 6408, Bureau of Mines, 1931, 22 pp. Clark, A. J., Metallurgy (of the Homestake Enterprise): Eng. and Min. Jour., Oct. 12, 1931, pp. 298–304.

Tonnage.—Combined capacity of three stamp mills, 3,700 to 4,300 tons per day.

Power.—Electric, from two company-owned hydro plants supplemented by company-owned coal-fired steam plant.

Assays (1929).—

South mill heads: \$4.90 per ton (all mills, approximately \$5.00).
South mill sand tails (heads to sand cyanidation) \$1.94 per ton.
South mill slime tails (heads to slime cyanidation) \$1.24 per ton.

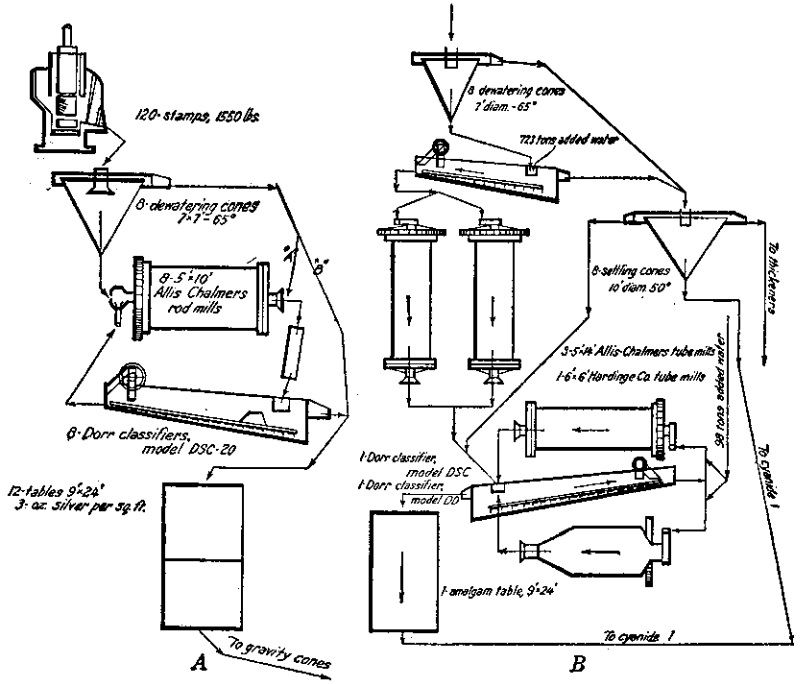

FIGURE 48.—Flow sheet of South mill, Homestake mine: *A*, Coarse grinding, 1927; *B*, fine grinding, 1927

Extraction.—

	Per cent
Recovery by amalgamation	62.8
Recovery by sand cyanidation	20.6
Recovery by slime cyanidation	9.6
Total recovery	93.0
Loss in sand tails	4.6
Loss in slime tails	1.6
Loss in slime wasted	.6
Total loss	7.0

Costs (1929).—

	Per ton
Crushing	\$0.038
Milling	.276
Cyaniding	.189
Total	.503

Labor and supply consumption (1929).—

South stamp mill:

Labor—

Tons stamped per man-hour	26.1
Tons ground (rod mills) per man-hour	25.8

Labor and supply consumption (*1929*).—Continued.

South stamp mill—Continued.	
Power—	Per ton
Per ton stamped .... kw-h..	3.892
Per ton ground .... do....	6.805
Stamp shoes .... lb. per ton..	.157
Stamp dies .... do....	.166
Rods .... do....	1.30
Liners	.364
No. 1 cyanide sand plant:	
Labor .... tons per man-hour..	9.5
Power .... kw-h. per ton treated..	1.033
Lime .... lbs. per ton treated..	2.74
Cyanide .... lb. (49 per cent NaCN) per ton..	.3885
Zinc .... lb. per ton..	.0303
Gold precipitated per pound zinc .... dollars..	64.94
Solutions per ton treated—	
Strong solution .... ton..	.235
Weak solution .... do..	.224
Low solution .... do..	.145
Total solution .... do..	.604
Slime cyanide plant:	
Labor .... tons per man-hour..	7.0
Power .... kw-h. per ton treated..	1.77
Lime .... lbs. per ton treated..	4.178
Cyanide .... lb. (49 per cent NaCN) per ton..	.533
Zinc .... lb. per ton..	.086
Gold precipitated per pound zinc .... dollars..	13.51
Acid used per ton treated .... lb..	.44
Solutions per ton treated—	
Strong solution	.31
Weak solution	.64
Low solution	.33
Total solution	1.28

Summary.—Amalgamation, followed by batch cyanidation. Ore is crushed underground in jaw crushers and reduced on surface to minus 3-inch size in gyratory crushers. This material is further reduced by stamp mills in three plants. (South mill, 120 stamps weighing 1,570 pounds each, capacity 1,900 to 2,200 tons per day; Amicus mill, 240 stamps weighing 1,100 pounds each, capacity 1,200 to 1,400 tons per day; Pocahontas mill, 160 stamps weighing 950 pounds each, capacity 600 to 700 tons per day.) Stamps are used to cope with an irregular size feed of hard slabby ore which is too coarse for ball mills. This use as secondary crushers is more economical under the conditions than would be the case if rolls were substituted.

Pocahontas mill stamps to minus 0.024 inch and amalgamates low-grade and oxidized ore, tailings passing to sand leaching at No. 2 plant. Amicus mill, stamping to minus 0.044 and 0.030 inch, employs inside and outside amalgamation. Tube mills in circuit with Dorr classifiers regrind sands for treatment on second set of amalgam plates. Ore is richer than at other mills and is finer ground. South mill uses stamps as crushers only; these discharge to dewatering cones, whence underflow is reground in rod mills and Dorr classifiers and sent with cone overflow to amalgamation plates. Plate tailing is dewatered in cones, reground in tube mills, and passed over another plate. Mercury is fed to rod mills but not to stamps. Sumps, launders, etc., act as traps. An amalgam plate is placed in launder between rod mill and classifier. Plates are dressed daily.

Cyaniding is entirely by batch methods in two sand plants and one slime plant. No grinding is done in solution. Chief problem consists of providing ample oxygen to offset consumption by ferrous compounds. Close classification is necessary to prevent blinding and channeling of charges. Aeration precedes cyanidation to render cyanicides harmless. Alkalinity is kept low. Sand and slime are separated in cones and thickeners. Underflow from two sets of gravity cones and one set of hydraulic cones, all in series, constitutes sand plant feed. Overflow goes to thickeners.

Sands are leached at two plants in redwood vats having usual wooden lattice, coco matting, and canvas filter bottoms. Feed is charged by Butters-Mein distributors along with lime and then drained, aerated from below under low pressure, and leached three successive times before washing. First leach is with barren solution; this is strengthened and used again for second leach, thus concentrating dissolved values in less solution. Third leach is with weak solution. Final tailing is sluiced out with water. A typical cycle requires a treatment period of about 154 hours. Slimes are thickened after overflowing from cones (as stated above) and piped 3½ miles to slime plant. Here the pulp is treated by batch leaching in Merrill presses. As with sands, cycle involves draining, aeration, and leaching by double passage of solution through charge, followed by washing and sluicing out of tails. Treatment time ranges from 7 to 10 hours, depending on amount of oxidized colloidal material present. Precipitation at all three plants is by the Merrill-Crowe zinc dust process. Celite is added with the zinc dust as a filter aid. Vacuum deaeration is practiced before precipitation, and slime pregnant solutions are clarified in a Merrill press. Sand solutions are naturally clear. Refining involves melting on a lead both to remove base impurities, remelting bullion in graphite crucibles and casting dorè bars.

ALASKA-JUNEAU CONCENTRATOR, JUNEAU, ALASKA [26]

Flow sheet.—Figure 49.

Ore.—Quartz stringers in slate and metagabbro. Gold mostly native and quite coarse, usually associated with small amounts of galena and sphalerite. Pyrrhotite and some pyrite are most abundant sulphides. Chief gangue material is metagabbro, which is hard and tough and breaks into slabs. Run-of-mine ore is mixture of fines and coarse material, and is usually very wet.

Tonnage (first nine months, 1929).—10,718 tons mined per day; 5,078 tons waste rejected per day; 5,640 tons fine milled per day.

Power.—Electric, from three company-owned hydro plants; during water shortage additional power is purchased from Alaska Gold Mining Co. Oil-fired steam plant maintained as stand-by. Power cost, 1929, $0.0288 per ton trammed.

Water supply.—From Gold Creek, via gravity flume and pipe line. Salt water pumped from Gastineau Channel during very cold weather. Consumption, 8,000 to 8,500 gallons per minute, none reclaimed.

Assays (first nine months, 1929).—	Per ton
Run of mine	$1.100
Sorted mill heads	1.919
Shipping concentrates	324.523
(Plus 60 per cent lead and 35 ounces silver.)	
Sorted waste	.192
Mill tails	.277
Total tails	.237

Extraction.—	Per cent
By sorting	91.74
As bullion	60.82
As shipping concentrate	17.69
Total	78.51

Losses.—	
In sorted waste	8.26
In mill tails	13.23
Total	21.49

Operating costs (first nine months, 1929).—
$0.2363 per ton trammed.
$0.3162 per ton fine milled.

Summary.—Sorting and gravity concentration. About half the tonnage mined is sorted out after coarse crushing. The balance is crushed and ground in stages by means of cone crushers, rolls, ball mills, and tube mills (operated as ball mills) in circuit with screens of various types and fed to two sets of Deister sand tables, after rough classification in spigot and V-tanks. These

[26] Bradley, P. R., Milling Practice at the Alaska-Juneau Concentrator: Inf. Circ. 6236, Bureau of Mines, 1930, 16 pp.

tables make rough concentrates, middlings, and tailings to waste. The middlings go without further treatment to Wilfley tables that make a concentrate (which flows to reconcentrating tables), a middling, and a tailing to waste. Deister concentrates are screened, oversize joining the Wilfley middlings in a sump. Undersize is reconcentrated on a second set of Deister tables, which make a high-grade gold concentrate, a shipping concentrate, a middling (which is further tabled on Deisters before regrinding), and a tailing to the sump.

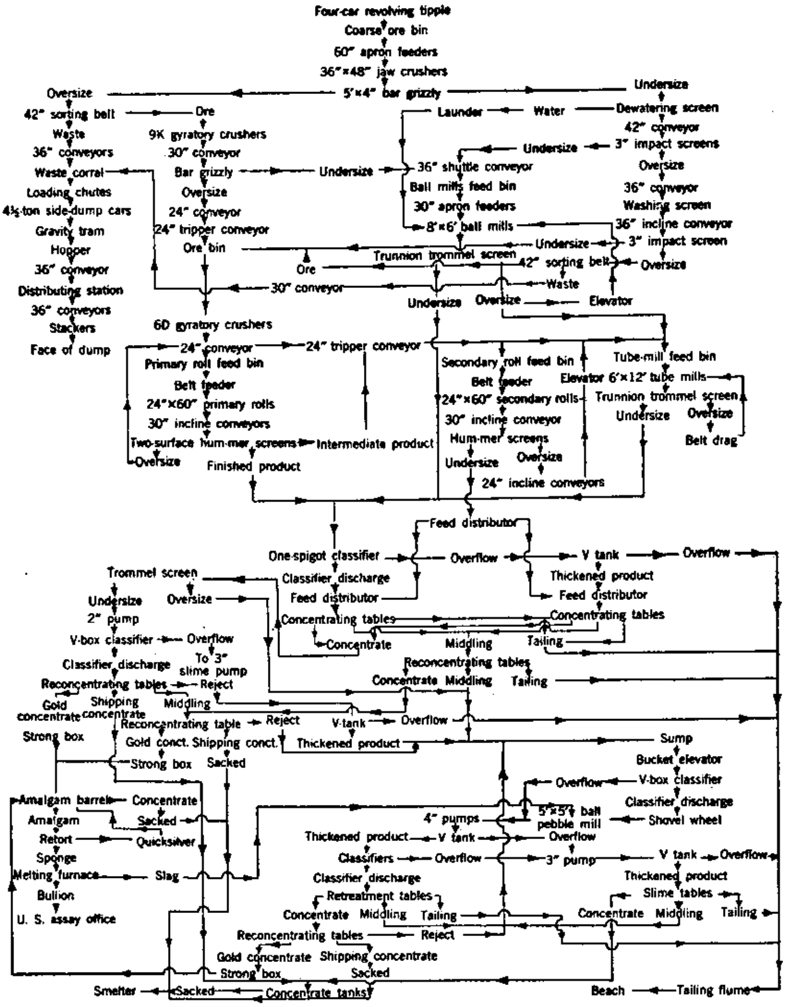

FIGURE 49.—Flow sheet of Alaska-Juneau mill

Material from the sump is reground in ball mills, partly thickened in V-tanks, and classified in cones. Spigot products are sent to Deister tables; these make a low-grade concentrate, a middling returned to the sump, and a tailing to flotation. Concentrates are retabled direct on Deisters, to give a high-grade gold concentrate, a shipping concentrate, and reject to the sump. Overflow from the last V-tank and classifiers is thickened, the overflow going to waste and the slime to Deister slime tables. These produce shipping concentrates, middlings to the sump, and tailings to flotation.

Flotation is done in a 2-cell M. S. sub-A machine, and concentration in the ratio of 80 to 100 to 1 produces a high-grade shipping product and a tailing to waste.

High-grade gold concentrates are amalgamated in a barrel; shipping concentrates, containing $350 gold and 60 per cent lead, are shipped in sacks to a smelter; tailings are flumed 3,000 feet to the beach.

MOUNTAIN COPPER CO., SHASTA COUNTY, CALIF.[27]

Ore.—Limonite-quartz gossan material, porous and friable. Contains 0.4 per cent copper, 50 to 55 per cent iron, 5 to 10 per cent silica, and a little arsenic and mercury. Crushed ore weighs 100 to 125 pounds per cubic foot. Gold very finely disseminated.

Tonnage (March, 1931).—543.4 tons per day.

Power.—Electric, purchased.

Assays (March, 1931).—

	Per ton
Mill heads	$1.654
Tails	.452

Extraction (March, 1931).—72.8 per cent.

Costs (March, 1931).—$0.432 per ton.

Labor.—Mill: 1 superintendent, 2 grizzly men, 2 crusher men, 2 tripper men, 2 vat men (filling), 6 vat unloaders, 3 solution men, 1 tailings man. For both mill and quarry: 2 mechanics, 1 assayer, 1 helper, 1 electrician, and 1 roustabout.

Supplies (March, 1931).—

Cyanide (NaCN)	pounds per ton ore	0.344
Lime	do	8.35
Zinc dust	do	.0257
Do	pounds per ton solution	.0382
Lead acetate (1 pound per shift)	pounds per ton solution, about	.0075
Power	kw-h. per ton ore	3.06

Summary.—Ore is mined by power shovel and hauled to the mill in trucks. It is crushed in a jaw breaker and two sets of rolls, with ½-inch screen, to ½-inch size. Crushed ore is conveyed by means of a 20-inch belt to vats (10 in number, each 25 feet 8 inches diameter by 11 feet effective inside depth and 276 tons capacity); a Jeffrey autotripper discharges the ore into the vats, and short spouts distribute it in even layers, to avoid coning and consequent solution channeling. In summer, when ore is dry, charge is moistened as each 1½-foot layer is deposited, to avoid segregation. One vat is filled (7 hours) and one discharged (by three men) on each of two shifts per 24 hours. Bottoms have coco matting and canvas lining with shoveling cleats and four discharge doors. Discharge goes to conveyor belts and tailings dump. Charge is leached in four periods as follows:

1. Strong solution (added from bottom), 0.8 pound NaCN, about 16 hours, including 5 hours standing period.
2. Weak solution (from top), 0.5 pound NaCN, about 24 hours.
3. Barren solution (from top), 0.4 pound NaCN, about 20 hours.
4. Water wash (from top), about 7.5 hours.

Total solution and washing, less than 72 hours.

Solutions used per 276-ton charge are:	Tons
Strong	195
Weak	90
Barren	180
Water wash	75

Pregnant solution is clarified through sand and precipitated by the Merrill-Crowe process. Lead acetate is added to prevent fouling and assist precipitation. Refining is by customary fluxing. A condenser is installed and recovers

[27] Young, George J., Cyaniding Low-Grade Ore: Eng. and Min. Jour., vol. 131, June 22, 1931, pp. 561–563.

40 pounds of mercury at each monthly cleanup. Good precipitation is partly due to presence of this quicksilver. Bullion fineness is 455½ parts gold and 180 silver per 1,000.

Successful cyanidation of this oxidized, copper-bearing material is partly due to fineness of the gold particles, porosity of the ore, and use of very dilute solutions. Low costs are obtained by bulk leaching, with careful vat filling to prevent channeling. Finer crushing to decrease tailings loss is planned.

CONIAURUM MINES (LTD.), SCHUMACHER, ONTARIO.[28]

Flow sheet.—Figure 50.

Ore.—Quartz and mineralized schist. Mill heads contain a large proportion of very wet fines.

Tonnage.—Capacity, 500 tons per day; 122,972 tons were milled in 1930.

Power.—Electric, purchased.

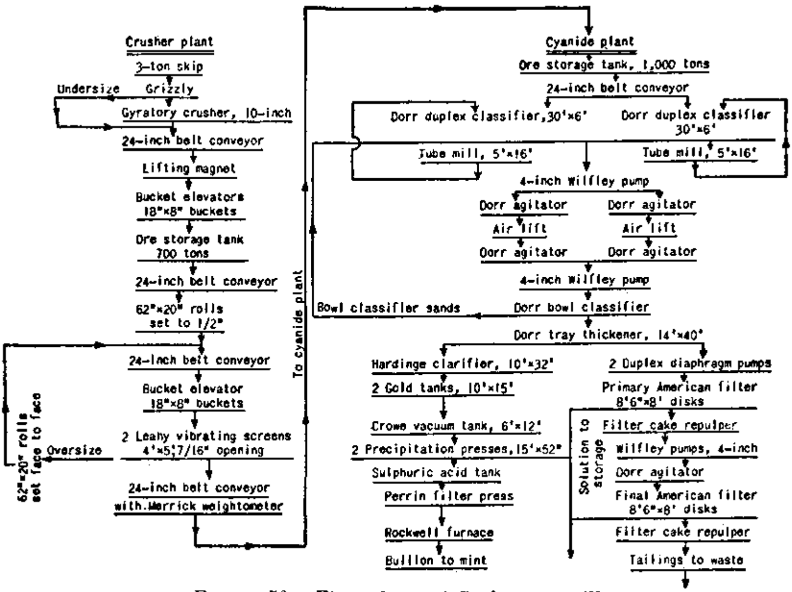

FIGURE 50.—Flow sheet of Coniaurum mill

Assays.—

		Per ton
Heads	(approximate)	$6.00
Tails	(estimated)	.22

Extraction.—96.35 per cent.

Operating cost (1930).—$0.939 per ton.

Supplies.—

Cyanide	lb. per ton	0.5
Lime (solutions kept at 0.75 lb. CaO per ton)		
Zinc dust (650 tons solution per 24 hours)	lb. per ton solution	1.5
Lead nitrate	lbs. per 24 hours	1.5
Grinding balls	lbs. per ton ore	1.8
Vibrating screens, life	weeks	6
Filter bags, life	days	130

Summary.—All cyanide, continuous agitation. Ore is crushed to 1¾-inch size in a gyratory, reduced in two sets of rolls to all minus 4 mesh and 72 per cent plus 28 mesh, and ground in cyanide solution by means of tube mills in

[28] Redington, John, Milling Methods and Costs of the Coniaurum Mines (Ltd.), Schumacher, Ontario: Inf. Circ. 6541, Bureau of Mines, 1931, 5 pp.

closed circuit with Dorr classifiers. Classifier overflow, containing 75 per cent of the gold already in solution, is given two successive treatments in Dorr agitators connected in series by air lift. Pulp passes from the last agitator to a bowl classifier, which acts as a concentrator in that heavy sulphides receive selective grinding; sands return to the tube mill, and overflow (60 per cent minus 200 mesh) is thickened and filtered on an American filter. A barren solution wash is given the cake on the filter; it is then repulped in barren solution and pumped to a Dorr agitator, whence it is sent to a second filter. Here a water wash is applied, cake is repulped and run to waste, and solution is pumped to storage, to precipitation, or into the circuit at various points as desired. Pregnant solution from the Dorr thickener is clarified in a Hardinge sand clarifier, lead nitrate is added, and solution is pumped to Crowe vacuum deaeration and thence to zinc dust precipitation in Perrin presses. Precipitate is cleaned up monthly, given a sulphuric acid wash, and refined in the usual manner. The chief trouble is occasioned by the wet fines, which cause some difficulty in the rolls.

KIRKLAND LAKE GOLD MINES (LTD.), KIRKLAND LAKE, ONTARIO [29]

Flow sheet.—Figure 51.

Ore.—Silicified and altered lamprophyre, quartz porphyry, syenite and diabase. Gold (native and in pyrite) is very finely divided in gangue. Some tellurides and a little silver present. Ore is tough and hard.

Tonnage.—Average, 145 tons per day; maximum, 200 tons.

Power.—Electric, purchased.

Water supply.—Mine drainage, plus purchased municipal water.

Assays (1930).—	Per ton
Heads	$11.38
Tailings	1.22

Extraction (1930).—89.27 per cent.

Operating cost (1930).—$1.389 per ton.

Labor.—20.7 tons per man-shift; 0.386 man-hour per ton.

Supplies.—		Per ton
Cyanide (50 per cent NaCN)	pounds	1.383
Lime	do	5.988
Zinc	do	.0705
Lead nitrate	do	.0099
Ball consumption	do	4.441
Liner consumption	do	.497
Water consumption	net gals	1,000

Summary.—All slime cyanidation by continuous countercurrent decantation. Grinding in solution is done in an open-circuit Hardinge ball mill, followed by a tube mill in closed circuit with a Dorr classifier and a classifying bowl. Overflow from the Dorr classifier (60 per cent minus 300 mesh) goes to the primary thickener (see flow sheet), while bowl overflow (96 per cent minus 300 mesh) goes to secondary agitation and decantation. This very fine grinding is necessary because of the intimate interlocking of gold and gangue; all the gold is not freed, but finer grinding is not economical. The pulp passes through three agitators and five thickeners and is then filtered on an Oliver filter, being washed with barren solution and then with water. Aeration is very important, and is promoted not only by the air lifts of the Dorr agitators, but chiefly by allowing the mill solution to plunge 12 feet through the air into the storage tank. Kieselguhr diffusers in the barren solution and storage tanks and in the thickener overflow are also effective. Sodium and barium peroxide are of no assistance.

Pregnant solution is clarified in a Butters-type filter, deaerated by the Crowe vacuum system, and precipitated with zinc dust in Perrin presses. Fouling

[29] Dixon, John, Milling Practice of the Kirkland Lake Gold Mines (Ltd.), Kirkland Lake, Ontario: Inf. Circ. 6508, Bureau of Mines, 1931, 13 pp.

of solution is largely prevented by adding 1½ pounds of lead nitrate per day to No. 1 agitator; this reagent also promotes good precipitation. The mill is operated by 1 solution man and 1 ball-mill man per shift, and 1 crusher man per day.

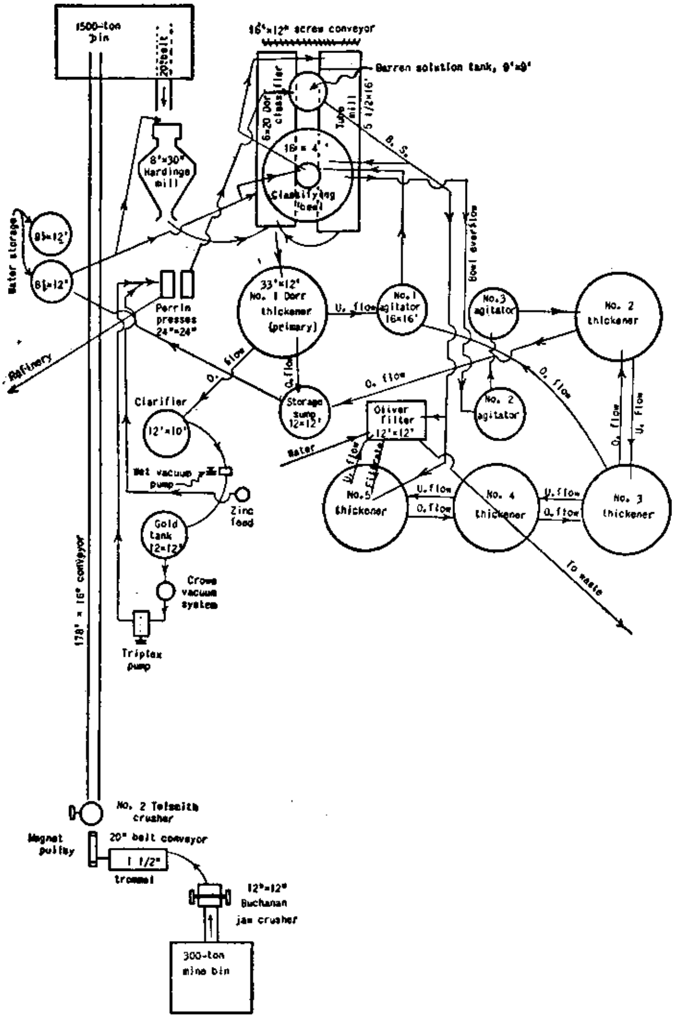

FIGURE 51.—Mill flow sheet, Kirkland Lake Gold Mines (Ltd.)

TECK-HUGHES MILL, KIRKLAND LAKE, ONTARIO [30]

Flow sheet.—Figure 52, cyanide plant.

Ore.—Altered, silicified syenites, lamprophyre, and porphyry with 10 to 20 per cent quartz, 2 per cent sulphides, and some tellurides. Gold, chiefly native, associated intimately with sulphides. Clay and colloids practically absent. Ore is medium hard.

Tonnage.—1,270 tons per day (270 tons in old mill, balance divided between 3 units of new mill).

Power.—Electric, purchased. Consumption, 25 kw-h. per ton.

[30] Read, Herbert N., Milling and Cyaniding Practice at the Teck-Hughes Gold Mines (Ltd.): Canadian Min. Jour., vol. 52, December, 1931, pp. 839–846.

Assays.—	Per ton
Heads	$15.00
Bowl-classifier overflow	4.50
Primary-agitator overflow	2.81
No. 4 secondary-agitator overflow	1.08
Final tails	.89

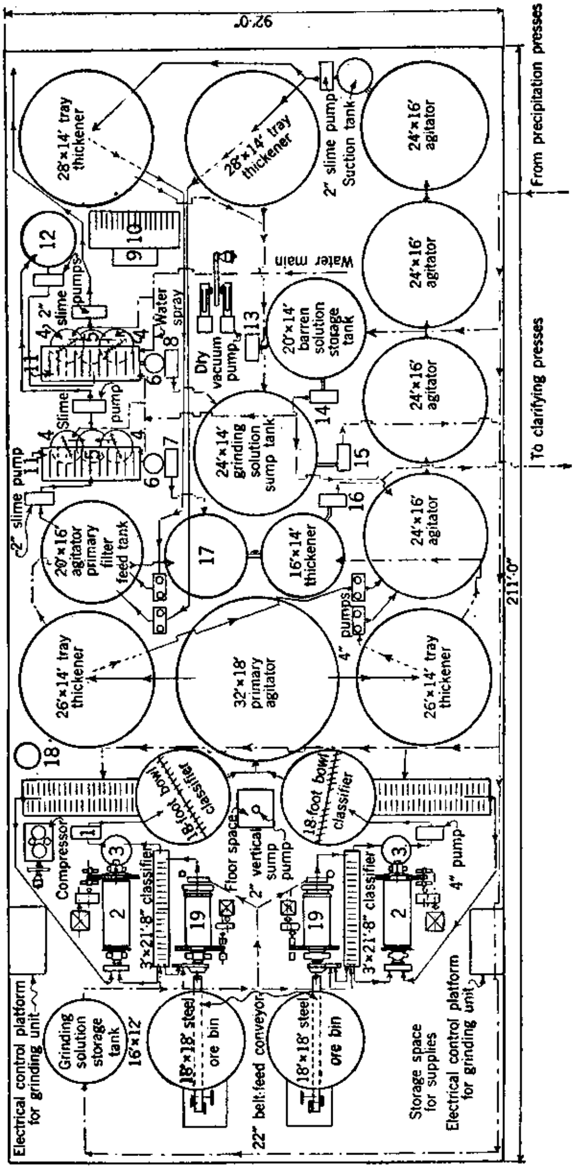

FIGURE 52.—Flow sheet of cyanide plant, Teck-Hughes Gold Mines (Ltd.): *1*, 4-inch pump; *2*, 2.5 by 16 foot tube mill; *3*, tank; *4*, emulsifier; *5*, emulsified-pulp tank; *6*, filtrate receiver; *7*, primary filtrate pump; *8*, secondary filtrate pump; *9*, acid treatment; *10*, washing layout; *11*, 8-foot 6-inch disk filter; *12*, 10 by 10 foot secondary filter-feed tank; *13*, barren-solution pump for washing floors; *14*, barren-solution pump for filter sprays; *15*, grinding-solution pump; *16*, rich-solution pump; *17*, 16 by 14 foot thickener-overflow solution tank; *18*, air receiver. *19*, 4 by 10 foot rod mill

Extraction.—	Per cent
In grinding and classifying circuit	70.00
In agitation and thickening circuit	24.13
In filtering circuit	.06
Total	94.07

Operating cost (year ended Aug. 31, 1931).[31]—$1.14 per ton ore before depreciation and general expense.

Supplies.—

Cyanide (49.5 per cent NaCN)	lbs. per ton ore	1.5
Lime (90 per cent CaO)	do	5.0
Zinc dust	do	.5
Zinc dust	ounce per ton ore (approx.)	1.7
Rod consumption (0.9 per cent C; 0.9 per cent Mn)	lbs. per ton ore	1.5
Rod-mill liner consumption (Mn steel)	do	.21
Tube-mill balls (1½-inch, forged steel)	do	3.0
Tube-mill liner consumption (white iron)	do	.22

Summary.—All slime cyanidation by continuous agitation. See flow sheet (fig. 52). Ore after crushing in jaw and cone or gyratory crushers to minus three-fourths inch is ground in rod and tube mills. Rod mills are in closed circuit with Dorr simplex classifiers. Tube-mill discharge and Simplex-classifier overflow go to Dorr bowl classifiers, from which sands return to tube mill, and overflow (25 per cent solids) goes to primary agitation. Pulp at this stage is 98 per cent minus 200 mesh, and much of it is near 400 mesh. Considerable concentration of sulphides and consequent selective grinding are accomplished by means of bowl classification. Primary agitation permits removal of much gold in thickener overflows ahead of filtration and provides oxygen so that dissolution takes place in the thickeners. After thickening, overflow goes to precipitation, while the pulp passes to four agitators in series. Here solution is added to give 70 per cent liquid; air is provided in the usual air lift and also at two other points in the agitator bottoms. A total of 1.8 cubic feet per ton is used at 18 pounds pressure. Pulp next goes to tray thickeners, overflow going to grinding-solution storage tank and underflow being filtered in two stages on American filters. Last filtrate goes to solution storage, and first filtrate, with primary thickener overflow, passes to Merrill clarifying filter presses, Crowe vacuum deaeration, and Merrill zinc-dust precipitation. Clean-up, refining, and so forth are done according to usual practice. Outstanding features are very fine grinding to unlock values, large volumes of air used to promote dissolution of gold and break down tellurides, careful classification in grinding circuit, and high dilution of pulp. Mill is operated as three parallel units of like design, with the old mill as a fourth parallel unit.

HOWEY MILL, RED LAKE, ONTARIO[32]

Flow sheet.—Figure 53.

Ore.—Porphyry mineralized with quartz veins and stringers, pyrite, and sparse other sulphides. Gold is finely and evenly distributed.

Tonnage (1931).—Capacity 600 tons per day.

Power.—Hydroelectric, transmitted 40 miles.

Assays.—	Per ton
Mine run (August, 1931)	$4.65
Mill heads (some waste sorted)	5.15
Tailings (estimated)	$0.22 to $0.26

Extraction.—95 per cent or more.

Operating cost (based on 15,500 tons per month, 1931).—$0.97 per ton.

Summary.—All cyanide agitation leaching. After crushing to 3-inch size, some waste is sorted out by hand. The ore is then reduced in a cone crusher and ground in solution to 40-mesh size by Hardinge ball mills in closed circuit with Dorr classifiers. A Dorr bowl classifier is provided for optional use and permits selective grinding of sulphides by retaining them in the grinding circuit longer than the lighter gangue. The ground pulp is thickened in Dorr thickeners, agitated in Dorr agitators, and sent to Genter thickeners. Overflow from both sets of thickeners passes through clarifying presses and Crowe vacuum deaeration to zinc-dust precipitation and refining. Genter thickener underflow is filtered on American disk machines. The coarse pulp is successfully handled in the thickening and agitation equipment with only minor mechanical departures from usual practice.

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[31] Seventeenth Annual Report, Teck-Hughes Gold Mines (Ltd.).

[32] Goodwin, W. M., Cyaniding 40-Mesh Pulp at Red Lake, Ontario: Eng. and Min. Jour., vol. 132, Aug. 10, 1931, pp. 109–110.

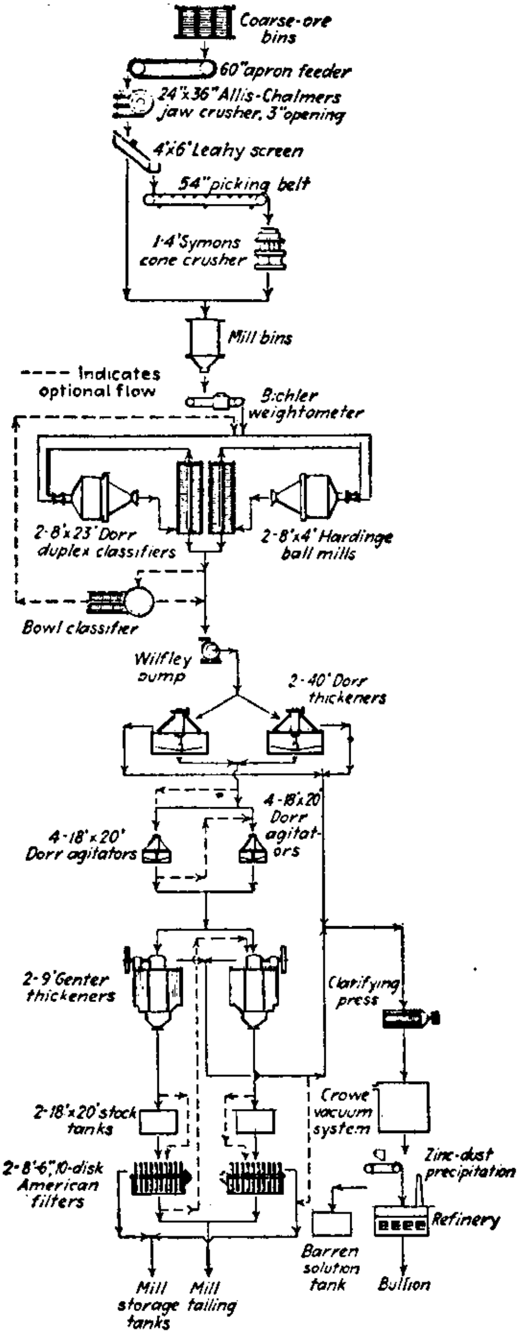

FIGURE 53.—Flow sheet of the Howey metallurgical plant. (After W. M. Goodwin, Eng. and Min. Jour., Aug. 10, 1931, p. 110)

SPRING HILL CONCENTRATOR, HELENA, MONT.[33]

Flow sheet.—Figure 54.

Ore.—Gold associated with pyrite, arsenopyrite, antimony, and bismuth minerals; 10 or 15 per cent is native and very finely divided. Gangue rock is hard, tough diorite. Pyrrhotite, barren of gold, is the most abundant sulphide.

Tonnage (July 1, 1929, to March 31, 1930).—153 tons per 24 hours.

Power.—Electric, purchased. Total power cost, $0.008 per kilowatt-hour.

Water supply.—Pumped from near-by stream; in dry periods 20 per cent of the requirement is reclaimed from tailings.

Assays (July 1, 1929, to March 31, 1930).—

	Per ton
Heads	$6.46
Concentrates (calculated)	63.57
Tailings	1.22

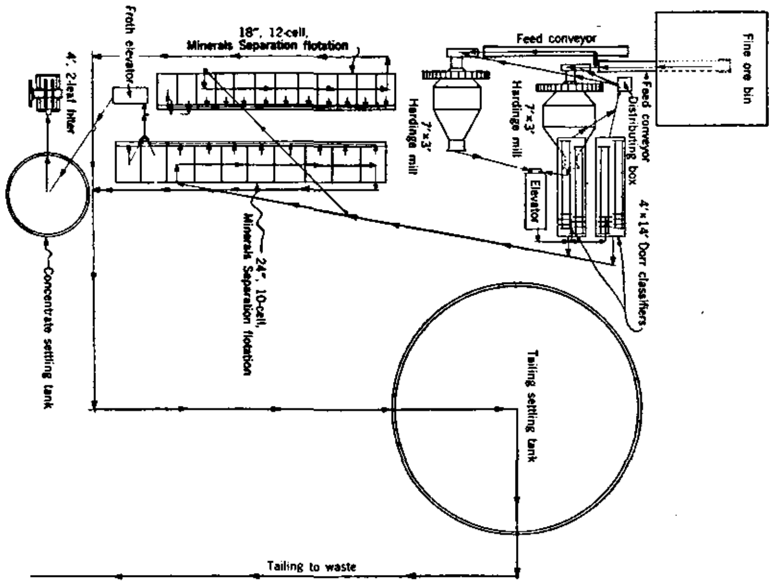

FIGURE 54.—Flow sheet of concentrator, Spring Hill mine

Extraction (July 1, 1929, to March 31, 1930).—82 per cent.

Operating cost (July 1, 1929, to March 31, 1930).—$1.007 per ton milled.

Supplies.—

		Per ton ore
New water	gallons	104
Grinding balls	pounds	1.30
Reagents—		
Pine oil	pound	.075
Potassium ethyl xanthate	do	.109
Aerofloat	do	.103

Summary.—Bulk flotation. Original flow sheet employed cyanidation but was abandoned after about one year's operation due to fouling of solutions by antimony and arsenic and consequent poor recovery. The ore is crushed in two successive jaw crushers and a cone crusher, being reduced to 6.7 per cent plus nine-sixteenths by 1 inch size. The fine ore is ground in Hardinge ball mills in closed circuit with Dorr classifiers, which deliver an overflow (23 per cent solids, 59.4 per cent minus 200 mesh) direct to M.S. flotation machines

[33] Grant, L. A., Milling Methods and Costs at the Spring Hill Concentrator of the Montana Mines Corporation, Helena, Mont.: Inf. Circ. 6411, Bureau of Mines, 1931, 8 pp.

where pulp is diluted to 19 per cent solids. These make a rough concentrate and a final tailing in all but the first two cells; the concentrate is cleaned in cells Nos. 1 and 2 and is pumped to a thickener. Concentration ratio is 12 to 1. Reagents are fed to the grinding circuit.

Thickener underflow is filtered on an American filter, and overflow is returned to flotation. Concentrates are shipped and tailings are laundered 500 feet to a pond. Fourteen men constitute the crew, exclusive of salaried staff.

DISCUSSION OF MILLING METHODS

HAND SORTING

The fields of use to which the various milling methods are applicable have been taken up in some detail in the foregoing pages. To provide a clear basis for comparison of these methods from the standpoints of metallurgical efficiency and cost of production a summary of their respective scopes of application is presented below.

HAND SORTING

Improving the grade of material sent to fine crushing by hand sorting is justified if the ore is amenable to such treatment and if the recoverable values in the rejected waste, plus the cost of sorting, are less than the cost of milling the waste. Sorting is important at many low-grade properties and is often beneficial at others treating rich ore.

AMALGAMATION

Amalgamation is used for ores of the free-milling type. It is peculiarly adapted to small-scale operations because of relatively low plant cost, simplicity, flexibility, and cheapness of operation in small units, and good recovery from variable grade and volume of feed. In larger mills treating ores containing appreciable amounts of coarse, free gold, amalgamation is usually desirable, unless the pulp carries substances that tend to sicken the mercury or unless the advantages of grinding in cyanide solution are particularly manifest. Gold recovered early on the plates is more cheaply recovered by amalgamation (or on blanket tables) than by other methods, danger of subsequent loss is minimized, and total production costs per ounce of metal are lowered. Removal of coarse gold by this means shortens the agitation period required in cyanide treatment or may prevent loss of coarse gold in a flotation plant adjusted to recover fine gold. When only small amounts of gold are amalgamable the use of plates complicates the flow sheet and is only justified as a device for catching "tramp" coarse gold which might otherwise be lost.

GRAVITY CONCENTRATION

Gravity methods of concentration are most applicable to ores in which the gold-bearing minerals are released from the gangue at relatively coarse sizes. They are used at many old mills where the capital cost of changing to flotation would not be justified and at some recent plants where the ore is not suited to flotation.

Shaking tables of the Wilfley and Deister types perform most satisfactorily on well-classified feed and require considerably more floor space per unit of output than for flotation. They are relatively inefficient on slime material.

The use of riffle or blanket tables (strakes) in lieu of amalgamating plates is widespread on the Rand and is gaining favor in some American camps. These devices are cheap to install and operate and, when followed by cyanidation, over-all results are often as satisfactory as with plates. Blankets are also widely used as scavengers after plates or concentrating equipment and in tailing sluices.

FLOTATION

In recent years this process has made great strides and is becoming more and more firmly established in the field of gold milling. The method is applied in a variety of ways. It may be used to make a rough concentrate and a clean tailing at rather coarse sizes (48 mesh or so) and thus eliminate a large tonnage of waste before finer grinding and further treatment of the concentrate are undertaken; it may be desirable as a means of saving small but worthwhile amounts of base metals which would be lost in straight cyanide treatment; it may be of great value as a means for removing, in a concentrate, such substances as arsenic or other materials deleterious to cyanidation, thus producing a tailing that can be economically cyanided; it may be adopted for treating refractory tailings, such as the carbonaceous slimes of the Mother lode; or it may simply prove a more economical method of treating nonrefractory ores. Flotation is in general applicable to ores in which the gold is associated with sulphide minerals or is of fine particle size and high grade, so that a stable froth can be maintained. Low-grade nonsulphide ores, according to Taggart,[34] are not ordinarily suited to the method because there is not enough metallic substance present to stabilize a froth.

CYANIDATION

Cyanidation is most applicable to sulphide or nonsulphide ores which are free from excessive amounts of cyanicides. Tellurides require a preliminary roast or very fine grinding in conjunction with excessive aeration or the use of bromocyanide or other special reagents. If considerable arsenic, antimony, or soluble copper, iron, or other cyanicides are present, the method becomes more complicated and expensive. Cyanidation is best suited to the treatment of large tonnages, since it requires a larger capital investment than other methods.

COMPARISON OF METHODS AND RECENT TRENDS

If an ore is wholly or partly free-milling and the gold is coarse the first step after grinding is generally plate amalgamation or recovery on riffle or blanket tables. Plate amalgamation will in general recover more gold than blanket tables, unless organic substances in the water or other materials promoting sickening are present. Blanket recovery, followed by barrel amalgamation, is somewhat cheaper, uses less quicksilver, and is not so dependent on the skill of the operators as plate amalgamation. If cyanidation or other

[34] Taggart, Arthur, work cited, p. 866.

treatment follows, blanket treatment may be preferable to the use of plates. At the 70-ton mill of Granada Gold Mines, Rouyn, Quebec, muskeg water caused fouling of the plates, and blankets were substituted with satisfactory results. Figure 42 is a flow sheet of the Granada mill. At the Sixteen to One mill in California, riffled tables have wholly replaced plates in the ball-mill plant.

With ores requiring cyanidation a strong tendency to do away with prior amalgamation has prevailed in some districts until rather recently. The greatest possible simplicity in the flow sheet was sought, and much importance was attached to grinding in solution. The elimination of plates in many if not most of such cases was probably sound engineering, but some tendency in the reverse direction is nevertheless apparent at present. At the Dome mill in Ontario the original flow sheet involved amalgamation followed by cyanidation, but later the mill was changed over to an all-cyanide plant. It is now planned to return to the principle of recovering the coarse gold ahead of cyanide treatment, and blanket tables are to be installed. This is in accord with experience on the Rand.

When shaking tables are used after amalgamation and ahead of cyanidation the concentrates, sand, and slime tailings are sometimes treated separately by cyanidation. Good results have been and are obtained in this way at Grass Valley and on the Mother lode in California. An alternative procedure developed in Ontario employs bowl classification to promote selective grinding and concentration of the sulphides within the grinding circuits; this eliminates the need of separate treatment of concentrates and makes for a simplified cyanide flow sheet and reduced plant investment.

Most ores which are amenable to gravity concentration are also suitable for treatment by present-day flotation methods which generally give superior metallurgical results at less cost, and a marked tendency is noticeable in the direction of substitution of flotation for gravity concentration. Gravity methods are still the best when appreciable amounts of gold can be liberated from the gangue and concentrated with relatively coarse grinding.

When the gold is finely disseminated in sulphides, very fine grinding is required for good cyanide extraction. Flotation may often make an equal or better recovery at much coarser sizes, since the sulphide particles can be floated when the gold is not exposed as it must be for cyanidation. An interesting development is noted at the McIntyre property in the Porcupine district. It was found that most of the gold could be concentrated into 15 per cent of the original tonnage by bulk flotation at 48 or 65 mesh; the concentrate was then amenable to cyanide treatment after regrinding. A new mill has been built which eliminates by this means much grinding expense. Furthermore, the cyanide treatment is susceptible of accurate control, and much less plant space is required for a greater tonnage than was formerly handled in the all-cyanide mill.

A distinct advantage of flotation at some properties is the ability to recover valuable base metals. The Spanish mine is an example of a small property making use of this fact.

When arsenic or other cyanicides are present flotation is a good substitute for cyanide methods. At Spring Hill, Mont., cyanide losses from arsenic and antimony led to adoption of bulk flotation in

1929, with satisfactory results. Concentrates from this mill are smelted.

Recent developments noted by the Mines Branch, Ontario,[35] are: Study of the possibility of flotation of ores containing submicroscopic gold so intimately locked in sulphides that cyanide extraction is unsatisfactory; recognition of the fact that inasmuch as 70 to 80 per cent of the gold in many ores treated by cyanidation is dissolved in the grinding circuit, with most of the remaining gold extracted on the filters, further filtration might be substituted economically for the long agitation period; and progress in improving and applying methods for the regeneration of cyanide.

Many considerations enter into the choice of methods or combinations of methods. Thus we may have straight amalgamation or blanket treatment in small plants; amalgamation and concentration (by gravity, flotation, or both), followed by cyanidation of concentrates and tailings or smelting of concentrates and cyaniding of tails; cyanidation followed by flotation of tailings; flotation followed by smelting or cyanidation of concentrates and discard of tailings; flotation followed by cyanidation of concentrates or tailings or both; and so on.

From the standpoint of cost, blanket recovery and amalgamation are cheapest to operate and to install. Flotation is cheaper to operate and requires a less expensive plant than cyanidation. Gravity concentration at relatively coarse sizes requires a less expensive plant than flotation. Gravity concentration plus flotation involves greater plant investment than straight flotation; but an amalgamation plus vanner concentration plant, like those on the Mother lode, is somewhat cheaper than an all-flotation mill. All-cyanide slime plants usually cost nearly twice as much as straight flotation mills, per ton of capacity. Many exceptions occur, but the above generalizations give a broad picture of the relative plant investment for different milling methods.

Sound judgment based on long experience is necessary in the proper design of flow sheets. Huge sums have been wasted through failure of small companies to recognize this fact and to entrust the technical problems of milling their ores to men capable of correctly solving them.

COSTS OF MILLING

Gold-milling costs vary between wide limits in different districts, being determined as they are by a great number of variable factors. Costs obtained in recent years at a few typical mills in the United States and Canada are presented in Table 13. For purposes of comparison the costs for earlier years at a few mills are grouped in Table 14.

Because the controlling conditions, such as nature of the ore, location, tonnage treated, water, and power supplies are in no way comparable, deductions as to the relative efficiencies of milling plants are difficult or impossible to draw from the cost sheets alone. In general, the large-tonnage plants have the lower costs per ton treated.

[35] Staff, Division of Ore Dressing and Metallurgy, Mines Branch, Department of Mines, Ottawa, Some Problems in the Treatment of Gold Ores: Canadian Min. and Met. Bull. 235, November, 1931, pp. 1260–1261.

It will be seen that unit costs in Table 13 range from $1.85 per ton at a 25-ton mill to as low as $0.24 per ton at a plant handling over 10,000 tons per day. Labor accounts for roughly 40 per cent of mill operating costs, with supplies and power comprising 50 per cent or more.

In comparing the figures of Table 13 and Table 14, the great strides that have been made in reducing the cost of gold milling are at once apparent.

TABLE 13.—*Recent gold-milling costs in the United States and Canada*

Mill and year	Tons milled for period	Average tons per day	Cost per ton milled					Remarks
			Labor	Supplies	Power	Miscellaneous	Total	
Porcupine United, 1929 and 1930.		25	$1.072	$0.374	$0.405		$1.851	Amalgamation and table concentration.
Argonaut, 1929	89,684	246	.23	.10	[1] .19	[2] $0.44	.96	Amalgamation and gravity concentration.
Homestake, 1929	1,437,935	4,000					.503	Amalgamation, sand cyanidation, and slime cyanidation.
Alaska-Juneau, 1929 [3]	[4] 2,904,578	[5] 10,718	[6] .1081	[6] .0994	[6] .0288		[6] .2363	Sorting, gravity concentration, and barrel amalgamation.
Coniaurum, 1930	122,972	[7] 337					.939	All slime continuous agitation.
McIntyre, year ended Mar. 31, 1931.	558,115	1,530	.2108	.3953	.1302	[8] .0147	[9] .9274	Continuous countercurrent decantation plus 200-ton pilot flotation plant.
Vipond, 1930	113,281	[10] 310					1.19	All slime agitation.
Hollinger, 1930	1,625,868	4,479					.6355	All slime agitation and countercurrent decantation.
Lake Shore, year ended June 30, 1931.	698,624	[10] 1,914					.990	All slime agitation.
Teck-Hughes, year ended Aug. 31, 1931.	396,200	[11] 1,085					1.14	Do.
Kirkland Lake Gold Mines (Ltd.), 1930.	52,768	145	.4289	.5715	.3677	.0209	1.3890	All slime agitation and continuous countercurrent decantation.
Spring Hill, July 1, 1929 to Mar. 31, 1931.	40,930	153	.426	.374	.207		1.007	Bulk flotation concentrates shipped.
Mountain Copper	[12] 16,847	543	.182	.176	.031	.043	.432	Bulk leaching.

1 Power, lights, and water.
2 Includes trucking and contract cyanide treatment of concentrates, $0.41; insurance, $0.01; and miscellaneous, $0.02.
3 Figures cover first 9 months, 1929.
4 Total tons trammed to mill.
5 10,718 tons mined and coarse crushed per day; 5,078 tons hand-sorted out as waste; 5,640 tons fine milled.
6 Based on tonnage trammed.
7 Based on assumption that mill ran 365 days. Only half the fine milling and cyaniding capacity was used.
8 Maintenance and repairs.
9 Fine milling only, $0.7510; crushing and tramming, $0.1764 per ton; total, $0.9274.
10 Based on assumption that mill ran 365 days.
11 Based on 365 working days. Daily tonnage has been 1,270 tons since May 1, 1931.
12 16,847 tons treated; 16,834 tons crushed.

TABLE 14.—*Earlier gold-milling costs in the United States and Canada*[1]

Mill	Year	Cost, per ton milled				Remarks
		Labor	Supplies	Power	Total	
Goldfield Consolidated	1913	$0.457	$1.105	$0.414	$1.976	Stamp amalgamation, concentration, concentrates smelted, sand and slime tails cyanided separately.
Alaska-Treadwell	1912				.228	200 stamps, concentration.
Do	1912				.186	300 stamps, concentration.
Do	1912				[2] 4.568	Concentrates cyanided.
Hollinger	1913				1.391	Concentration and cyanidation.
Do	1924	.3901	$0.5535		.9436	All slime cyanidation.
Do	1925	.3507	.5536		.9043	All slime cyanidation.
Teck-Hughes	1924	.60	.93	.42	[3] 2.36	Do.
McIntyre	1924				1.015	Do.
Homestake	1925				.22	Amalgamation, sand and slime cyanidation.
United Eastern	([4])				1.994	All slime cyanidation.
North Star	1915				[5] 1.14	Amalgamation, concentration, and slime cyanidation.
Melones	1915(?)				.50	Amalgamation, concentration, and cyanidation.
Alaska Gastineau	1917				.24	Gravity concentration.
Do	1920				.28	Gravity concentration.
Le Roi No. 2	1921				1.405	Tabling and differential flotation.

[1] Data for first 5 mills taken from Peele, Robert, Mining Engineers' Handbook: 2d ed., John Wiley & Sons, 1927, pp. 1484 to 1494, and 1955; data for last 6 mills taken from Taggart, Arthur, Handbook of Ore Dressing: John Wiley & Sons, 1927, pp. 124, 125, 130, 867, 962, 963.

[2] Cost per ton of concentrates cyanided.

[3] Total includes $0.41 for maintenance.

[4] Average 1917–1924.

[5] Includes tailings loss of $0.25 to $0.35 per ton.

Part 4.—COSTS OF PRODUCING GOLD

Costs of mine development and stoping, total underground operating, and milling per ton of ore have been presented and briefly discussed in the foregoing sections of this paper. It would have been of interest and value if costs could have been given covering exploration, development, and equipment in the early nonproductive stages of gold-mining enterprises. Such figures were not available, however. The Bureau of Mines is about to undertake a study of the cost of developing and equipping small and medium-size mines both in the precious-metal and base-metal fields, and it is hoped that at a later date considerable data on this subject can be made available.

This section will present current production costs per ounce of gold at a number of gold mines in the United States and Canada.

LACK OF COMPLETE DATA

The current cost figures do not, however, give a complete picture of the costs over the entire life of the various properties. In some instances the purchase price of the properties and costs for development, plant, and equipment may have been wiped out in earlier years, so that these items are not reflected in current cost data. In other instances the charges for depletion and depreciation may only approximately represent an average of these items over the life of the properties.

Costs of early prospecting, exploration, and development are not included in the current cost figures, and they do not reflect losses which may have been incurred by previous unsuccessful companies or even by the present companies during unprofitable periods of operation which may have occurred.

If all these items were added to the current costs they would doubtless add measurably to the total costs of gold production, as indicated by current figures.

The amounts which have been expended in the unsuccessful search for, and exploitation of, gold deposits and which can obviously never be known are doubtless enormous. Disregarding losses through fraudulent promotions these sums, if added to the total costs incurred in gold production by successful companies, would, it is safe to say, give a total cost of gold produced in the United States and Canada much greater than its market value. In this respect, however, gold mining is comparable to the mining of other ores and is not radically different from other commercial ventures in most of which the reward goes to comparatively few successful enterprises, a vastly greater number being unsuccessful.

Gold mining is therefore a legitimate business and, as the affairs of the world are now organized, a very necessary one. It is admittedly highly speculative in nature, its appeal lying in the possibilities for exceptional returns if successful. The most successful ventures have usually been based upon a combination of wise management, technical operating skill, and an element of chance. With all due respect to the faith, courage, and persistence of the prospectors and of the capitalists who have ventured their money, and to the part played by managerial and technical skill in the successful exploitation of gold deposits, good luck has been an important factor in many successful enterprises.

As stated above, although data on current costs of gold production at operating mines do not give a true picture of the total cost of producing gold, such figures are considered to be of sufficient value to be given a place in this paper.

FACTORS AFFECTING COSTS

It has been pointed out in previous sections that the characteristics of the ore deposits; grade of ore; size, occurrence, distribution, and mineral association of the gold particles; and other natural features affect the methods of mining and milling and the production costs. Also it has been noted that the management policy and the efficiency with which methods are employed are reflected in the costs per ton of ore.

The same factors obviously affect the costs per ounce of gold, the cost considerations on this basis being if anything yet more complicated. Since the income of a gold mine is based upon the value of fine gold produced and not upon the ton of ore mined and milled the cost per unit of the metal produced will determine the profit earned.

As these factors have already been considered, further discussion here would be merely a repetition. The available cost figures are therefore given in tabulated form, with only brief comments thereon. The lowest-cost producers based on these figures are in general those working the higher-grade ores. The extent to which the lower costs may be approached by the high-cost mines is limited by natural conditions beyond the control of the operator, and variations within these limits will depend upon management and operating skill.

Due to these factors and, to a smaller extent, to differences in cost-accounting methods comparison of the costs at the different mines would obviously be misleading. The data do, however, show an interesting range of costs. Costs at properties operating at a loss or showing no profit have been deliberately omitted. Many other operating companies do not publish their reports or if they do so the data are very incomplete. Costs of producing gold from placer operations are not available at present, but it is hoped that some such costs can be given later in a contemplated bulletin devoted exclusively to placer mining.

COST STATISTICS OF EARLIER YEARS

Before current costs of gold production are presented cost statistics of earlier years may be of interest. The figures in Tables 15 and 16 were compiled by Charles Janin and appeared in an earlier publication.[1] The operating costs in these tables do not include cap-

[1] Report of a Joint Committee Appointed from the Bureau of Mines and the U. S. Geological Survey by the Secretary of the Interior to Study the Gold Situation: Bull. 144, Bureau of Mines, 1918, 84 pp.

ital expenditure, interest on investments, depreciation of plant, depletion of ore reserves, etc. The costs per ounce of gold have been added to Janin's figures by multiplying costs per dollar recovered by 20.67.

CURRENT COSTS

The figures in Table 17 have been taken principally from published annual reports of the companies. In some instances they have been segregated in a different manner from those in the original reports that they may be on a more comparable basis. The operating costs as given conform more closely than do the overhead and depreciation charges. In the table the Wright-Hargreaves costs allow a credit of $0.975 per ton for ore developed in excess of that milled, while in the costs for the other mines no credit is given for increased reserves. The footnotes in the table indicate other details of the cost assemblies.

TABLE 15.—*Gold-mining statistics for 1917*

[Compiled by Charles Janin from questionnaires sent to largest producers]

State	Number of men [1]	Number of tons treated	Total recovery	Tons per man per year [1]	Recovery per ton	Operating cost per ton	Cost per dollar recovered, cents [2]	Cost per ounce of gold recovered [2]
Quartz mines:								
Oregon	80	31,893	$311,410	400	$9.76	$6.77	69.3	$14.32
California	2,141	939,131	6,140,530	438	6.54	4.26	65.2	13.48
Alaska	1,600	3,153,364	3,724,191	1,970	1.18	1.07	90.7	18.75
Colorado	1,760	1,152,714	8,037,218	654	6.97	4.71	65.7	13.95
South Dakota	2,176	1,677,623	6,609,442	720	3.94	2.99	75.9	15.69
Nevada [3]	1,300	866,983	8,498,658	666	9.80	6.88	70.1	14.49
Arizona	350	166,204	2,429,872	474	1.46	7.52	51.5	10.64
Montana	200	79,647	939,866	398	9.29	6.19	66.7	13.75
Total	9,607	8,067,559	36,691,187	840	4.54	3.19	70.4	14.57

State	Number of men	Cubic yards treated	Total recovery	Cubic yards per man per year	Recovery, cents per cubic yard	Operating costs, cents per cubic yard	Cost per dollar recovered, cents [2]	Cost per ounce of gold recovered [2]
Dredging:								
United States	887	65,654,228	7,645,133	74,000	11.4	5.2	45.2	$9.34
Including Alaska	1,024	66,789,343	8,759,917	65,214	13.2	6.3	48.2	9.96

[1] Approximated at some mines.
[2] Exclusive of amortization of interest on capital.
[3] Includes silver to the value of $5,003,824.

TABLE 16.—*Comparative statement showing details of operations at most important gold mines in the United States, grouped by States; dredging operations combined in one group*

[Compiled by Charles Janin]

1913

State	Tons treated	Total recovery	Total operating cost	Recovery per ton	Cost per ton	Cost per dollar recovered	Cost per ounce of gold recovered
California	697, 606	$4, 464, 755	$2, 430, 748	$6. 31	$3. 48	$0. 54	$11. 16
Alaska	1, 361, 604	3, 598, 334	1, 767, 295	2. 34	1. 30	. 49	10. 13
Colorado	741, 067	6, 544, 823	4, 207, 105	8. 83	5. 68	. 64	13. 23
South Dakota	1, 540, 961	6, 184, 421	4, 576, 654	4. 01	2. 97	. 74	15. 30
Nevada [1]	797, 228	11, 954, 534	5, 597, 082	15. 00	7. 02	. 47	9. 71
Arizona	151, 891	1, 825, 234	1, 172, 509	12. 00	7. 72	. 64	13. 23
Montana	44, 726	414, 594	247, 335	9. 27	5. 53	. 60	12. 40
Total	5, 335, 083	34, 986, 695	19, 998, 728	6. 56	3. 75	. 57	11. 78
Total dredging [2]	51, 405, 333	5, 518, 942	2, 683, 084	[3] 10. 6	[3] 5. 2	. 49	10. 13

1918—FIRST SIX MONTHS [4]

State	Tons treated	Total recovery	Total operating cost	Recovery per ton	Cost per ton	Cost per dollar recovered	Cost per ounce of gold recovered
Oregon	14, 785	$145, 206	$131, 358	$9. 82	$8. 88	$0. 90	$18. 60
California	328, 587	3, 347, 126	2, 144, 000	10. 20	6. 52	. 64	13. 23
Alaska	1, 304, 000	1, 700, 000	1, 600, 000	1. 30	1. 22	. 94	19. 43
Colorado	616, 604	3, 367, 633	2, 449, 821	5. 46	3. 97	. 72	14. 88
South Dakota	827, 104	3, 300, 000	2, 572, 672	3. 99	3. 11	. 78	16. 12
Nevada	184, 594	2, 135, 386	1, 748, 570	11. 60	9. 47	. 82	16. 95
Arizona	87, 130	1, 409, 147	762, 269	16. 20	8. 74	. 54	11. 16
Montana	36, 277	392, 574	256, 109	10. 80	7. 06	. 65	13. 44
Total	3, 399, 081	15, 797, 072	11, 664, 799	4. 65	3. 43	. 70	14. 47
Total dredging [2]	29, 074, 604	3, 140, 733	1, 733, 542	[3] 11. 1	[3] 5. 9	. 55	11. 37

[1] Recovery given from Nevada mines includes silver value as follows: 1913, $4,828,824; 1918, $1,235,000, estimated.
[2] Cubic yards.
[3] Cents per cubic yard.
[4] Figures for 1918 not given for some mines reporting for previous years.

TABLE 17.—*Recent costs of producing gold at lode mines in the United States and Canada*

Mine and year	Average recovery per ton milled	Cost per ton of ore milled						Cost per ounce of gold recovered	
		Mining and development	Milling	Total direct operating	Overhead, marketing, taxes, etc.	Depreciation	Total	Direct operating	Total
Alaska-Juneau; 1930	$0.8602	$0.2869	$0.2271	$0.5140	[1] $0.0956	$0.0531	$0.6627	$12.35	[2] $14.84
Homestake:									
1929 [3]	4.5328	1.7839	.5030	2.2869	.6489	.5494	[4] 3.4852	10.43	[4] 15.90
1930 [3]	6.1755	2.3257	.6732	2.9989	[5] .7707	.7363	[4] 4.5059	10.03	[4] 15.08
Tom Reed:									
Apr. 1, 1929, to Mar. 31, 1930	([6])							9.24	14.26
First 6 months, 1931								8.96	10.00
Cresson Consolidated Gold Mining Co.; year ended Aug. 31, 1929	7.64	3.48	[7] 3.42	6.90	.32	.01	7.23	18.67	19.56
Lake Shore:									
Year ended June 30, 1929 [8]	15.00	4.920	1.220	6.140	[9] .200	.81	7.15	8.46	9.85
Year ended June 30, 1931 [8]	13.10	3.851	.990	4.841	[9] .969	.87	6.68	7.64	10.54
Hollinger:									
1929	6.56	2.86	.65	3.51	.66	.04	4.21	11.06	13.26
1930	6.31	2.98	.64	3.62	.40		4.02	11.86	13.17
Teck-Hughes; year ended Aug. 31, 1931	15.08	3.60	1.14	4.74	.60	.72	6.06	6.49	8.31
McIntyre; year ended Mar. 31, 1931	8.30	[10] 3.215	[11] .93	4.14	[12] .70	.64	5.48	10.25	13.71
Wright-Hargreaves; 1930	11.03	[13] 4.02	1.10	5.12	.61	.23	[13] 5.96	[14] 9.59	[15] 11.17
Vipond; 1930	7.00			4.26	.54		4.80	12.58	14.17
Dome; 1928	7.47	2.39	[16] 1.01	3.40	.62	.81	4.83	9.41	13.37
Premier, British Columbia: 1929	[17] 7.64			3.21	.52	1.31	5.03	[18] 8.65	[18] 13.59
United Eastern; Jan., 1917, to May, 1925 [19]	19.20	4.40	2.01	6.41	1.85		8.26	6.90	8.89

[1] After deducting $0.0306 expense on outside prospecting.
[2] After deducting $1.0795 credit per ounce of gold for lead and silver value.
[3] 1929 tonnage, 1,437,935; 1930 tonnage, 1,364,456.
[4] Depreciation estimated. Annual reports combine depreciation and depletion.
[5] Includes charge of $0.1567 per ton cost of Ellison shaft fire. Miscellaneous credits and balances of $0.0802 per ton deducted.
[6] Approximately 1 ounce gold per ton.
[7] Cost of freight and treatment charges on ore at customs mill.
[8] 1929 tonnage, 364,015; 1931 tonnage, 698,624.
[9] No allowance for taxes in 1929 report; $0.649 allowed for taxes in 1931 report.
[10] Includes exploration $0.108 and examination of prospects $0.024.
[11] Includes coarse crushing and conveying $0.176 per ton.
[12] Includes bullion marketing and mine office $0.1368; administration and general, $0.1718; general insurance, $0.0317; and maintenance and heating of buildings $0.081.
[13] After crediting $0.975 for excess tons developed over tons milled.
[14] If excess development credit is not given the figure becomes $11.42.
[15] If excess development credit is not given, the figure becomes $13.00.
[16] Includes crushing and conveying $0.144.
[17] Gold only, silver average 9.1 ounces per ton.
[18] No credit for silver.
[19] Figures are for entire life of operation exclusive of depletion, depreciation of plant, prepaid development, Federal income taxes, and litigation expense.

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