THE UTAH COPPER ENTERPRISE

DANIEL C. JACKLING

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In this issue we publish the last of a series of articles on an enterprise that may fairly be regarded as the greatest copper mine in the world. It is a veritable mountain of copper ore and is yielding at the rate of 17,000,000 pounds of metal monthly. To those whose experience has been mostly in the winning of the precious metals, the sight of such a vast reserve of ore is astonishing. Usually the more difficult part of mining is to find the ore; it is to the uncertain distribution of gold and silver in veins and lodes that the speculative character of precious-metal mining is due. The search for ore overshadows the question of exploiting it cheaply. On the other hand, a mining engineer looking at the Utah Copper mountain realizes that the anxiety as to quantity is removed, all that remains is the application of skill in breaking, moving, and beneficiating the ore so cheaply as to ensure a profit large enough not only to amortize the capital required to establish systematic industry but to yield a profit commensurate with the financial risk involved in such an undertaking. The prospector stands aside; the engineer steps forward. Then is realized the definition of engineering as used in the charter of the Institution of Civil Engineers: "The art of directing the great sources of power in Nature for the use and convenience of man." The force of gravity, the energy of falling water, the heat latent in coal, and the expansion of steam are harnessed, like a quadriga, by him, in order that he may quickly and easily, and therefore inexpensively, break the metal-bearing rock, transport it, crush it, and subject it to a number of complicated processes so that at the end of a series of operations he may have marketable ingots of pure metal. Such are the operations that we have passed in review. The cost has ranged from 8.85 cents per pound in 1908 to 6.95 cents in 1916, rising, in consequence of war conditions, to 10.995 cents in 1917 and to over 17 cents per pound during the latter part of 1918. The profit has not paralleled the cost because the price of copper, ranging between 12 and 32 cents per pound during ten years, has been a decisive factor, compensating for the higher cost of war-time. The profit in 1914 was only 5.22 cents, as against a maximum of 19.22 cents per pound in 1916. In 1918 the profit will probably be between 8 and 9 cents per pound and the output about 203,000,000 pounds from about 12,600,000 tons of ore. The splendid earning capacity of the property has been obtained at the expense of metallurgical thoroughness. The prime purpose of mining is to make money, therefore the most efficient metallurgical practice is that which yields the largest dividends, making due allowance for the earning power of money and the prospective life of the mine. In Europe mines may be run with an eye to the interest of posterity; in this country our perspective is shorter and more practical. Shareholders prefer to re-invest their own winnings rather than postpone them for the enjoyment of their children or grandchildren. Hence methods that to a philosopher seem wasteful. The accumulation of residual tailing may become a source of revenue to others in days to come, the discard at the mine may prove attractive to the more skilful operator of a later period, but the salient fact is that the ore deposit is now being exploited in a manner likely to give the largest revenue to the present proprietary. That suffices, and more than suffices. To the daring and initiative that conceived the various stages of development and equipment the author of the articles has paid just tribute. The Utah Copper is an epitome of Western energy, resourcefulness, and persistence. To the end of the current year the mine will have produced 80,000,000 tons of ore averaging 1.4% copper, enabling the company to pay $92,015,782 in dividends and to accumulate nearly $50,000,000 in working capital, thus showing a total profit of $142,000,000 in eleven years. The figures are eloquent; they speak for the skill in controlling men and things that has been applied by Messrs. D. C. Jackling, R. C. Gemmell, George O. Bradley, Frank G. Janney, and the staff of young men whom they directed to this great accomplishment.

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THE HISTORY

THE GREAT SALT LAKE was discovered in 1825 by James Bridger, a fur-hunter, while seeking the source of the Bear river. He tasted the brackish water of the inland sea and wondered if it were an arm of the Pacific ocean. The lake was first seen by the Mormons on July 24, 1847. A band of them, numbering 145 and led by Brigham Young, had migrated from Missouri in search of a new home. It is recorded that when Brigham Young stood on the summit of the pass over the Wasatch range and saw the valley of the Jordan outspread before him, he exclaimed: "It is enough; this is the right place." He had seen it in a vision, he claimed, and foresaw the future glory of the new Zion that was to be planted in that happy valley. The proper name of the Mormons is the Church of Jesus Christ of Latter-day Saints, a name adopted by them in 1834 while they were established in Missouri. This sect was founded by Joseph Smith, Jr., at Manchester, New York, in 1830, in consequence of a vision in which the Book of Mormon, in golden tablets, was revealed to him by the angel Maroni, the son of Mormon, both of these, it is alleged, having survived a fierce war waged between groups of Hebrew settlers that came to America from Jerusalem viâ Chile. At one time the word Mormon was supposed to represent the English transliteration of the Greek word *mormoo,* but Joseph Smith denied this derivation, and said that it came from the Egyptian *mon,* meaning good, and the English word 'more', abbreviated to 'mor', so that it meant literally 'more good'! By the end of 1848 the Salt Lake settlement had a population of 5000.

The Mormons thought to escape the jurisdiction of the United States government, but they were thwarted by the treaty of Guadalupe Hidalgo, signed on February 2, 1848, at the close of the Mexican war. By this treaty Mexico ceded an immense stretch of Western territory to the United States, including the region over which Brigham Young and his followers had but lately established control. Early in 1849 the Mormon community was organized as the State of Deseret, with Young as Governor. According to the Book of Mormon the word 'Deseret' means "land of the working bee"—a good symbol for this industrious people and one that is perpetuated by the beehive that now appears on the seal of the State of Utah. Deseret comprised not only the Utah of today but also Arizona, Nevada, and parts of New Mexico, Colorado, Wyoming, and California. Brigham Young claimed a sway imperial in its perspective. In 1850 Deseret was admitted into the United States as the Territory of Utah; it had shrunk in area, but still included portions of New Mexico, Colorado, and Wyoming. On January 4, 1896, the present State of Utah was organized and admitted into the Union. The word Utah is a corruption of Ute, a Shoshone Indian tribe. I shall end these references to the early days of the City of the Saints by quoting a description of the Salt Lake valley as it appeared to the famous traveler Sir Richard Burton on August 25, 1860, when he stood in Emigrant Gap, or 'Emigration Kanyon,' as he called it. Burton wrote in his diary[1] thus:

"The sun, whose slanting rays shone full in our eyes, was setting in a flood of heavenly light behind the bold jagged outline of Antelope Island, which, though distant twenty miles to the north-west, hardly appeared to be ten. At its feet, and then bounding the far horizon, lay, like a band of burnished silver, the Great Salt Lake, that still innocent Dead Sea. South-westward also, and equally deceptive as regards distance, rose the boundary of the valley plain, the Oquirrh Range, sharply silhou-

[1]'The City of the Saints, and across the Rocky Mountains to California.' By Richard F. Burton; 1862.

etted by a sweep of sunshine over its summits, against the depths of an evening sky, in that direction so pure, so clear, that vision, one might fancy, could penetrate behind the curtain into regions beyond the confines of man's ken. In the brilliant reflected light, which softened off into a glow of delicate pink, we could distinguish the lines of Brigham's, Coon's, and other kanyons, which water has traced through the wooded flanks of the Oquirrh down to the shadows already purpling the misty benches at their base.'' Since then many shadows have crossed the range, but the sunsets retain the same qualities of beauty, and the clear ether still provokes the imagination of man. Sir Richard spells 'canyon' in his own way and undoubtedly was misled by false analogy to speak of ''Brigham's'', instead of Bingham, canyon. He was exhilarated by the fresh mountain air, in common with every traveler, so that it seemed to him ''Switzerland and Italy lay side by side.'' To him the Salt Lake valley, ''this lovely panorama of green, and azure, and gold—this land, fresh as it were, from the hands of God,'' was worthy to be the Zion of the Latter-day Saints and even of later sinners.

The discovery and development of mineral deposits in the Bingham district, like that of the territory of Utah as a whole, was retarded by the opposition of the Mormons, who intended to make agriculture the basis of their development and therefore discouraged the search for minerals. ''This opposition'', as S. F. Emmons has said, ''and the natural obstacles in the way of cheap mining, or of an economic reduction of the generally rather refractory ores, acted as an effectual bar to the development or even the discovery of the mineral resources of the Territory in its early days.''[2] The transcontinental migration that followed the discovery of gold in California did not lead to any important find of ore in this region, because the leaders of the Mormon church were successful in preventing it, ''fearing that the excitement and unsettling influence of mining would turn away their people from the more monotonous and peaceful occupations of agriculture and thereby interfere with their great work of reclaiming the desert, and fearing, also, that the restless and sometimes rather lawless class of people who are attracted by mining excitement might prove a disturbing element in the population and tend to subvert their almost autocratic authority.''[3] Orson F. Whitney, a Mormon bishop, in his history of Utah, exclaims: ''Who wished to see Deseret, peaceful Deseret, the home of a people who had fled for religious freedom and quiet to these mountain solitudes, converted into a rollicking, roaring mining camp? Not the Latter-day Saints!''

The logic of events proved too strong for the Mormons in Utah as it proved too strong also for the Boers in the Transvaal. Indeed, the analogy is not sufficiently appreciated. The Boers left the Cape Colony in order to escape from British control; they believed in the practice of slavery, quoting biblical authority therefor; and they wanted to have ample room for their pastoral industry. So they crossed the Vaal. The Mormons, desiring to escape from the legal control of established American communities, believing in the practice of polygamy, for which likewise they quoted the biblical text, and, hoping to develop a purely agricultural industry in an isolated region, migrated into the western wilderness and settled in the valley of the Jordan on the shores of the Great Salt Lake. Boer and Mormon* alike expected to remain detached from the civilization they had abandoned, each expected to establish his own way of living in the outer wilderness, each was disappointed and thwarted by the oncoming tide of industrial progress. The pioneers of mining, in their eager search for mineral wealth, invaded the Transvaal and Utah, at first peaceably and then aggressively, establishing themselves assertively as their numbers increased, until the long arm of modern progress, which neither Boer nor Mormon could defy, reached forth into the new communities and claimed suzerainty. Slavery was as repugnant to the Briton as polygamy was repugnant to the American. The maintenance of these practices undermined the status of the Boers and Mormons, respectively; it created a prejudice under cover of which the invading prospector was enabled to obtain protection, ending in the submersion of the political islands that the Boer and the Mormon alike had tried to create in the midst of a continent.

So Brigham Young failed to realize his purpose of establishing a separate people, but he succeeded, beyond his dreams, in laying the foundations of a thriving community on the shore of the great lake. Any disparagement of the Mormons on account of their opposition to mining or on the score of their polygamous practices must be coupled with a hearty admiration for their energy as pioneers and with high praise of their thrifty enterprise. To this day they set a good example to the people of the West by their intensive farming, collective loyalty, and intelligent co-operation.

The existence of silver ore near the Great Salt Lake became known in 1857, but Mormon influence prevented the development of mines. In 1862, during the Civil War, the Third California Infantry happened to become stationed at Fort Douglas, overlooking Salt Lake City. Many of the soldiers in this volunteer regiment had seen something of gold mining in California; their commander was Colonel Patrick E. Connor, who, it is said, looked to immigration to settle the Mormon question, then becoming acute. He encouraged prospecting and granted furlough to his men so that they had frequent opportunity for exploring the neighboring mountains, when not required to prevent depredations by the Indians—Piutes and Goshutes—and to keep an eye on the Mormons, who

[2]'Economic Geology of the Mercur Mining District, Utah.' U. S. G. S., 1895.

[3]Ibid. Op. cit.

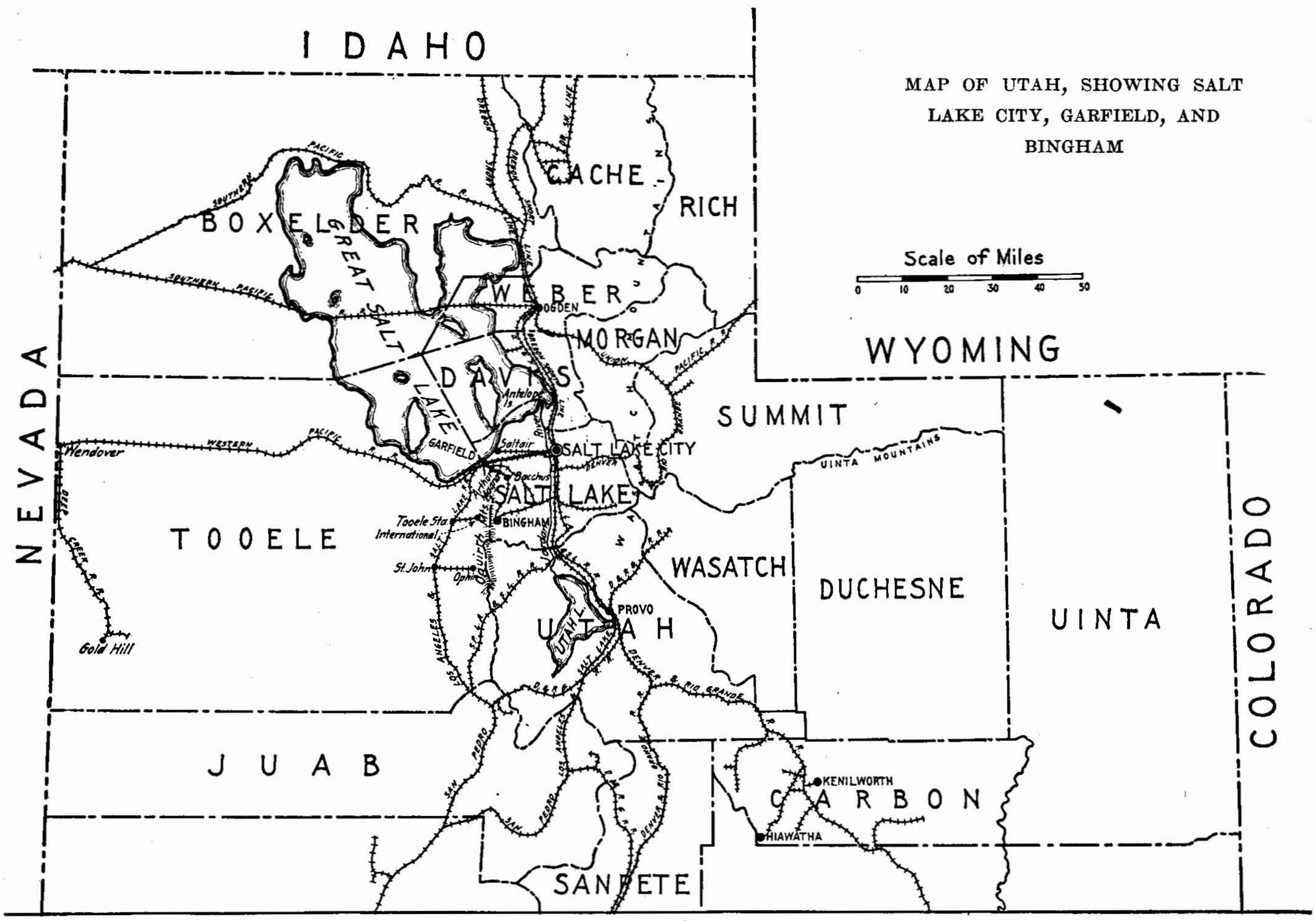
*It is worthy of note that the Mormon community included a large proportion of British.

were unfriendly. Bancroft says:[4] "The first systematic efforts at prospecting, made by permission of General Connor,[5] when in command at Fort Douglas, were ridiculed in the Tabernacle, and later, when mining projects were brought forward by Gentiles, they were steadily discountenanced." Whitney says[6] of Connor: "His object was to reconstruct Utah, to put the Mormons under and render the Gentiles paramount. To effect that object he strained every effort of his energetic soul; hesitating not to grossly exaggerate, not only the growth of the infant industry of mining, making it appear a very giant at its birth, when everyone knows that for years it was a mere babe in arms, never attaining to any proportions until after the advent of the railway, but also the general condition of affairs at Salt Lake City and throughout the Territory."

At that time the Bingham district was known to the Mormons for its fine timber, red pine three feet in diameter being common. Early in the autumn of 1863 an apostate Mormon, George B. Ogilvie, found fragments of lead ore in the canyon and took them to Fort Douglas, where they were exhibited to Colonel Connor. On September 17, 1863, the discovery was located as the 'Jordan Lode', which was the first mining location made in Utah. In the following December a mining district, the first in the Territory, was organized under the name of West Mountain, which is the English for the Indian word Oquirrh. It included the whole of the Oquirrh range.

Bancroft describes the discovery as follows: "In 1863 Captain A. Heintz and a party from Camp Douglas discovered argentiferous galena and copper in Bingham canyon, on the east slope of the Oquirrh range, near the Jordan, and about thirty miles south of Salt Lake City. A mine was located in September of that year by a man named Ogilvie, and in December following, a mining district was established, named the West Mountain, and including the portion of the range between Black Rock, at the southern end of Great Salt Lake, and the fortieth parallel. In 1871 this district contained 35 mines."

Whitney's description may be compared with the foregoing: "A party of soldiers from Camp Douglas were guarding some horses belonging to the garrison which [the horses, not the garrison—Editor] had been sent to graze in Bingham canyon. They were joined one day by General Connor and a picnic party of officers and their wives from Camp [Douglas], and one of the ladies, while



MAP OF UTAH, SHOWING SALT LAKE CITY, GARFIELD, AND BINGHAM

[4]'History of Utah', by H. H. Bancroft. Page 741.

[5]Bancroft anticipates Connor's promotion; he was appointed major-general of the Utah militia in 1870. He played a prominent and useful part in Utah's early history.

[6]'History of Utah', by Orson F. Whitney; Vol. II, p. 111.

rambling on the mountain-sides, picked up a loose piece of ore. The soldiers at once prospected the vein, discovered it, and sticking a stake in the ground made their location, since which Utah has been known to the world as a rich mining camp. Another account, by the historian E. W. Tullidge, states that a man named Ogilvie, while logging in the canyon, found a piece of ore, which he sent to General Connor, who had it assayed. It was then, according to Mr. Tullidge, that Connor organized his picnic party and proceeding to Bingham Canyon located the mine, which was named the Jordan. Soon afterward Connor wrote some mining laws and held a miners' meeting at Gardner's mill on the Jordan river, where the laws were adopted and Bishop Gardner elected recorder of the West Mountain mining district. Thus was the ball set rolling."

In June 1864 another mining district, the Rush Valley, was formed, covering the western slope of the range and leaving the eastern slope to the West Mountain district, a name that, despite its inappropriateness, has persisted to this day. In 1864 the Jordan Silver Mining Co. was organized under Californian laws, and vigorous prospecting was begun; but these early operations languished, in spite of several handsome outcrops of galena, because of the many obstacles to mining. The lack of railroad transport rendered all supplies excessively expensive; a shovel cost $2.50, a keg of powder, $100. Moreover Mormon influence was effective in discouraging the settlers in the Salt Lake valley from participating in the development of the lead deposits. The discovery of gold, in 1864, by a party of Californians, returning from Montana to spend the winter at Salt Lake, led to the systematic washing of the gravel in Bingham canyon. A nugget weighing a little more than an ounce was found and these alluvial operations altogether are said to have yielded $1,000,000 during the ensuing six or seven years.

The first mining at Bingham was prompted by the silver-bearing lead ore that outcropped on the hillsides, by the gold found in the gravel of the gulch, and also by the gold mined from quartzose lodes penetrating the limestone. In 1882 four stamp-mills had been built to crush this gold-bearing silicious ore. The success of the lead mining was due not only to the silver contents of the ore in such celebrated mines as the Jordan, Galena, and Winnemucca, but also to the deep oxidation of the mineral, yielding easily reducible carbonate ores even at considerable depth, in such mines as the Brooklyn, Lead, and Yosemite.

According to Bancroft, "the first shipment of ore from Utah was a carload of copper ore from Bingham canyon, hauled to Uintah,* on the Union Pacific, and forwarded by Walker Brothers to Baltimore in June 1868." This was before the transcontinental railway was completed by the linking of the Union Pacific and Central Pacific railroads at Ogden on May 10, 1869, and it was two years before Salt Lake City was joined to the main line by the construction of the Utah Central railroad. The mention of copper so early is interesting, for no large shipments were made until December 1896, when 5000 tons of sulphide ore was shipped from the Highland Boy mine. It is worthy of note that the original Highland Boy company was started as a gold-mining venture by Samuel Newhouse, as promoter, and Thomas Weir, as manager. These two had so little idea of developing a copper mine that they built a cyanide mill to extract the gold in the oxidized ore. This mill ran for several months, but it was not a success, largely because the copper, of which they had failed to take note in their experimental work, interfered with the recovery and caused an abnormal consumption of cyanide. After this mill proved a failure and the company's affairs reached a critical stage, Mr. Newhouse, it is said, went to Denver in order to raise money to meet his delinquent payrolls, and while there he received a telegram from Mr. Weir informing him that ore containing 15% copper had been struck in the lower adit. This saved the day. Another story asserts that in the course of exploratory work in the zone of oxidation a winze penetrated sulphide ore. This so alarmed the management that the winze was covered. Later, failing to develop a successful gold mine and the price of copper making it attractive, the winze was reopened and a discovery of copper sulphide was announced.

At first the copper deposits were overlooked. They were of low grade and not as easily smelted as the lead ores. The shipment of copper-sulphide ore from the Highland Boy in 1896 gave the first promise of a new departure and led to the transfer of this mine, with other claims, to the Utah Consolidated Mining Co., which, three years later, completed a smelter of 250 tons capacity. In 1897 the Stewart No. 2 and a number of adjacent claims were acquired by the Boston Consolidated Mining Co. These are mentioned by J. M. Boutwell in Professional Paper No. 38, U. S. Geological Survey, and in that report he has an addendum bringing the record of development down to 1902, but there is no suggestion of the enterprise that was to give world-wide fame to Bingham.

It is probable that the first mining on the present Utah Copper property was done at the time when Colonel Connor's soldiers began to prospect in the hills enclosing Bingham canyon, that is, soon after 1862, for the Soldier Tunnel, by its name, perpetuated that tradition. The oldest claim now included in the property is the Washington, which was located in 1865, whereas the latest claim, the Jubilee Fraction, was located on November 29, 1910, seven years after the Utah Copper Company had been organized.

The story of the Utah Copper enterprise begins with Colonel Enos A. Wall, of Indiana. As he himself acknowledges smilingly, his military title is one that he owes to his friends. His parents were North Carolinian;

*Given as 'Utah' in the report by Boutwell.

he started his mining career in Colorado in 1860 and went from there to Montana in 1863, varying the search for gold with general business as a freighter and trader in the material and supplies exchanged between that Territory and Utah, to which he came in 1868, remaining for 14 years. Then for five years he was chief stockholder and superintendent of the Wood River Gold & Silver Mining Co., an important enterprise at Bullion, Idaho, where he won the regard of his fellow-citizens so as to be elected to the upper house of the Territorial legislature and become President of that body of Solons, as, I believe, a newspaper reporter would call them. In 1885 he returned to Utah, engaging in mining at Mercur and elsewhere.

In July 1887 he went to Bingham, where he noticed a number of prospecting drifts and inclines driven during early days on the small fissures that traverse the quartzite and monzonite on both sides of the canyon. His attention was attracted by a discoloration on the hillside, visible from the road. The stream of water issuing from a spring just above the site of the present 'pit' of the Utah Copper mine had been conducted to a placer near the site of the present railway station. The bare rock on the hillside had become discolored and the gravel in the gulch likewise was stained green by the coppery solution. Upon examination, the ridge of rock proved to be an outcrop of monzonite impregnated with copper sufficiently to assay 3% for a length of 300 ft. An abandoned 'tunnel', 90 ft. long, had been driven into the hill on about the level of the present 'pit'. This tunnel had followed a short-lived fracture that had yielded pieces of ore rich in chalcocite, but the work evidently had proved unprofitable. Entering the tunnel, Wall broke a sample; upon the fresh face of the rock, under the green coloration, he saw that the monzonite was impregnated with black specks of chalcocite and bornite, suggesting a similarity to the ores of Butte, with which he was familiar. He sampled the tunnel, omitting the 20 ft. next the surface, where the copper-bearing rock was oxidized, and obtained an average of 2.4% copper by assay. Numerous tests by panning showed that a concentrate assaying 30 to 40% copper could be produced.

Upon enquiring at the Recorder's office, he ascertained that a large part of the ground adjoining and surrounding this exposure of mineral had been abandoned and therefore was subject to re-location; so he staked two claims, which he named 'Dick Mackintosh' and 'Charles Read', after two of his local friends. This gave him an area of 3000 by 600 ft., except a small fraction subject to conflict at one end. Subsequently he located another adjacent claim, which he named the 'Frank Cushing'. He found other old workings, one of them being a tunnel 250 ft. long on the opposite, or east, side of the gulch. This was nearly on the same level and about 700 ft. northeast of the one first inspected; it followed the so-called Quinn fissure, a gash marked by an irregular enrichment with chalcocite, similar to many other short-lived fractures traversing the monzonite mass. The ore in this fissure assayed 5 to 40% copper, but it was not in quantity sufficient to justify the method of selective mining that the former owner of the claim had attempted to apply. A new tunnel 600 ft. northward and on the same side of the gulch had been driven 200 ft., and was in continuous ore averaging 1.8% copper. These facts indicated an extensive dispersion of the copper. He investigated the titles of the claims adjacent to his own locations, meanwhile keeping his hopes to himself. He even agreed with the road-supervisor that the dumps be used for road-making, being willing to have the value of the ore ignored. The local wits called it 'Wall-rock'.

At that time he did not have enough money to start systematic development, but, from year to year, he did his assessment work. Later he made some money by sundry deals at Ophir and Mercur. For example, he sold the Brickyard mine, at Mercur, and cleaned up $60,000 on that. He developed the Yampa mine, near the Highland Boy, and made such a showing as to be able to sell it to George H. Robinson, for Moore & Schley, at a price of $150,000, as against the cost to himself of $40,000. He was not a promoter, but a miner and a dealer in mines, backing his personal judgment successfully and establishing a good bank-credit in Montana and Utah. He held to his faith in the copper-bearing 'porphyry' at Bingham, and looked forward, some day, to acquiring financial strength sufficient to enable him to provide the plant and equipment necessary to exploit the deposit on the large scale that it demanded. By aid of his winnings, he had been able to perform the assessment work on his own property as it became due; he had bonded other claims, and paid for some of them; so that when the success of the Highland Boy started a copper excitement in 1896 he had acquired ownership of 200 acres and had done $20,000 worth of exploratory work, as represented by about 3250 ft. of tunnels, drifts, and cross-cuts.

Enter Capt. Joseph R. De Lamar. This actor has played a part in mining romance all the way from the Sangre de Cristo mountains to Lake Nipissing. He has been known to give the comic touch to the heavy performance of *la haute juiverie* at New York and to render foolish the antics of their super-expert. The Captain is a remarkable man, and he has had a picturesque career. He was born at Amsterdam 75 years ago. He has been a diver; he has commanded merchant-ships between New York and the Bermudas; thus his captaincy is of the sea. He has bought prospects and developed mines in Georgia, Colorado, Idaho, Nevada, Utah, and California. His name is on the postal map. I am told that he is a hard worker; a man of keen judgment, capable of quick shrewd decisions; possessing the rare faculty of looking ahead. Capt. De Lamar has proved himself alert and enterprising; he is an adventurer, using the term in its old sense of one willing to venture, as the Cornish used to speak of the shareholders in a mining enterprise and as the proprietors of the Hudson's Bay Company are still called officially: "the adventurers trading in the Hudson's Bay territory." Had the Captain lived in the spacious times of Great Elizabeth he would have harried the Spanish main, chased the heavily-laden galleons, and taken tribute of the *conquistadores.* In short, Capt. De Lamar is a notable figure in American mining.

De Lamar had known Colonel Wall at Mercur, for, among other dealings, he had bought from him the Brickyard mine in 1894. At that time the Captain's chief of staff, or manager, was Hartwig A. Cohen, now in San Francisco. In 1895 Cohen came to examine Wall's copper prospects; he took a few samples* and made some hand-tests, by panning, the result being a favorable opinion. De Lamar obtained a six-months option on three-quarters of the property at $375,000. A test was made on 76 tons of ore in a small stamp-mill, called the Markham, in the lower part of the town of Bingham. This ore came from the Mackintosh tunnel, then about 300 ft. long, and from two other similar prospecting drifts. The test yielded a concentrate containing 28 to 33% copper, on an ore assaying 2%, the recovery being 60 to 62%.

No business ensued. De Lamar thought the recovery discouraging and the ore too poor; moreover he was intimidated by the weakness of the copper market, which, near the close of 1895, suffered from disturbed financial conditions, the quotation for electrolytic copper falling to 9¾ cents per pound, as against 12 cents a few months earlier.

Three years later, in 1898, De Lamar, probably encouraged by the rise in copper, which was quoted at 12 to 12½c., in the summer of that year, again asked for an option, in order to make further tests and investigations. He obtained an option on a quarter interest at $50,000 and on a second quarter at $250,000. A preliminary sampling was undertaken by Robert C. Gemmell (Univ. of Michigan, '84), at that time engineer for De Lamar at his Golden Gate mine, at Mercur. Oxidized copper ore was exposed wherever the rock was not hidden by vegetation. On the east side there was more brush than on the west side of the gulch. Some tests were made in a 5-stamp mill, called the Rogers, below the mine, by Daniel C. Jackling (Missouri S. of M. '92), who also was a member of De Lamar's staff, being then metallurgist at the Golden Gate mill. Everybody seemed pleased with the investigation, including De Lamar himself, but that astute financier told Wall that he would like an extension of time in order to do some exploratory work in the mine, and that he was prepared to undertake it if he could acquire a larger interest. Wall replied that he would sell three-quarters of the property for $750,000 cash. That ended the negotiations.

Shortly afterward, at the end of 1898, De Lamar and Cohen had a disagreement, the immediate consequence being that Cohen resigned as manager for De Lamar, his place being given to Victor Clement, on a salary of $36,000 per annum and an eighth interest in anything he found, that is, in any new mining venture introduced by him to the Captain. Clement had returned in 1896 from Johannesburg, where he had been manager of the Simmer & Jack mine, controlled by the Consolidated Gold Fields of South Africa, Ltd. He had met Wall, and a friendly acquaintance had developed. From Gemmell and Jackling he heard about the Wall property; thereupon, early in 1899, he told Wall that he had been going over Cohen's figures and thought he saw the making of a successful venture; he informed Wall that he himself would participate in any business that might result with De Lamar and he could guarantee a square deal. Clement took Gemmell with him to Bingham, and they went over the ground. The result was an offer, by Clement, for De Lamar, to purchase a quarter for $50,000 outright, with a year's option on another quarter at $250,000, and on a third quarter at $1,250,000. This proposal was accepted. It was Clement's intention to prove the property and then sell the third quarter through his financial friends in London, thereby obtaining the money needed to build a mill and a railroad from the mine to the mill. During the year of the option De Lamar had the right to test and explore. Clement put James Mason in charge of the mining work and spent $25,000 in extending the drifts and in driving new cross-cuts, all of which were carefully sampled by Gemmell. The mill-tests were placed in charge of Jackling, who repaired the old Rogers mill, in the gulch just below the Ohio Copper mine and conveniently near the Wall property. The mill was equipped with a 5-stamp battery, two Wilfley tables, and a vanner. Henry M. Crowther acted as superintendent. The mill-tests served to check the sampling of the workings by Gemmell, and to determine the amenability of the ore to ordinary concentration. I am glad to be able to quote from the report made by Jackling at that time.

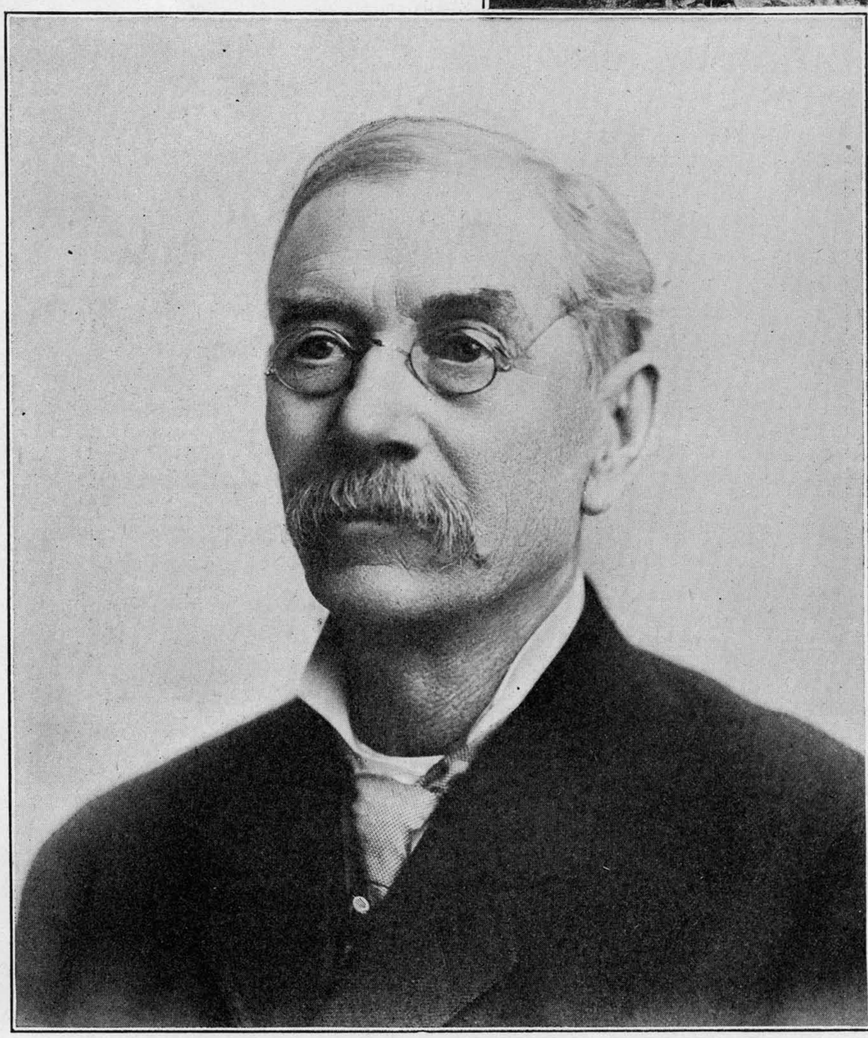
This report, dated September 18, 1899, is addressed to Cohen as manager for De Lamar; it is extremely interesting when read in the light of later developments. The report was the result of collaboration between Messrs. Jackling and Gemmell, the latter writing the portions that bore upon the development of the mine, the probabilities of further discovery, the character of the ore-body, and the average value of the ore as determined by sampling. I take pleasure in reproducing the map of the principal workings, with the results of the sampling as done by Mr. Gemmell at that time. The assay-figures were written on the original map at the points where the samples were taken, but they would have been illegible when the map was reduced to the size of one of our pages, so I give them separately in another column, with the outlines of the various tunnels, rendering identification easy.

*Which, he tells me, he assayed by the volumetric cyanide method in a little house on the dump of the Mackintosh tunnel.

TRAIN-LOAD OF COPPER ORE CROSSING VIADUCT OVER MARKHAM GULCH, AT BINGHAM. THE COPPER MOUNTAIN IS IN THE BACKGROUND.

CAPT. J. R. DE LAMAR

WHO HAD AN OPTION ON THE UTAH COPPER MINE

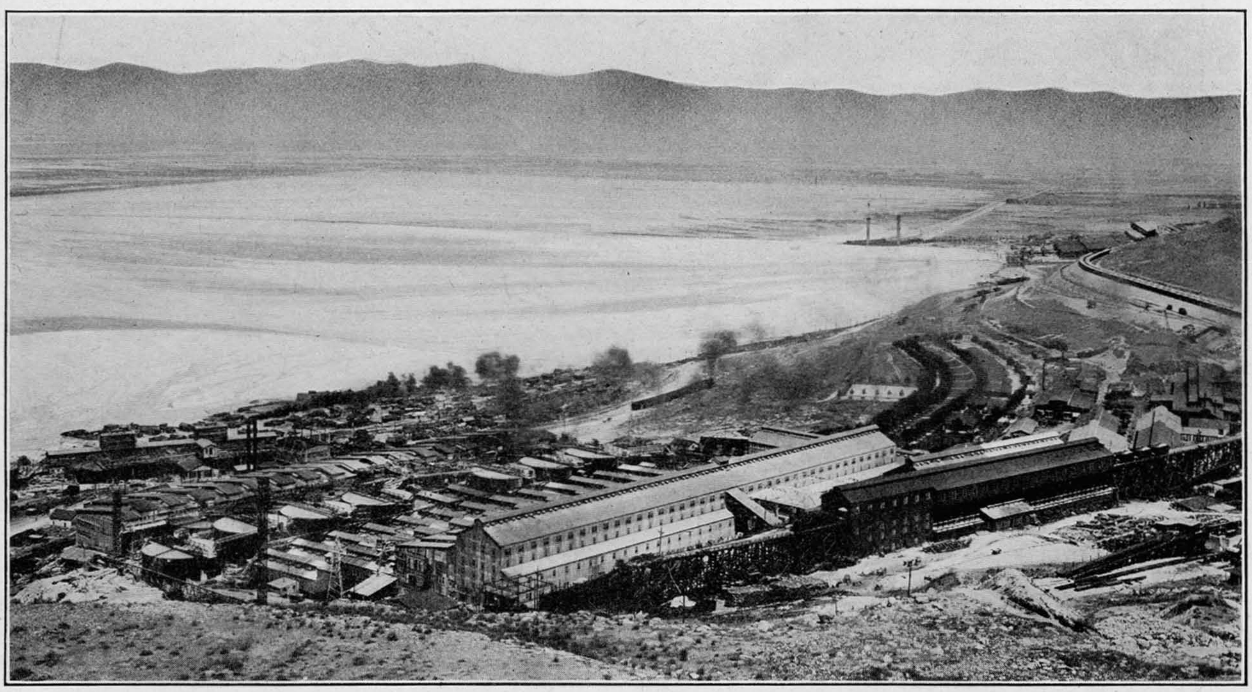


COLONEL ENOS A. WALL

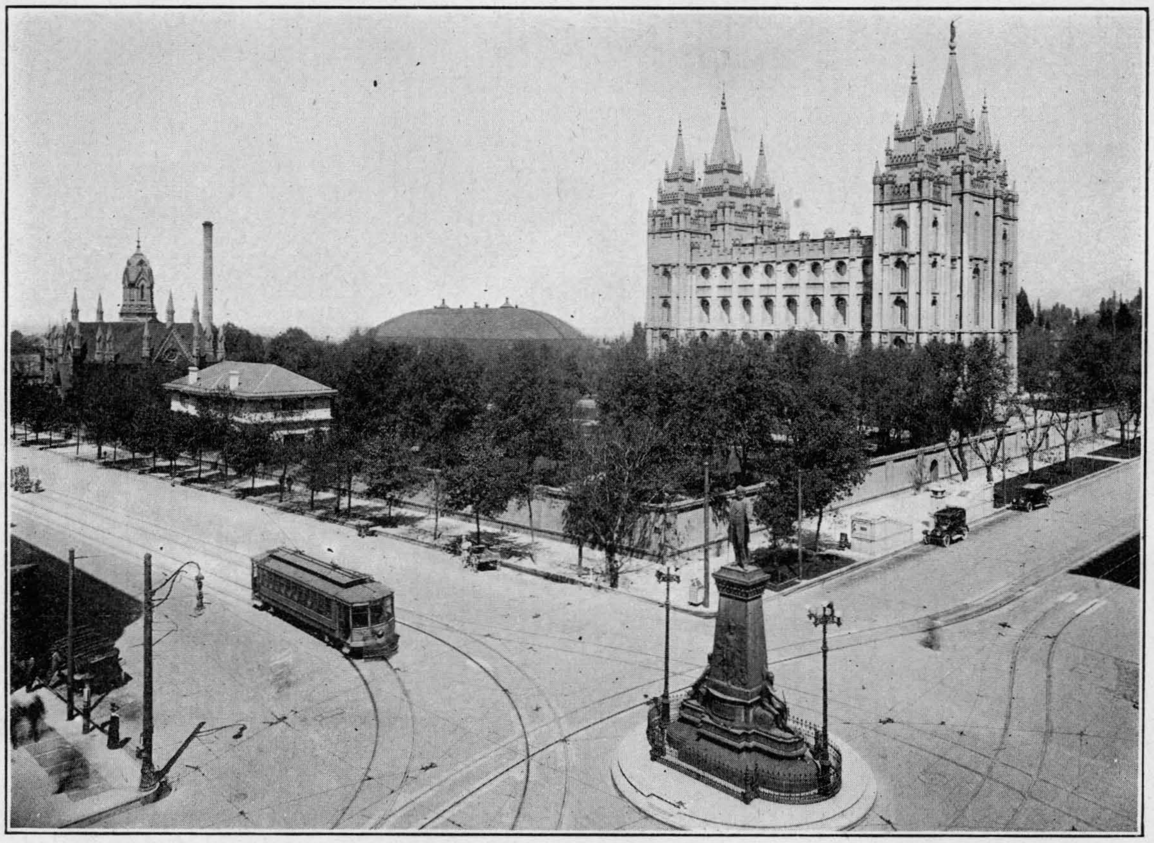
ORIGINAL OWNER AND PROSPECTOR OF THE UTAH COPPER PROPERTY

ROBERT C. GEMMELL

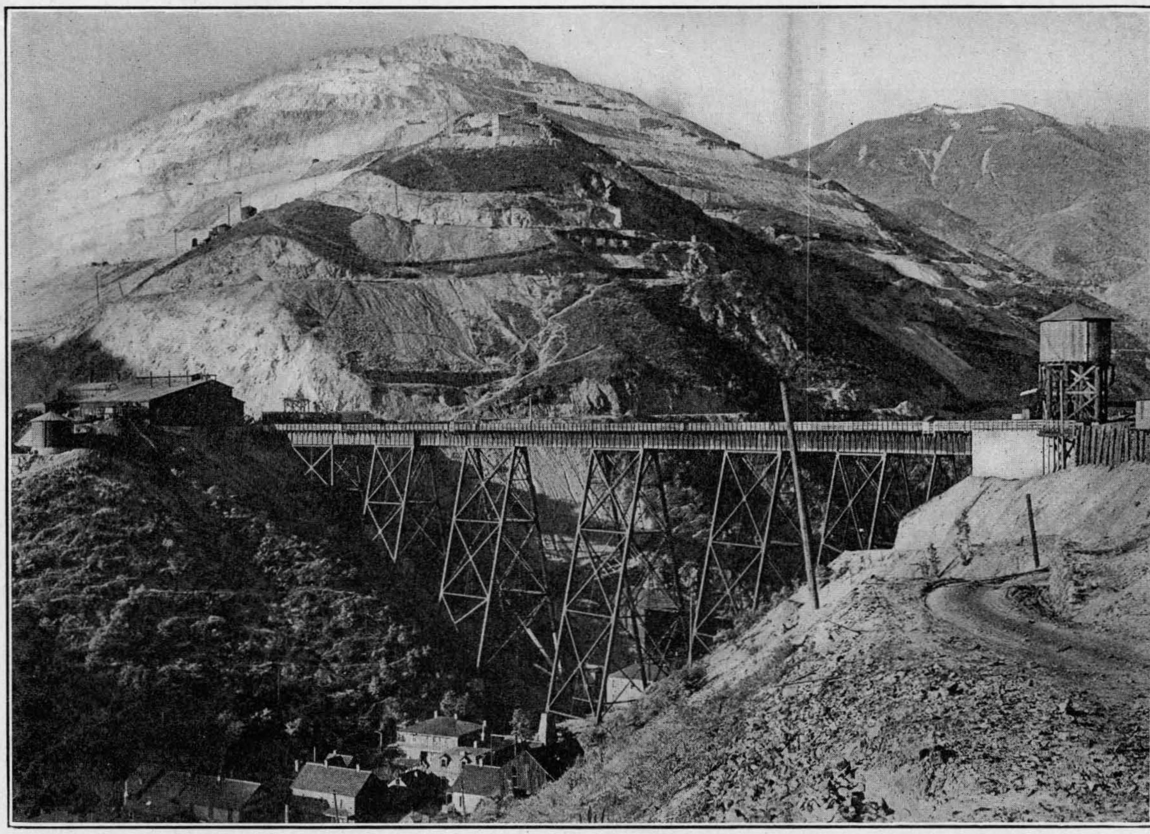
GENERAL MANAGER FOR THE UTAH COPPER COMPANY



THE MILLS OF THE UTAH COPPER CO. THE ARTHUR MILL IN THE FOREGROUND AND THE MAGNA MILL IN THE RIGHT BACKGROUND; TAILING-POND IN CENTRE



THE TABERNACLE AND TEMPLE, SALT LAKE CITY



THE UTAH COPPER MINE, WITH THE CARR FORK VIADUCT IN THE FOREGROUND

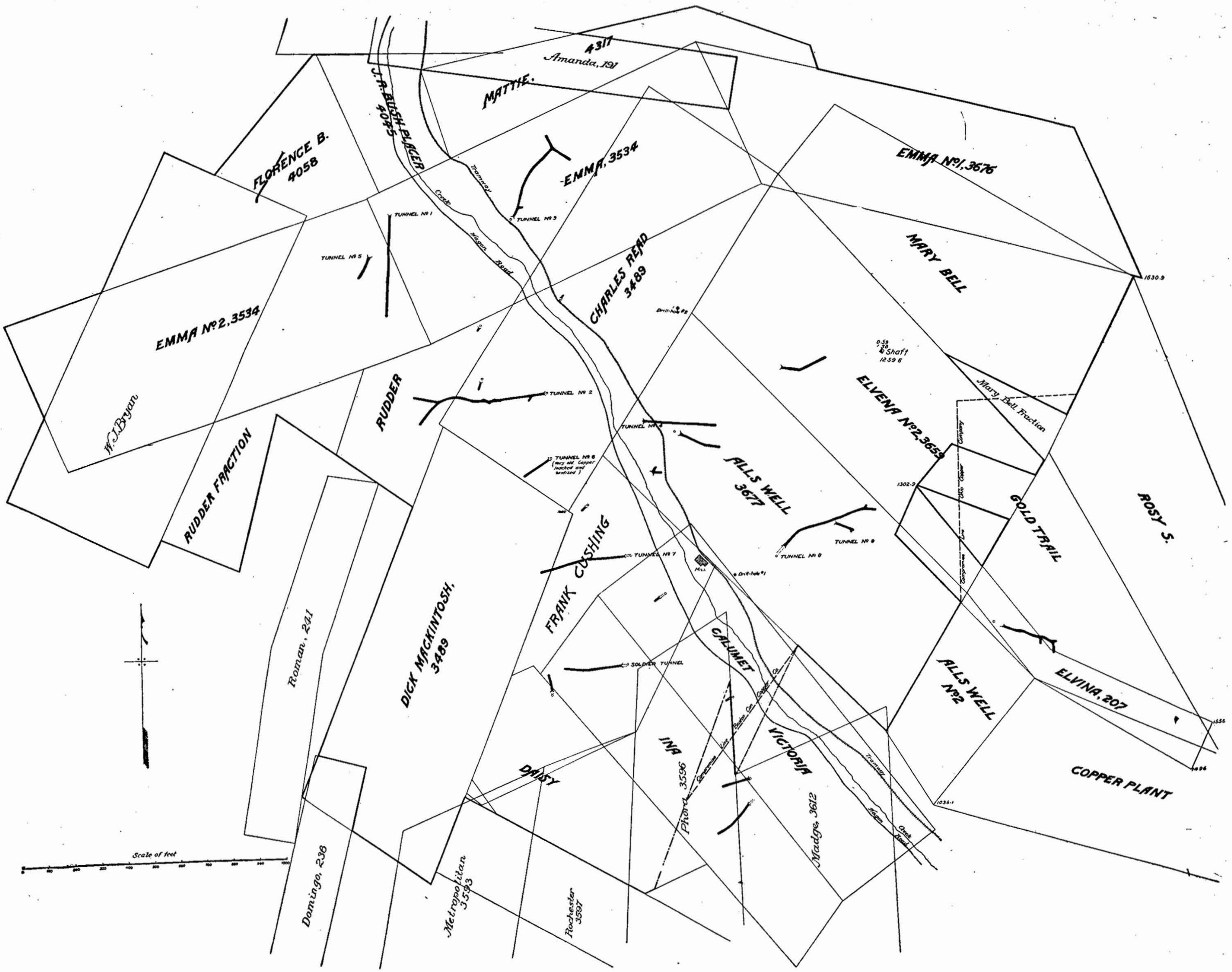
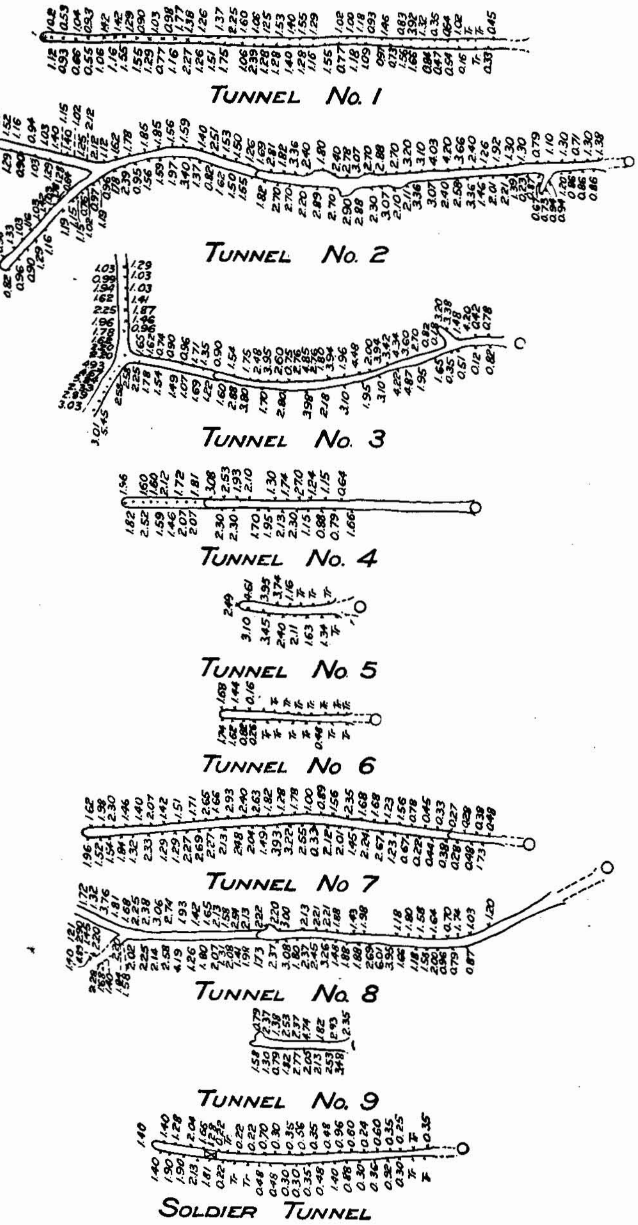


FIG. 1. MAP OF THE WALL PROPERTY, NOW UTAH COPPER MINE, WHEN SAMPLED IN 1899

In his report, Mr. Jackling describes the ore-bearing rock as "a very silicious fine-grained porphyry," which has been so fractured naturally that when blasted the ore breaks into small fragments. Where Bingham canyon cuts this porphyry the width of ore is 3000 ft. "The



ASSAY-PLANS OF THE PRINCIPAL WORKINGS

(See map on preceding page)

copper occurs in the form of grains of very pure chalcopyrite quite uniformly distributed throughout the mass, there being very little iron pyrites, or other base sulphides of any kind associated with it. Near the surface the oxidizing action has been such that the larger percentage of copper has been converted into carbonates, and, to a certain extent, silicates, and in most cases the copper thus oxidized has been, to a very great extent, leached out of the rock, leaving the values practically nothing on the surface, but increasing rapidly with depth, both as to value and percentage of copper present as sulphides." Thus he recognizes the effect of secondary enrichment. He states that only the tunnels numbered 1 to 9 and the Soldier tunnel have been considered in calculating the amount of ore developed, "no attention" being paid "to shallow surface workings or to incline-shafts sunk on small veins." Evidently, for the sake of simplicity, the workings that were used in making these calculations were numbered from 1 to 9, inclusive, and were so shown on the map accompanying the report. In order to correlate this numbering with the workings shown on the original sampling-map, as reproduced herewith (Fig. 1), it is necessary to say that the order of enumeration from 1 to 9 was Tunnel 1, Tunnel 12, Quinn tunnel, Tunnel 2, Tunnel E, Tunnel 6, Tunnel 15, Tunnel C, Tunnel D, and Soldier Tunnel. Samples from raises in these tunnels were used in arriving at an average, but the results from the churn-drill holes were questioned, and therefore were not used. Samples were taken at intervals of 10 ft. on each side, alternating, so that the workings were really sampled every 5 ft. The samples were reduced and quartered in the usual way. Mr. Gemmell tells me that notes were made as to whether the samples represented oxidized or sulphide ore, the assays of the oxidized ore being used in calculating the average value of the 'cap' and the assays of the sulphide samples for calculating ore. The original blue-print, fom which the accompanying map was copied, shows a number of shallow surface workings, such as pits, open-cuts, short drifts, and the like. Generally these surficial diggings had been made at particularly favorable spots, often where the gash-veins of richer oxidized ore had been detected; therefore, the assays of samples from these shallow workings were discarded. Assays of ore from the two tunnels called 9 and 10 on the blue-print were ignored, because these tunnels were not on the Wall property. The total length of workings on the Wall property, including the 2520 ft. done during the terms of De Lamar's option, amounted to 5770 ft. The report states that in estimating the average grade of the concentrating ore 461 samples were used, and as these samples were taken every 5 ft., as explained, they would represent an aggregate length of workings equal to 2305 ft. About 40 ft. of surficial rock (consisting of soil, detritus, and leached porphyry) would have to be stripped and dumped to one side, says Mr. Jackling. He takes 300 ft. as the average length of ore proved by the tunnels, and 200 ft. as the average maximum depth of ore. The length, as exposed in the tunnels on both sides of the canyon, is 2000 ft., "and it is assumed that the ore will be quarried to a plane level with the gulch." Thus he indicates the method of mining he has in view. From the dimensions given, he estimates 167,200,000 cubic feet, which, at 13½ cu. ft. per ton, is equivalent to 12,385,000 tons of ore. This assumes "an equally developed orebody on either side of the canyon." The average assay is 2% copper, as determined from 461 samples broken in the ten principal tunnels, omitting the 40 ft. of stripping. The mine-sampling is confirmed by the mill-tests. In conclusion, Mr.

Depth	Hole No. 1 %	Hole No. 2 %	Depth	No. 1 Hole %	No. 2 Hole %
0 to 5	...	...	225 " 230	2.30	2.16
5 " 10	1.02	...	230 " 235	2.05	2.00
10 " 15	1.01	0.10	235 " 240	1.92	1.86
15 " 20	0.64	0.10	240 " 245	3.46	1.98
20 " 25	0.51	0.15	245 " 250	2.25	1.82
25 " 30	0.54	0.18	250 " 255	1.77	1.54
30 " 35	0.50	0.24	255 " 260	1.89	1.32
35 " 40	0.92	0.28	260 " 265	2.40	1.52
40 " 45	1.21	0.52	265 " 270	1.89	1.34
45 " 50	1.49	0.71	270 " 275	1.13	1.30
50 " 55	1.68	1.30	275 " 280	1.51	...
55 " 60	1.54	1.05	280 " 285	1.13	...
60 " 65	1.50	0.94	285 " 290	1.01	...
65 " 70	1.71	0.54	290 " 295	1.13	...
70 " 75	1.75	0.62	295 " 300	1.01	...
75 " 80	1.58	0.84	300 " 305	1.26	...
80 " 85	1.62	1.42	305 " 310	0.90	...
85 " 90	1.62	1.71	310 " 315	0.78	...
90 " 95	1.87	0.62	315 " 320	0.89	...
95 " 100	1.62	0.62	320 " 325	0.88	...
100 " 105	1.75	0.80	325 " 330	0.66	...
105 " 110	1.87	1.32	330 " 335	0.88	...
110 " 115	1.52	1.24	335 " 340	0.75	...
115 " 120	1.62	1.31	340 " 345	0.77	...
120 " 125	2.00	1.12	345 " 350	0.30	...
125 " 130	1.66	1.12	350 " 355	0.66	...
130 " 135	2.20	0.45	355 " 360	1.06	...
135 " 140	2.18	0.52	360 " 365	0.53	...
140 " 145	2.17	0.71	365 " 370	0.40	...
145 " 150	1.92	1.20	370 " 375	0.40	...
150 " 155	1.79	1.72	375 " 380	0.93	...
155 " 160	1.40	2.34	380 " 385	0.79	...
160 " 165	1.94	2.08	385 " 390	0.80	...
165 " 170	1.94	1.92	390 " 395	0.75	...
170 " 175	1.81	1.84	395 " 400	0.92	...
175 " 180	1.29	1.82	400 " 405	1.20	...
180 " 185	1.83	1.72	405 " 410	1.06	...
185 " 190	2.20	2.18	410 " 415	1.04	...
190 " 195	2.33	2.24	415 " 420	1.72	...
195 " 200	1.91	2.17	420 " 425	0.79	...
200 " 205	2.02	2.12	425 " 430	2.39	...
205 " 210	2.02	2.20	430 " 435	1.45	...
210 " 215	1.77	2.38	435 " 440	1.86	...
215 " 220	1.89	1.78	440 " 445	1.59	...
220 " 225	2.93	1.71	445 " 450	1.25	...

Jackling says that "an estimate of 25,000,000 tons, in addition to that already developed, is conservative as a probability of further production. This estimate is based upon a supposed width, at the northerly end of the property, of 600 ft., at the southerly end 1000 ft., at the middle 2400 ft., with a length of about 3000 ft., and a depth of about 110 feet." All these expectations have been more than confirmed by mining operations. His synopsis is as follows:

Ore at present developed	12,385,000 tons	
Average assay value	2% copper	
Average thickness of waste to be stripped	40 feet	
Average thickness of orebody developed	110 feet	
Estimated cost of stripping waste, per cubic yard	75 cents	
Estimated cost of mining ore, per ton	40 cents	
Estimated cost of hauling ore to mill, per ton	15 cents	
Estimated cost of concentration, per ton (on 2000 tons per day basis)	45 cents	
Estimated total cost of mining and concentration, per ton	$1.14	
Saving made by concentration on 2% ore	75%	
Estimated cost of smelting and refining concentrates:		
Per ton of concentrates	$11.39	
Per ton of raw ore	0.71	
Profits per ton of raw ore (price of Lake copper, 18 cents per pound)	2.83	
Actual cost of producing one pound of copper	0.06	
Profit per month, on basis of 2000 tons daily capacity, taking price of Lake copper at 14 cents	$90,000	
Profit per month at present price of copper (18 cents per pound)	162,000	
Estimated cost of mine equipment		$21,600
Estimated cost of railroad and equipment		232,000
Estimated cost of concentrator		356,000
Estimated cost of smelter and refinery		223,000
Estimated cost of steam-power plant		170,000
Estimated cost of shops, etc		20,000
Total		$1,022,600

It will be noted that the actual cost of producing copper is estimated at 6 cents per pound. In this estimate no charge is made for interest on the investment, only ordinary repairs on the equipment being considered. Including depreciation and interest on investment, the cost is estimated at about 7 cents per pound. The appraisal of the mine is based on an average price of 14 cents for Lake copper, which at that time was the standard and was worth nearly a cent per pound more than electrolytic copper.

In describing the tests made in the Rogers mill, Mr. Jackling states that the equipment consisted of five 650-lb. stamps, two Wilfley tables, and a 6-ft. vanner, the last, however, "was scarcely used at all, as the belt was not in good shape, and it was impossible, with the appliances at hand, to separate the slimes and take them direct to the vanner without using such an excess of water that the work of the machine was worthless." The ore was classified roughly in a hydraulic sizing-box, the coarser material going to one Wilfley table and the remainder, with a large excess of water, to the other table. The mill-tests may be summarized:

Weight of sample lb.	Crushing mesh	Assay: Heading %	Assay: Tailing %	Assay: Concentrate %	Recovery %
418,800	20	1.42	0.43	22.16	58.0
258,400	25	2.04	0.51	23.69	76.5
68,200	30	1.70	0.45	20.98	103.4
149,000	30	1.84	0.34	22.85	97.7
151,600	25	1.37	0.42	16.15	58.9
191,400	25	1.08	0.32	18.95	67.36

The last two tests were made on dump ore, partly leached. It will be noted that the better results were obtained by finer crushing. Discrepancies between recoveries calculated from the differences between the heading and the tailing and between the heading and concentrate produced are due, says Mr. Jackling, "to carelessness in handling concentrates and allowing them to become mixed, and, in this way, weighed and credited to the wrong test." This applies particularly to the third test. The summary of results is given as follows:

Total dry weight of ore milled	574.82 tons
Average assay-value in copper	$1.55
Total dry concentrate	29.339 tons
Average assay of concentrate	21.75%
Recovery, by assay of ore and concentrate	71.70%
Concentration	19.59 into 1

The actual returns from the sale of the concentrate to the Germania smelter, near Salt Lake City, showed 57,000 lb., containing 54,601 lb. net dry weight, assaying 22.10% copper, 0.12 oz. gold, and 1.40 oz. silver; the contents therefore being 12,067 lb. of copper, with 3.27 oz. gold, and 38.22 oz. silver per ton. Payment was made on 94% of the copper at $2.50 per unit, or 12½c. per pound, making $1419.60. All the gold was paid for at $19 per oz., making $62.24. On the silver 95% was paid at 59¼c. per oz., making $21.56. Thus the total valuation at the smelter was $1503.40, equivalent to $55.07 per ton of concentrate. From this a smelting charge of $7.98 was deducted, leaving $47.09 net per ton of concentrate, or $1.94 per ton of crude ore. Mr. Jackling estimated the total cost of mining at 54 cents, of haulage to the mill at 15 cents, and of concentrating at 45 cents, making a total of $1.14 per ton. He assumed a recovery of 75% on a 2% ore and a ratio of concentration of 16:1. The cost of roasting, smelting, and refining he estimated at $11.73 per ton of concentrate. This concentrate would yield 451 lb. copper, 1.26 oz. silver, and 0.114 oz. gold; making an aggregate value of $75.26, as against a total cost of $29.97, leaving a gain of $45.29 per ton of concentrate or $2.83 per ton of ore milled.

While this examination was in hand, Clement got into a wrangle with De Lamar over his one-eighth interest, the Captain claiming that the Wall business had been introduced by Cohen, of his own staff, while Clement insisted that the Captain's staff had turned it down and that he himself had initiated the later negotiations, which had proved fruitful. Clement resigned and Cohen came back. Again the Captain dropped the deal, save for the quarter he had bought at $50,000. He had also spent $46,000 in the course of the examination, exploration, and testing. It is likely that his quarrel with Clement was one reason for abandoning the option; another being a not unreasonable timidity at tackling a mining venture based on such low-grade ore and requiring so much capital to place it on its feet. Clement might have helped to dispose of an interest on advantageous terms in London, but that possibility was spoiled by his resignation.

In 1901 Clement went to Mexico; so did Gemmell. From Mexico, Clement wrote to Wall occasionally, assuring him of his continued interest in the mine at Bingham. He had an idea of persuading Volney Williamson of Spokane to join him and Wall in developing the big prospect. Meanwhile De Lamar retained the quarter interest that he had bought for $50,000 in 1899, and it was necessary to get him out. Thereupon Wall told Clement that if he would buy out De Lamar for $100,000, he would sell a quarter to Clement for $50,000, provided the property was incorporated and sufficient capital raised for development and equipment. De Lamar's quarter-interest worried Wall; it spoiled his sleep. He obtained an option on it for 90 days at $100,000, but nothing came of that. An injunction obtained by Clement forbade delivery by the bank. Clement died at Saltillo, in Mexico, on April 26, 1903. I record the fact regretfully even 15 years after.

Now comes another turn of events. Mr. Jackling had taken part in the tests made for De Lamar on the 'porphyry' ore in 1898 and 1899; he had written the report that summarized the results of those tests and of the accompanying examination of the mine itself. At that time, however, he was junior to Cohen and Clement. He was then only 30 years of age, but he had already proved himself a capable and resourceful metallurgical engineer, and he had repeatedly advised De Lamar that the Bingham venture was full of promise. A masterful character, possessed of strong initiative, starting life poor and without friends, but a graduate of the Missouri School of Mines, he was beginning to feel his way to his natural position as a captain of industry. He realized that to turn Wall's prospect into a profitable mine it would be necessary to spend much money, at least $3,000,000; and in those days, before any of the low-grade 'porphyry properties' had been proved a basis for successful exploitation on a big scale, it was almost impossible to engage the attention of capitalists in such a venture.

Before proceeding with my history, I will give instances to corroborate this last statement. My friend James W. Neill, now in the far North-West of Canada, told me the following story a few months ago: "One day, in 1897, Colonel Wall came to my office at Salt Lake City and said to me: 'Jim, I have a copper mine,' and I said 'Colonel, where is it, and how big is it?' He said, 'It's in Bingham, and it's a mile wide and as long as a railroad, and it's got more copper in it than the Anaconda had at surface.' I said, 'Colonel, that's easy; the Anaconda had no copper at the surface.' Later Col. Wall gave me an option on his three-fourths interest in his claim, the chief of which were the Dick Mackintosh and Charles Read, and I also had an option on the entire holdings of what is now the Ohio Copper Co., then the Cook group. I offered these holdings to Benjamin Guggenheim, in New York. He referred me to his engineer, Hermann A. Keller. The answer that I got to my representations that there was a mass of ore there, as shown by the tunnel on the Dick Mackintosh claim, which ran $2\frac{3}{4}$% copper, was that the tailing from the Parrot mill, at Butte, which had been under Keller's charge, ran as much in copper, and he couldn't see where he could make any money on that class of ore; so, naturally enough, my proposition was not attractive."

Mr. Cohen tells me that he submitted the business to the late Benjamin Guggenheim in 1900 and to Charles A. Coffin, of the General Electric Co. in 1902, with his own report stating that the mine showed 18 million tons of 1.6% ore, concentratable in the ratio of 15:1. In 1902 he presented the deal to John Hays Hammond, shortly before Messrs. Penrose and MacNeill took hold. In the same year L. C. Trent had an option and offered it to the Tharsis Sulphur & Copper Co., of Glasgow, Scotland. Colonel Wall tells me that an engineer named Wilson examined the property for Marcus Daly in 1901, and that Joseph Clark visited Bingham in 1902, becoming so favorably impressed as to ask if he might submit the project to his brother, Senator W. A. Clark; but nothing happened. The fact that a mass of $1\frac{1}{2}$ to 2% copper ore was unattractive 15 to 20 years ago should surprise no one; it required constructive imagination of no common order and unusual financial courage to undertake the large-scale exploitation of such a deposit at that time.

To return to our story: Soon after the last examination of Wall's property, in the summer of 1899, Mr. Jackling resigned from Capt. De Lamar's employ and went to Republic, in Washington, where he designed and built a mill for Clarence McCuaig and other Canadian capitalists. In 1901 he went to Colorado Springs and became associated again with Charles M. MacNeill and his other former employers, now his friends, as consulting engineer to the United States Reduction & Refining Co. At the very start of his career, at Cripple Creek in 1894, he had made the acquaintance of MacNeill and also of Spencer Penrose and Charles L. Tutt, these three being the organizers of the U. S. Reduction & Refining Co., which operated two big mills at Colorado City, close to Colorado Springs, on ore coming from the mines at Cripple Creek. For this company Jackling re-built and managed the Bartlett zinc-pigment plant at Canon City in 1902. Meanwhile he had not forgotten the Wall property at Bingham. He spoke about it to his friends in Colorado and they undertook to back him; so, at the close of 1902, just before Christmas, while at Salt Lake City on other business, he called upon Colonel Wall and asked him for an option, but without success. It happened that both Jackling and Cohen were in Salt Lake City in connection with a suit brought by De Lamar over the interpretation of an electric-power contract made during the time when they were in the Captain's employ; so they met. Jackling discussed the Wall property with Cohen and stated to him that if he could persuade Wall to grant an option he (Jackling) could get his Colorado associates to provide the capital necessary to develop the enterprise. Cohen went to Wall and spoke of having New York friends that would be willing to find money for the Bingham property if Wall would give a reasonable option. Wall was willing to sell half of his holdings for $400,000, but he imposed conditions covering the equipment and development of the mine; and he demanded that a mill to treat 500 tons daily be built by the supposed New York buyers, who also were first to purchase De Lamar's quarter. The negotiations broke down until Cohen obtained the help of the Salt Lake banker, William S. McCornick, who aided Cohen in persuading Wall to come to terms. The result was that on January 23, 1903, Wall signed an option to Cohen on "two-fourths undivided interest" at $350,000 in cash, of which $50,000 was payable on March 9 and $300,000 on June 7 of that year. In this agreement Wall recorded his willingness to join in the organization of a stock company, retaining the right to nominate one member of the governing board. Cohen took the option to Jackling, who took it to MacNeill, in Colorado. Thereupon MacNeill, Spencer Penrose, and his brother R. A. F. Penrose, came with Jackling to Utah and visited Bingham. Cohen's agreement was replaced by a new one, in the name of Spencer Penrose. By the terms of this new deal Wall was to receive $385,000 for 55% of the entire property and the MacNeill-Penrose group was to buy De Lamar's quarter, leaving Wall with a 20% holding in both shares and bonds. The option was for six months, with the privilege of an extension for twelve months more on payment of $5000 in cash for each monthly extension of time. They did use seven months extra and for that they paid $35,000, so that Wall eventually received $420,000 in all. They bought De Lamar's quarter for $125,000.*

R. A. F. Penrose selected F. H. Minard to examine the mine. Mr. Minard's report is dated April 23, 1903. It is of particular interest to me because in 1899 he was my assistant in the examination of the Camp Bird mine, another famous bonanza. Mr. Minard brushes aside the non-essential geologic features, and places his finger on the dominant fact that the 'porphyry' has intruded into the limestone and quartzite, and that "with the exceptions noted on the north and east, the entire property is in the porphyry, and it is this formation which carries the copper." He says: "Copper occurs in this porphyritic formation in small particles of iron and copper pyrites which are very finely disseminated through its entire mass." He states that Bingham canyon has been cut, by erosion, through this copper-bearing intrusive mass, so that exploratory work, by means of tunnels aggregating 2500 to 3000 ft. in length, has afforded facilities for sampling. His samples were taken with hammer and moil, at intervals of 10 ft. "Each section was stripped and cleaned of all loose rock and copper stains, and the sample taken of the hard fresh rock, avoiding all seams or veins of ore. The cuts were made up both sides of the tunnel and across the top, and each sample weighed 50 pounds, and some of them heavier. These samples were all sacked and sealed immediately after taking; they were then dried and broken to the size of almonds and quartered twice, then broken to the size of small peas and quartered, making duplicate samples, one of which was sent to Colorado City, and the other to Canon City for analysis." Mr. Jackling was stationed at Canon City, Colorado, at that time. The assay-plan showed a zonal occurrence of the copper, as also did the two previous samplings of the property. "All the maps," he says, "show that there is a distance of low values for about 50 ft. from the portals, then they rise rapidly on going farther into the hill and maintain a high average to a point 260 ft. in, and beyond that point they fall to an average which is between the first low and the succeeding high averages, showing conclusively that, following the contour of the hill for a distance of 50 ft. and normal to the slope of the hill, there is a zone of the porphyry which has been leached, and the value of the copper contents is not over 0.75%; that immediately below this leached zone is another zone which has been enriched, that is, the copper which was leached from the top zone has been carried down by the water and re-deposited in the zone below, and enriched the original rock which formerly carried only the same percentage of copper as is now contained in the zone below the enriched one. This enriched zone averages 2% copper and extends for 100 ft. below (normal to the slope of the hill) the leached zone. This enriched zone is a little thicker—probably 150 ft.—at the bottom of the canyon, as is shown by the horizontal distance on the maps showing the drill-holes. These holes

*Capt. De Lamar died soon after this was written. See Mining & Scientific Press of December 14, 1918, which contains an editorial appreciation.

were sunk near the bottom of the canyon. This is due to the constant tendency of the water to gravitate toward this level, making a larger quantity [of ore] here than on the sides. Another and equally important proof of this enriched zone, is the occurrence in the same horizon of high values of the well recognized secondary minerals of copper, namely covellite and bornite. The zone of material below the enriched horizon is the original rock unaltered by surface erosion or leaching, and averages 1.1% of copper, and constitutes the bulk of the deposit."

I give Mr. Minard's own words because it is well to place them on record. He recognizes a leached zone that extends 50 ft. from the surface, and within this zone the monzonite averages 0.75% copper. Next comes a zone of enrichment, 100 to 150 ft. thick, within which the copper abstracted from the surface zone has been precipitated upon the primary deposit, raising the copper content to 2%. Underneath the zone of secondary enrichment is the primary deposit containing 1.1% copper.

The average of all his samples was 1.6%, this being the average of the two upper zones, namely, those of leaching and enrichment. He did not estimate any ore below the zone of enrichment nor did he include any ore above the level of the creek, but only for 300 ft. above the creek, for 150 ft. at right angles to the hill-slope, and for 1600 ft. north and south. Thus he concluded that it was "reasonable to suppose" a mass of 9,000,000 tons of 1.6% ore available.

The only obstacle to success was the possible insufficiency of water to treat this ore by concentration. He considered that at least 2000 gallons was required per ton of ore treated, and referred to two possible sources of supply. The last paragraph of the report is as follows: "The property in question has excellent merit, but one cannot be enthusiastic over it on account of the water problem. There is no doubt as to the existence of the ore, under the conditions already noted, which can be readily and cheaply mined, and if the water supply can be satisfactorily determined I would recommend the erection of a 200 or 300-ton plant, to make extended experiments covering a period of at least a year, only upon the condition that you can acquire an interest in the property for the building of this plant without any payment whatever. It would also be advisable to acquire at the same time an option on the remaining interest. You would then be in shape to further develop the property, keeping constantly in mind the distribution of the ore along the lines as explained in this report."

The samples taken by Mr. Minard were shipped in duplicate to Canon City and Colorado Springs, where, after re-sampling for assay, they were collected and forwarded to the mill at Colorado City, to be subjected to a concentration test, which confirmed the results obtained in the stamp-mills at Bingham.

On June 4, 1903, a company was organized, called the Utah Copper Company, under the laws of Colorado, the capital being $500,000, in shares of $1 each. Meanwhile an experimental plant of 300 tons capacity—called the Copperton mill—was built at Bingham. It was completed in April 1904. Shortly afterward Messrs. Penrose and MacNeill bought De Lamar's surviving interest of one-eighth and re-organized the company, this time incorporating in New Jersey with a capitalization of $4,500,000 in $10 shares. The date of this second incorporation was April 29, 1904. On July 1, 1904, there followed an issue of $750,000 worth of 7% bonds to run for three years and convertible into stock at par. These bonds were largely underwritten by the promoters themselves, for the Colorado group had made a good deal of money at Cripple Creek and 'Dick' Penrose, the distinguished geologist, owned a third of the Commonwealth mine, at Pearce, Arizona, which had yielded several million dollars in profit. Here I may mention that $250,000 was all the cash that was put up at the start in order to launch the Utah Copper enterprise. Colonel Wall was left with $150,000 in bonds and 90,000 shares of stock. Jackling and Cohen received a 5% commission, which was paid in stock. They still had to find the working capital—several million dollars—required to develop and equip the mine on an adequate scale. In the summer of 1904 the purchase of a block of stock was considered by, among others, the General Electric Company, for whom D. M. Riordan, assisted by E. Gybbon Spilsbury and W. Lawrence Austin, made an investigation, and reported favorably. Mr. Spilsbury's measurements showed that there was "at least 5,000,000 tons of 1.98% copper ore developed." He says† that the Copperton mill "had been brought to the point of profitable operation, the profits based on 13c. copper amounting even at that time to 70 cents per ton of ore treated." These estimates of Mr. Spilsbury fully corroborated the earlier figures of Messrs. Jackling and Minard; nevertheless they were received with skepticism not only by his own clients but by others that heard of them; indeed it was insinuated, so he tells me, that he had succumbed to the undue influence of Messrs. MacNeill and Penrose! However, when the business was heartily recommended by Mr. Riordan, on the strength of Mr. Spilsbury's report, to the directors of the General Electric, they seemed inclined to close the deal, until one of them remarked that he did "not believe the damn figures." That undoubtedly was the mental attitude of many others to whom the business was presented. In 1906 the capitalization was increased to $6,000,000. During this year the Denver & Rio Grande Western railroad extended its line from Bingham to the mill-site at Garfield. This work was completed early in 1907; the first train of ore being hauled from Bingham to Garfield over the new line on April 19. The Bingham & Garfield Railway, controlled by the Utah Copper Company, was completed and the first ore shipped over this line on September 14, 1911. In 1906 the remainder of the original bond-issue of $750,000 was retired, at a premium of 5%, in order to clear the way

†In a letter to the writer.

for a new issue of bonds, namely, \$3,000,000 at 6%, this money being required to build the Magna mill at Garfield. This issue was convertible into stock at \$20 per share. The bond-issue was underwritten by the Guggenheims, in the name of the Guggenheim Exploration Co., and they, at the same time, acquired a block of stock. In February 1907 Hayden, Stone & Co. underwrote the purchase of 60,000 shares of stock at \$25 per share, making \$1,500,000, and in June 1908 the same firm of brokers underwrote an issue of \$1,500,000 of bonds convertible into stock at \$20 per share, these bonds being sold in order to provide funds for expanding the capacity of the mill. Thus the capitalization of the company became augmented to \$7,500,000.

Two years later, in January 1910, the capitalization was increased to \$25,000,000, of which \$16,244,900 in \$10 shares has now been issued. Out of the stock issue, amounting to \$8,282,240 made in 1910, the sum of \$3,100,000 was paid for the property of the Boston Consolidated, and \$4,455,120 for 1,000,152 shares of Nevada Consolidated, the latter being a highly successful copper enterprise at Ely, Nevada. From the day when production began, in 1907, to the end of 1917, the mine has yielded 67,220,700 tons of ore, averaging 1.428% copper, producing 3,118,385 tons of concentrate, averaging 19.81%, and containing 617,785 tons of copper, enabling the company to pay \$75,770,882 in dividends and accumulate a working capital of \$48,293,528.

Colonel Wall was elected a director of the company when it was organized in 1903, but he resigned in 1908, owing to friction with his colleagues on the board. The result was to create bitter feeling between him and Mr. Jackling, a feeling that found vent in a journalistic vendetta, for Wall started a paper called 'Mines and Methods', which was published at Salt Lake City and under the not too scrupulous editorship of Claude T. Rice waged bitter war against Jackling and his associates from September 1909 to August 1913, when it ceased to appear. This animosity was due, as far as I can learn, not to personal wrong but to intense disappointment, on the part of Colonel Wall, in not being made manager of the company and in seeing the credit for the creation of the enterprise pass to Mr. Jackling, whose virile personality increasingly dominated the operations. One cannot withhold some measure of sympathy from the old miner, who stuck to his dreams so long and showed such patience in consolidating the property first, and then in nursing it during the long years that passed before his dream came true; but such sympathy is not inconsistent with admiration for the persistence and ability of the younger engineer, who, with equal faith in the prospect, brought the dream to fulfilment.

THE MINE

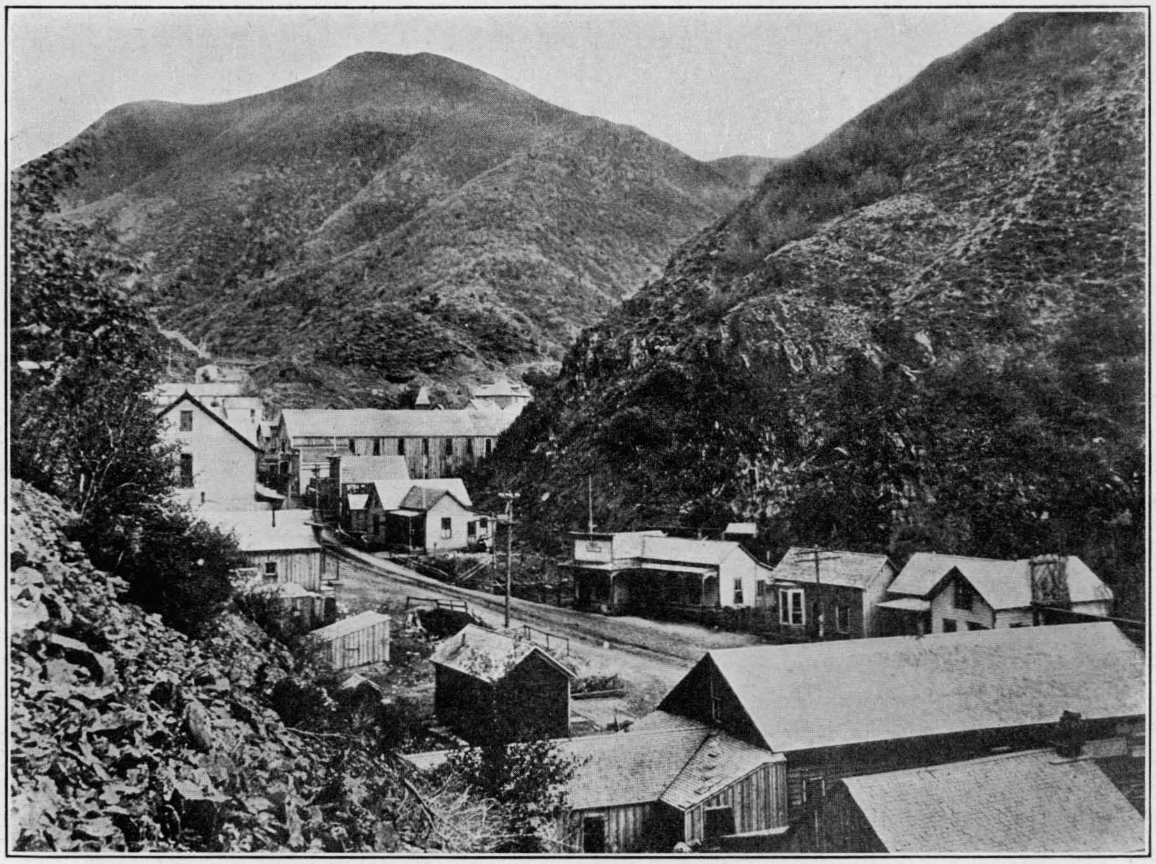
INTRODUCTION. It was 5:25 a.m. as I lay awake expecting the call that was to arouse me from sleep. The air was vibrant with the singing of birds in the Temple gardens. That sounds poetic and Oriental, but, I hasten to say, the temple was Mormon and the singing of birds was the twittering of sparrows, holding their morning concert before the roar of city traffic drowned their effort to express the joy of life. The quiet of an unawakened community brooded over Salt Lake City. I started to dress, feeling superior, for at Ogden, only 38 miles westward, the hour was 4:30 and if the Daylight Saving law had not played fast and loose with Greenwich time it would have been 3:30. I felt like an early bird and had none of the qualms proper to the matutinal worm.

In due course, after breakfast at the Union Pacific station, I caught the 6:55 train for Bingham, which is at the far end, 36 miles distant, of the Bingham & Garfield railway. The sun was beginning to assert himself over the rim of the Wasatch range as the train started. Every seat was taken, for the train carried 700 men from Salt Lake City to their work at the mills on the other side of the valley. They looked like a pleasant lot of fellows, such as, for example, one would not be afraid to be shipwrecked with. Most of them were reading their morning papers, of which Salt Lake has several, and of a good kind. The Hearst infliction has not been visited upon the Latter Day Saints, as yet. Looking around, I estimated that only 10% of the men were foreigners. They pay $10 per month for their commutation, which takes them daily to and from the Arthur and Magna plants of the Utah Copper company, at the foot of the Oquirrh range, which, with the Wasatch mountains, encloses this southeastern edge of the Dead Sea of America. The valley itself is cultivated with the thrift characteristic of the Mormon people, but much of it is marshy with salt water and defies reclamation. Soon I noticed an embankment like a railroad-grade on the left of the train and wondered what it was until I detected pools of green water at the foot of it. The effluent water was stained with copper; it was the tailing-dam restraining the discard from the big metallurgical works. This dam cut off the view southward, but the smoke now veiling the snow of the Oquirrh mountains suggested intensive industry. Soon the train enfiladed the series of buildings enclosing the Arthur mill, and more than half the passengers alighted. Within five minutes the Magna plant was reached, the train was reduced in length, and the remaining cars proceeded half-empty to Bingham. The railroad follows the eastern face of the Oquirrh range; the sun shimmered across the vast stretch of tailing, estimated at 65,000,000 tons averaging 0.5% copper, as dead metallurgically[1] as the salt waste that it blankets. Beyond Magna the country is more gracious; fields of young grain and alfalfa in clean patches of productive verdure fringe the saline plain at the base of the hills that mark the older shore of the shrinking lake. A group of buildings and a chimney belching smoke mark the settlement of Bacchus, a name not without irony in this 'dry' State, but signifying a stimulant more active than the juice of the grape, for at Bacchus is the Hercules Powder Company's factory, producing the dynamite used at the big mine to which I am taking you.

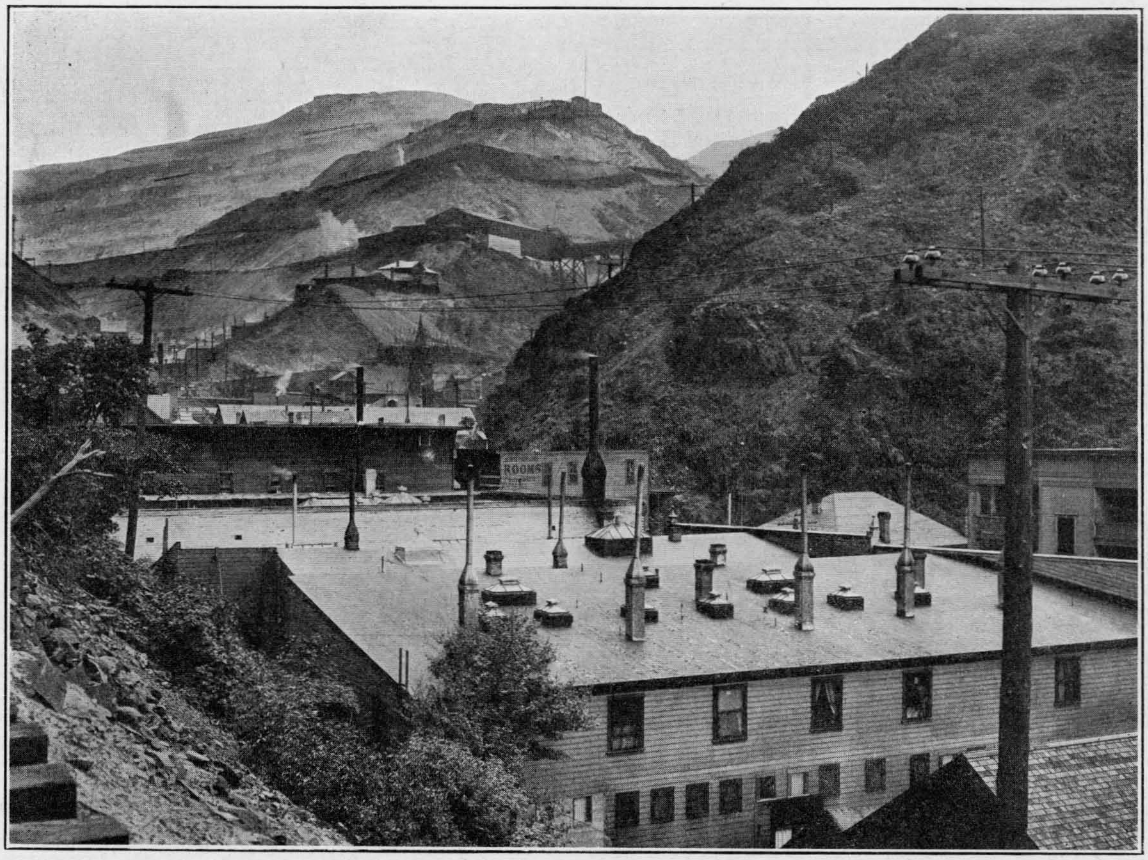
The scenery was tame. The rounded hills, from which the snow had but lately gone, were freshly green. A train of 42 cars laden with ore passed noisily. The gradient increased; excavations on the higher slopes indicated the approach to a mining district; the canyon narrowed, on the opposite side I saw another line of railroad;[2] the train plunged through four murky tunnels and then emerged suddenly into the smoky obscurity of Bingham. The town itself was not discernible, the terminus being high on the mountain-side, and when I alighted I followed the other passengers into what looked like an elevator, but proved to be a tram-car running on a steep incline into the town. An inquiry proved that I had blundered, for the company's office was above, on the level of the railroad, so I ascended by the tram and found my way to the warehouse where Mr. J. D. Shilling, the superintendent of the mine, has his headquarters. My letter of introduction having been presented, I was conducted to the upper gate of the railroad-yard. Passing through the warehouse one could see barrels containing copper wire, to be used for the peaceful purpose of electric conduction; also copper rings resembling the bands that are put around shells, suggesting the warlike use of the metal. These rings, I ascertained, were 4-inch

[1]The Assessor for the county has made an effort to value this mass of tailing for taxation, even going so far as to place an arbitrary valuation of $50,000,000 upon it.

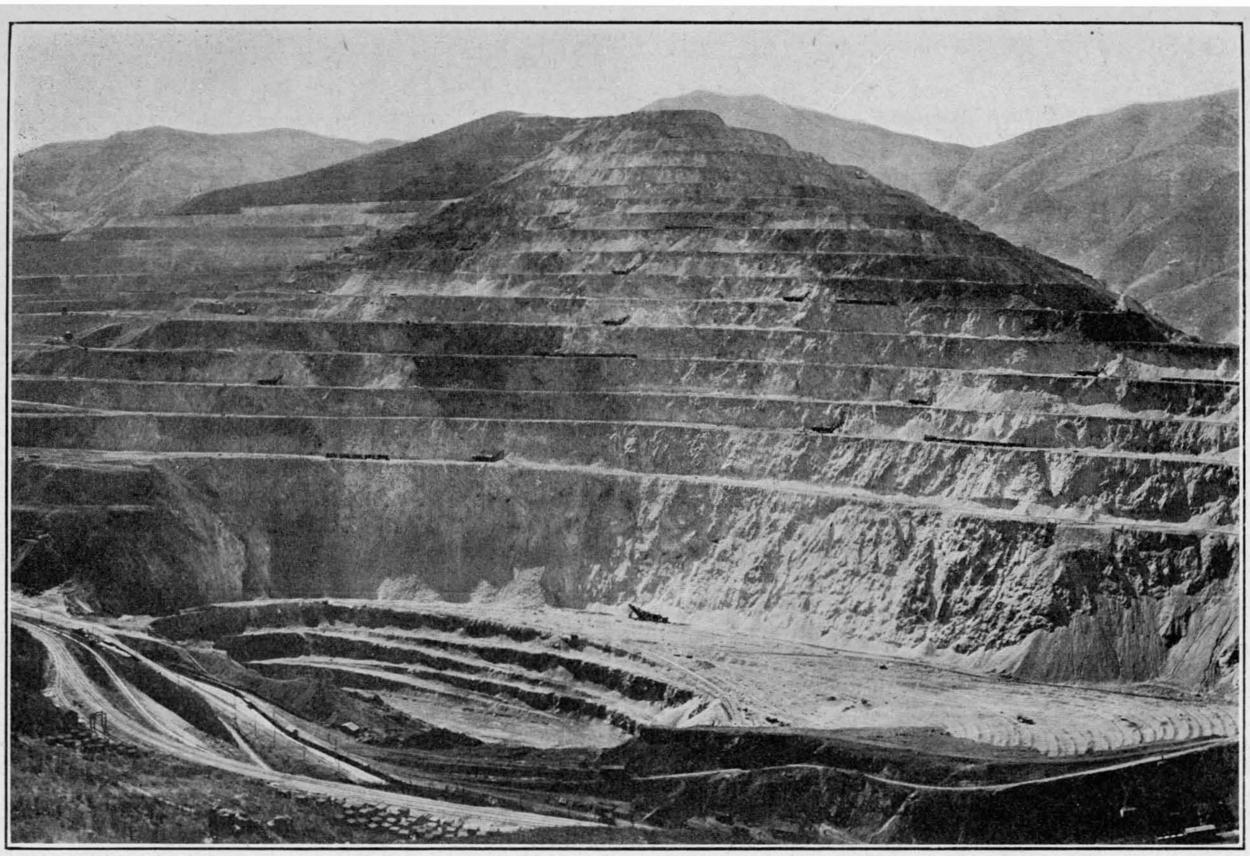
[2]The Denver & Rio Grande, which transports 7500 tons of Utah Copper ore daily.



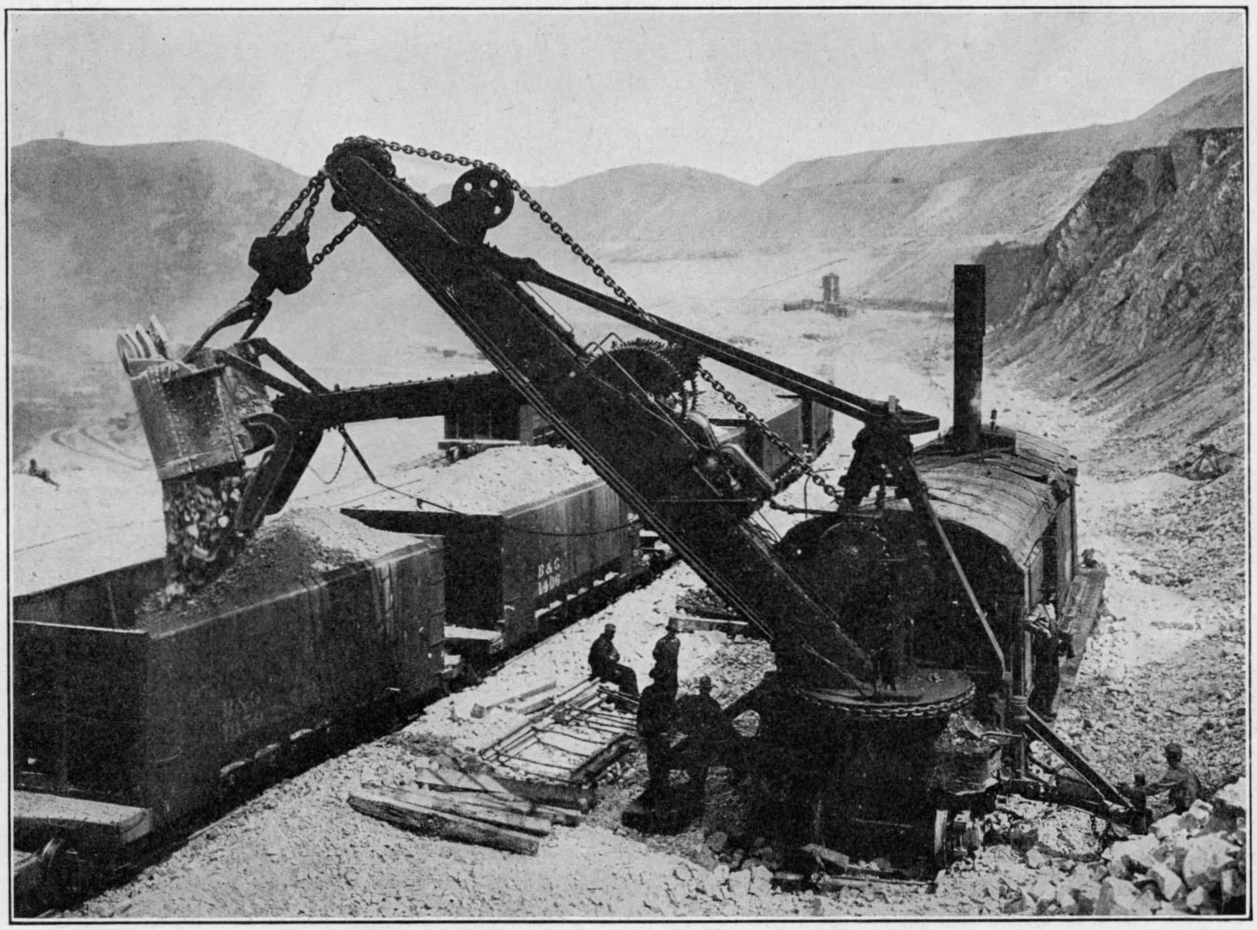
THE COPPER MOUNTAIN AS IT APPEARED IN JULY 1900, FROM THE JUNCTION OF BINGHAM AND CARR FORK CANYONS



THE SAME VIEW AS IT APPEARED IN JULY 1918



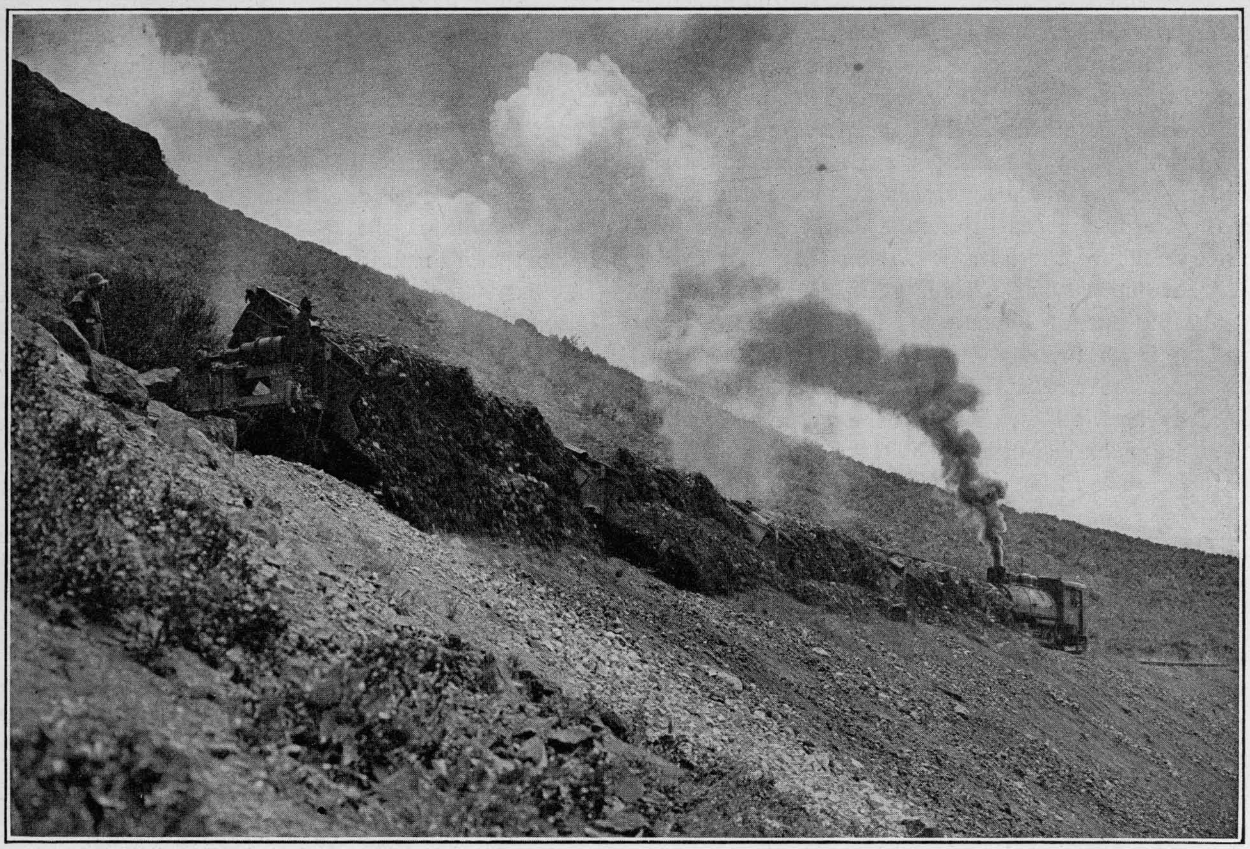
THE HEART OF THE UTAH COPPER MINE



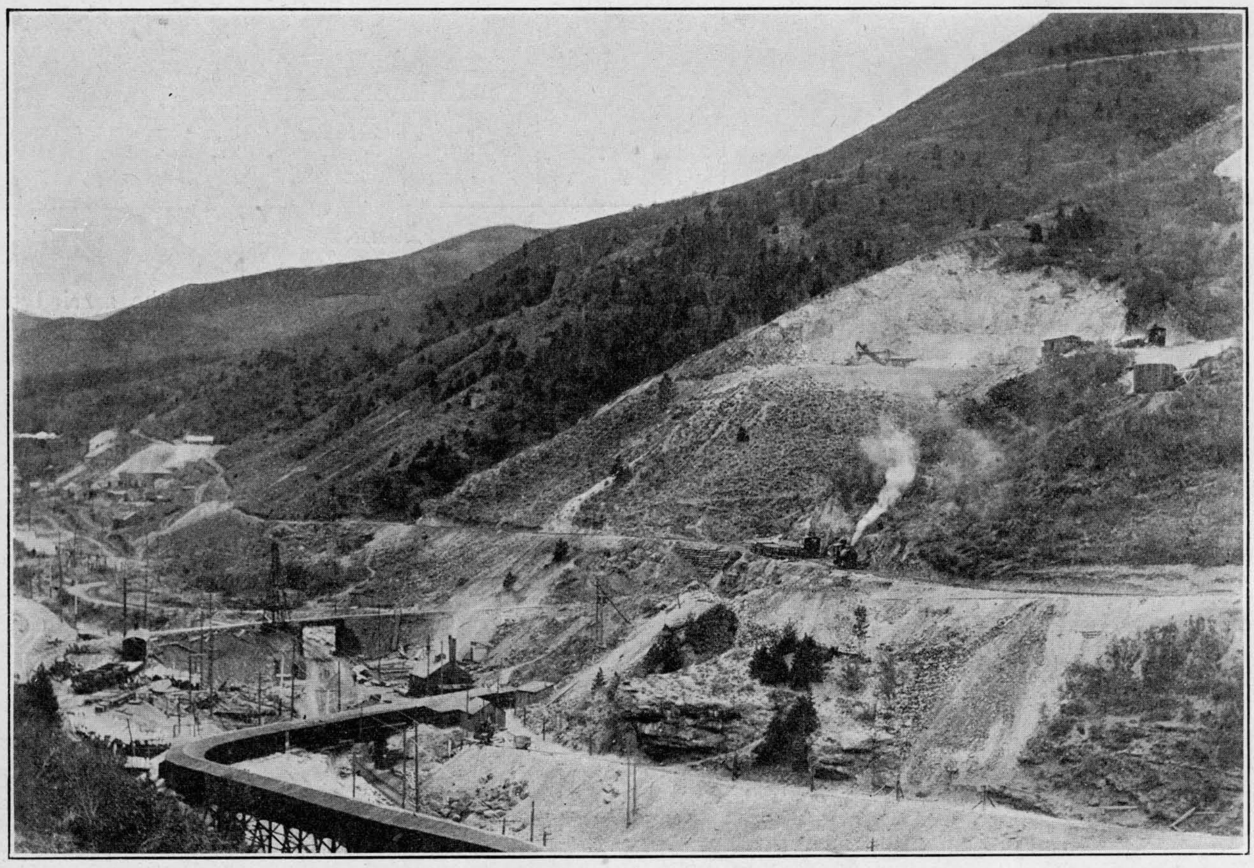
STEAM-SHOVEL LOADING ORE INTO RAILROAD-CARS

AN INGERSOLL DRILL AT WORK

GENERAL VIEW, SHOWING THE TOWN OF BINGHAM IN THE FOREGROUND, THE UTAH COPPER MINE IN THE BACKGROUND, AND THE BINGHAM & GARFIELD RAILROAD IN THE CENTRE. THE WHITE DUMP ON THE LEFT IS PART OF THE OXIDIZED ORE OF THE 'CAP'; THE TERRACES EXTENDING AWAY FROM THE MINE WORKINGS ARE DUMPS OF 'STRIPPING', OR WASTE ROCK, IN GALENA GULCH, ON THE LEFT, AND IN INGERSOLL GULCH, ON THE RIGHT.



THIS LOOKS LIKE A LANDSLIDE, BUT IT IS A TRAIN OF CLARK CARS UNLOADING CAP-ROCK ON THE DUMP



THE FIRST STEAM-SHOVEL STARTING WORK ON THE WEST SIDE OF BINGHAM CANYON IN AUGUST 1906 ferrules, used as a filling, or 'skim', at the point where the flues join the crown-sheet of the super-heaters of Baldwin locomotives. On the shelves was $30,000 worth of brass fittings for steam-machinery. Thus some of the uses of copper were indicated at the place where it was being mined. From the railway station one could see, through the smoke made by trains and steam-shovels, a mountain deeply scarred—the edge of the great mine—and from it, above the haze, a glorious flag flew assertively. That flag, 20 by 40 ft., cost $156, and was bought with contributions, of 25 cents to $1 apiece, made by the workers on the occasion of the campaign for the Third Liberty Loan. After a short walk along the railroad-track and over a viaduct, I reached the upper gate, where Mr. T. S. Carnahan (Columbia '05) met me and led me to the upper office at the mine itself.

ore. Such has been the dream of the prospector from time immemorial. Mount Morgan in Queensland, Australia, was a mountain of gold ore, in so far as its summit was excavated bodily and sent to the mill, although now the output comes from a definite lode-channel within the core of the hill—500 ft. high—on which the outcrop was discovered. Mount Lyell and Mount Bischoff in Tasmania are the sites of big mine-workings that follow orebodies rich in copper and tin, respectively; Mount Davidson was enriched by the Comstock lode, and the Rammelsberg by its famous deposit of silver; but none of these was a 'mountain of ore' so truly as this great quarry, the face of which is 1500 ft. in vertical height and 3600 ft. wide at the base.

To realize the bigness of the mine, it is best to ascend half-way up the opposite hill-slope, on the east[3] side of

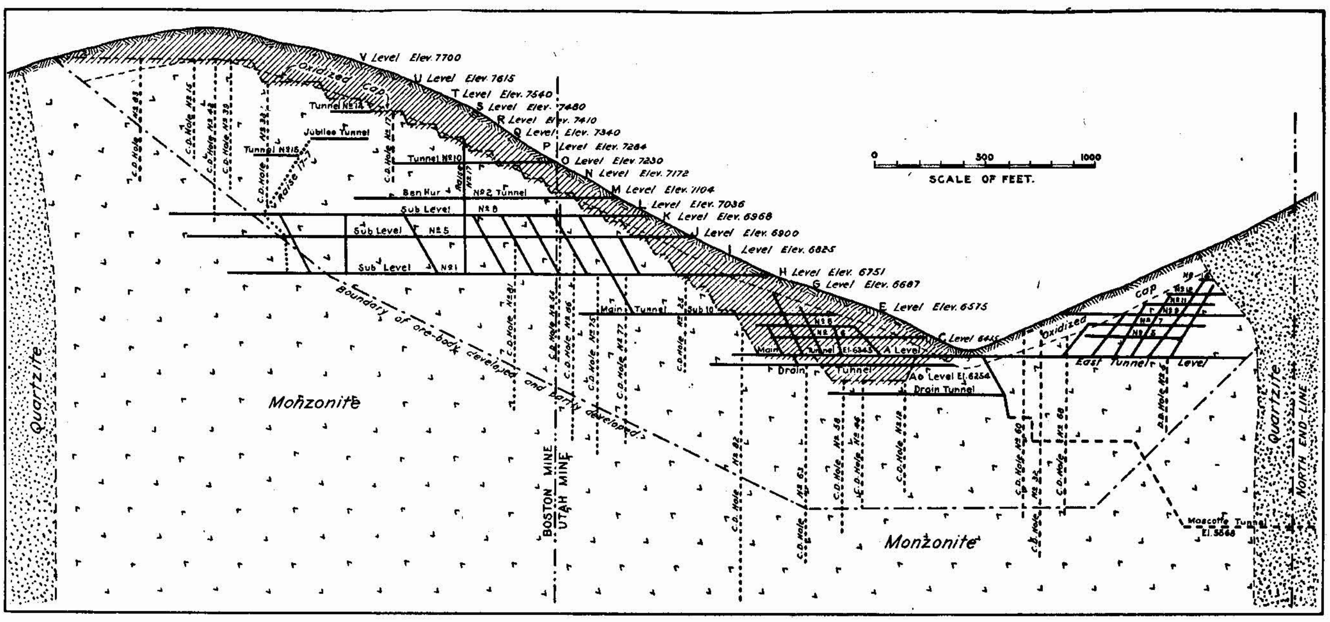


FIG. 1. CROSS-SECTION OF THE UTAH COPPER MINE

Arriving at the office of the engineering staff, I noticed on a lower door a sign in Greek:

ΕΙΔΟΠΟΙΗΣΙΣ
ΔΕΝ ΧΡΕΙΑΖΟΜΕΘΑ ΕΡΓΑΤΑΣ

Nobody seemed to know what it meant, but upon entering the lower office it was explained that the first word stood for 'Notice' and the script meant that no laborers were wanted. The men employed at the mine are of many races, nations, and creeds. Of the 1800 employees at this mine, 600 are Americans and 1200 are foreigners, who are classified as follows:

Japanese and Koreans	416
Greeks	406
Italians	151
Armenians	72
Albanians	55
Fifteen other nationalities	100

THE MINE of the Utah Copper company is a mountain impregnated with copper—a veritable mountain of

Bingham canyon; there one faces the serried terraces that mark the successive slices now being cut out of the mass of copper-bearing rock. It is an impressive picture of highly organized human industry. The crest has been removed already, but, foreshortened by distance, the mountain still looks like a pyramid, the levels of successive excavation suggesting the step-like cross-section of one of the famous Egyptian tombs. The outer cover of the hill is colored red, by oxidation; the mass itself is gray. It is a huge theatre, in which the actors are 1800 men; but so big is the stage that they are hardly discernible at this distance. Ore-trains, like children's toys seen from afar, run along the levels and black steam-shovels vomit puffs of smoke as they dig energetically into the piles of broken rock made by blasting. The smoke from the engines and the little black figures here and there

[3]The canyon runs north and south, except between the settlements of Upper and Lower Bingham, where its course is east and west for 1½ miles. See Fig. 2.

give a touch of the infernal to the picture, but the suggestion is contradicted by the blue sky that canopies the scene. On top of the hill, like a redoubt, is a tank to which water is brought by gravity from Middle canyon, four miles west, to be conducted to the different levels for drinking, for the boilers, and for other purposes.

At noon, from this point of vantage, I watched the blasting. Before it begins, those in control of the operation on the various terraces indicate the fact that all is ready by blowing one long and three short calls from the whistle of a steam-shovel. When all, in turn, have given this notification, the main whistle, blown by compressed air, sounds a general alarm, consisting of an irregular number of long and short calls. This is the signal for the men in the pit to 'spit' their fuses. As soon as the men on the level above see that the fuses in the pit are lighted, they spit theirs, and so in succession up the series of levels or terraces. Meanwhile the main whistle continues to sound the warning at frequent intervals. As the blasting is completed each steam-shovel gives two long whistles, and when all have reported, the main whistle answers with two long calls, thereby notifying all the men on the hill that the blasting is finished.

First a series of shots is heard in the pit, these representing the blasting of big rocks—'block-holing.' Then a less noisy explosion is followed by a rush of broken rock down the face of one of the terraces. The most effective holes make the least noise. The rumble of running ground is heard above the successive explosions. Some of the holes emit smoke like a cannon. Fume and dust enliven the scene. Everything is on a big scale; as much as 25,000 tons of ore has been broken in one blast, of 10 or 12 holes. Soon the uproar shifts to the right shoulder of the mountain and over it to the other side, awakening fresh echoes in the background of hills.

South and eastward the levels are extended for the disposal of the waste that is stripped from above the ore, for most of the oxidized cap has lost so much of its copper by leaching as to be merely an overburden. At the beginning of this year 121 acres had been stripped of capping, and 138 acres additional had been partly stripped. The dumps of waste represent a face of serrated ground almost as big as the mine itself.

It is a big mine in the open air. Winter does not seriously hinder operations; hardly 24 hours is lost per annum, although last winter twice, for half a shift during blizzards, it was impossible to move the ore-trains until the tracks were cleared by the snow-plow. Being open to the sky, the men are visible all the time; this is not without a salutary influence during labor troubles. There is nothing secret about these *al fresco* operations, which are always open to public view.

GEOLOGY. The mountain lies between the forks of Bingham canyon, the main ravine lying on the east side and Carr Fork on the west. Geologically the core of the deposit is monzonite, an igneous rock of porphyritic habit, that is, it has a groundmass of granular texture speckled by a characteristic mineral, like raisins in a pudding, in this case the phenocrysts being chiefly feldspar, but also biotite. Monzonite is a rock resembling granite; it is intermediate between syenite and diorite, the proportion of alkali feldspar (orthoclase) to soda-lime feldspar (plagioclase) being less than two-thirds and more than one-third. The monzonite-porphyry at Bingham has intruded from below into the older rocks, notably sandstone, but also limestone, in such a way as to bend or break the bedding-planes, changing the sandstone into quartzite and the limestone into marble. The plastic magma that, on cooling and crystallization, became monzonite exerted such pressure from below as to bend the sedimentary series into folds and other deformations, producing faults and fissures through which mineralizing solutions ascended. These solutions deposited their metallic contents as they approached the surface, where a lowering of temperature and a decrease of pressure favored precipitation. It has been shown by geologic investigation[4] that the deposition of the ore is closely connected with the intrusion of the monzonite into the quartzite, these two rocks constituting the prevailing terrain of the Utah Copper mine. It is likely that the process of enrichment was started by contact metamorphism, by which the copper minerals were deposited near the monzonite-quartzite contact and subsequently were segregated or concentrated at the contact and on both sides of it, chiefly in the monzonite, for a distance sufficient to begin the making of a big orebody. Hydrothermal activity, such as is the usual sequel to an igneous intrusion, caused mineralizing solutions to circulate along the fractures in the monzonite, particularly near its upper periphery, penetrating the mass of the rock so thoroughly as to effect a dissemination of metallic sulphides, namely, chalcopyrite and pyrite, over an area a mile long, more than half a mile wide, and to a depth greater than has been attained by the deepest drill-holes. All the augite and hornblende of the original rock have disappeared, most of the magnetite, and much of the feldspar. These minerals have been replaced, wholly or in part, by quartz, sericite, pyrite, and chalcopyrite. The replacement of magnetite by the metallic sulphides is the most significant fact. Some of the pyrite has been replaced by chalcopyrite.[5]

A later period of fracturing, followed by further hydrothermal activity, led to the formation of small veins of copper ore, together with an intense silicification and an increase of porosity throughout the mass of copper-bearing monzonite. As yet no mass of ore, that is, of rock rich enough in copper to justify mining, had been formed by the alchemy of nature. The copper was too thinly

[4]'Economic Geology of the Bingham Mining District,' by J. M. Boutwell. Professional Paper No. 38, U. S. Geological Survey. 1905.

[5]J. J. Beeson. 'The Disseminated Copper Ores of Bingham Canyon, Utah'. Trans. A. I. M. E. Vol. LIV, pp. 356-401. This paper, published in 1916, throws new light on the genesis of the orebody, and I am indebted to it for much of my information.

distributed to constitute 'ore' and would have been unworthy the miner's attention if a period of 'secondary enrichment' had not supervened. The intense fracturing of the rock had formed channels for the circulation of water from the surface; the sericitization, that is, the transformation of part of the feldspar into hydrous potash-mica, had increased the porosity of it; and the oxidation of the uppermost portion of the copper-bearing rock, by the seepage of rainfall and snow-water, had decomposed the chalcopyrite into soluble salts, which descended in obedience to gravitation and became precipitated at a lower horizon in such a manner as to augment the richness of the primary deposit. The secondary copper, precipitated by the reaction between the copper-sulphate solution and the primary chalcopyrite, took the form of the minerals covellite (CuS) and chalcocite (Cu_2S), the latter predominating in depth.

In order that an orebody of large vertical extent could be produced it was necessary that erosion should be less rapid than oxidation, thereby permitting the descent of freshly leached copper solution into the mass of primary mineral. That is what happened. The 'cap,' as now seen, consists of a few feet of soil and about 115 ft. of monzonite that has been leached, becoming brown, in contrast to the dark-gray color of the porphyritic copper ore, which extends to a maximum depth of 1800 ft. The average thickness of ore 'developed' and 'partly developed' at the end of 1917 was estimated officially at 538 feet.

The chemical reactions and operations by which the copper of the oxidized zone was leached and carried down so as to enrich the rock below is suggested by sundry observations made and recorded[6] by Mr. Beeson. When the cap is removed and the gray ore-bearing rock underneath it is freshly exposed, the gray ore appears wet, owing to the sulphate solutions percolating from the cap. In a short time the face of rock shows a green stain, extending for 20 inches[7] vertically, and in the course of a few days the evaporation of this acid moisture leaves a thin coating of crystalline copper sulphate. Again, Mr. Beeson noticed how a churn-drill, penetrating 45 ft., remained "scoured and clean" so long as it was in the brown cap, but in approaching the contact with the unoxidized rock it became spotted green, and when the drill had penetrated deeper it showed a coating of native copper, which must have been precipitated in the four or five minutes elapsing between the stopping and hoisting of the drill-bit, and must have been derived from the solution formed by the action of the drill-water upon the moist rock. On another occasion a candle lowered into a drill-hole burned brightly in the portion that was in the brown cap, but became extinguished near the bottom, which was in the gray ore. Even the rapidly reciprocal movement of the bailer did not mix the gas and air sufficiently to change the observed condition. No smell of sulphur gas was detected. It seems probable therefore that the oxygen of the air was being used in changing ferrous into ferric sulphate. A third observation indicated the formation of copper carbonates where water charged with carbon di-oxide came in contact with copper sulphate, such carbon di-oxide being produced by the decay of organic matter on the brush-covered mountain during the change of seasons. Every spring the snow-water carries the copper leached from the cap into the narrow subsidiary gulches. In one of these, the Apex Yard gulch, where the descending water percolates through old filling or waste, containing remnants of ore, it can be seen precipitating its red metallic copper upon scrap-iron at the rate of 15 pounds of metal per 1000 gallons of water.

By similar chemical reactions the copper leached in the zone of weathering was carried downward by the ground-water into the porous and fractured monzonite, there to be precipitated on the chalcopyrite, pyrite, and probably bornite, coating and replacing these primary minerals with covellite and chalcocite, the higher sulphides of copper.

The deposit is called a 'porphyry' orebody because the copper has impregnated an igneous rock having the porphyritic or speckled habit; it is also called a 'disseminated' copper deposit because the metal is distributed comparatively uniformly in minute particles throughout the mass; but to the geologist the Utah Copper orebody is an example of concentration rather than scattering; nature has brought together the copper from two sources, adding that which originated from below to that which was derived at a later stage from above. The net result has been to produce a mountainous mass of a billion tons of rock containing 1% copper per ton,[8] out of which the miner will select as much as it is profitable to exploit, such selection being determined by the cheapness with which he can break, transport, and treat the ore, and then refine the resultant metal. The operations of nature, mechanical and chemical, are supplemented by those of man, who is a great mimic, to the end that, among other things, the metal needed for munitions of war, for the conduction of electric energy, and for the transmission of speech may be available to that complicated system of life we call civilization.

The accompanying map shows the position of the Utah Copper mine in relation to the geologic structure of the locality. Most of the orebody coincides with a peninsular exposure of monzonite, but it also includes portions of the quartzite surrounding the monzonite. This rock extends eastward into the ground of the Ohio Copper, south-westward into the Utah Metals and Utah Consolidated (Highland Boy), and south-eastward into the Niagara mine of the United States Smelting, Refining & Mining Co.; but it has not proved ore-bearing anywhere outside

[6] Op. cit. pp. 299, 390, 394.

[7] Mr. Beeson says 20 ft., but that is probably an error.

[8] This is merely my own generalized statement, of geologic interest; the estimate of ore in the mine is given in the annual report and is quoted elsewhere.

the Utah Copper property except in the Ohio Copper company's ground, where, however, the ore now being won comes entirely from the quartzite adjoining the monzonite. See Fig. 1.

THE WORKINGS. Let us now visit the mine itself—the theatre-like quarry that I have tried to describe in a general way. Mr. Carnahan was my guide and to him I owe most of my information. At the base of the tiers of levels, averaging 69.5 ft. apart in vertical height, is the 'Sub A' pit, the bottom of which is at an altitude of 6249 ft. above the sea. A green pool of water on the floor of this pit suggests copper, but fails to indicate that an adit, called the Mascotte tunnel, 12,000 ft. long, conducts the drainage under the opposite ridge (on which we stood to get our view of the mine), into the Salt Lake valley on the other side. See Fig. 1. This adit serves as a motor-haulage level from the Ohio Copper mine to the concentrating-mill at Lark. The 'Sub A' pit, which forms the lowest level of the Utah Copper surface workings, is 681 ft. above this adit and drains into it as follows: a 6-inch churn-drill hole, 60 ft. long, connects the bottom of the pit with a drainage-drift, which, in turn, connects with the main shaft of the Ohio mine and through it with the Mascotte adit. From the 'Sub A' pit to the top of the mountain the vertical distance was 1600 ft., but now that the crest has been removed the top bench is 7700 ft. above sea-level. The lowest working in use is the 'A' pit, which is 105 ft. above the 'Sub A'. When we began our walk, a steam-shovel in the pit was puffing lustily as it loaded a train of five cars, each shovel, or 'dipper', transferring 7 tons from the pile of ore that had accumulated at the foot of the slope. The next, or F, level is 240 ft. higher. On the face of ore—a cliff of gray rock—one could see men, like black ants, so small they seemed in contrast to the size of the excavations. These 'bank-men' were 'barring down' loose ore

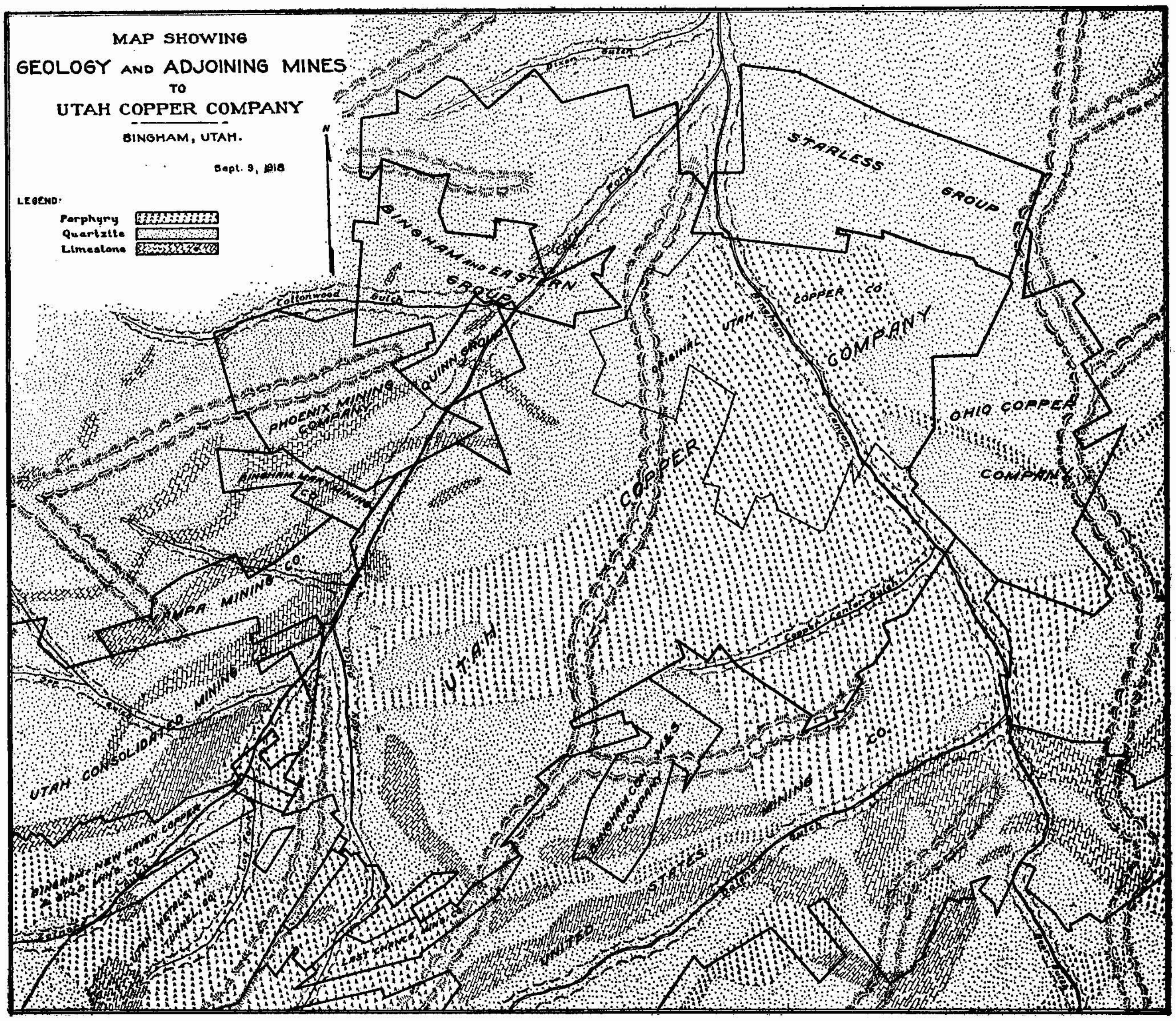


FIG. 2. GEOLOGY OF BINGHAM, AFTER U. S. GEOLOGICAL SURVEY, BUT REVISED AND CORRECTED TO DATE

so that it might descend and accumulate like an artificial talus on the floor of the pit ready for the steam-shovel. Like cats they clung to the precipitous face, holding to a rope suspended from above. The banks slope at an average angle of 50°. The terraces are from 40 to 250 ft. wide, averaging 100 ft. in width. These 'bank-men' are Italians, probably from the Val D'Aosta, mountaineers and smugglers such as I used to employ when manager of the Grand Clos silver-lead mine on the French-Italian border near Briançon in 1891. I noted a man pouring water from a barrel into a pipe leading to the drillers below, on the face of the cliff. Walking along the F level, we came to a group of men engaged in drilling. The crew consisted of a machine-man with two helpers. The drill was an Ingersoll 3½-in. machine, known as F 24. The holes are made 30 ft. deep in the floor of the terrace, and not quite as long—28 ft.—in the face. When starting, a box is put around the collar of the hole, in order to exclude loose bits of rock; this is covered with a scrap of paper (no part of a treaty) from the dynamite-box and kept in place by a stone until the time comes to load the hole and blast it—which time arrives when ore is needed directly below, that is, when the steam-shovel moves to that part of the pit. The dynamite is Red H, No. 4; it is 80% ammonium nitrate and 10% nitro-glycerine, but it is marked 60% because it produces the same proportion of gas as the straight 60% nitro-glycerine dynamite. The explosive[9] is manufactured, as previously stated, at Bacchus, by the Hercules Powder Co. Black powder is no longer used; it was employed in the short drifts, or 'gophers', that were driven into the hill at the start as part of the system of 'bank-blasting'.[10] On the top levels, however, it is still the custom to use black powder in the Keystone drill-holes. No. 3 Keystone machines are used to drill the bank on the two uppermost terraces. They have been found economical at these particular levels because the banks are relatively high and the terraces wide. During June 369,050 pounds of explosive was used in shifting 1,035,103 cubic yards of rock. The holes are chambered at the bottom so as to make room for the effective use of the maximum possible amount of dynamite at the bottom. The signal indicating that the hole is about to be 'sprung' is one long whistle followed by three short ones. The dust and gas shoot forth like a geyser. Two long whistles signify that the operation is finished. These signals, of course, are warnings to the men working in the immediate vicinity, particularly on the level below. All the way up the hill there are 'powder-men', who have to be informed before blasting of any kind can take place, in order to tell the workers when to move out of danger from flying stones. The man on the level above gives the warning and indicates the direction of safety. Three or four huts made of old railroad ties are distributed over each terrace, so as to afford shelter when a blast is in progress. Not many employees are hurt. A 'Safety First' campaign was organized several years ago, with the most beneficial results, and recently a 'safety engineer' has been appointed to supervise this department.

Looking at the nearest bank of ore, I noted that the general grayness was blotched with green where the copper sulphate, exuding by seepage, had dried. Toward the edges of the huge quarry the yellowish brown of the cap (oxidized monzonite and iron-stained quartzite) covered the gray ore-bearing rock. The best ore is a light-gray monzonite, the feldspar of which has been turned to sericite. At the south end is a darker and poorer monzonite, and to the north is the quartzite. The brown cap, with its copper carbonate, staining it in patches of green and blue, is broken independently, for separate treatment, as we shall see. Not much of the quartzite is ore. All that is not 'ore', and therefore 'waste', is removed into a neighboring gulch, so as not to interfere with the excavation of ore. As the eye ranges over the successive levels and envisages the scene, it is difficult to realize that 1800 men are at work, for only those near-by are visible. The younger Rockefeller (John D. Jr.), when here a year ago, said: "It is the greatest industrial spectacle in the world"—and he ought to know.

The hill, now cut and scarred beyond the recognition of the pioneers, is honeycombed with 60 miles of underground workings, which were made chiefly to explore the orebody and to ascertain its dimensions. See Fig. 1. This work was done between 1900 and 1915. For three years, from April 1904 to June 1907, all the ore that was milled came from underground stoping and development, but subsequently, as the stripping of the hill advanced sufficiently to permit of systematic open-air mining, a steadily increasing proportion of ore has come from the steam-shoveling operations, until in March 1914 no more ore was broken underground. Fully 6,250,000 tons of ore has been derived from the underground workings, besides 1,518,691[11] tons mined by the Boston Consolidated company before its mine was merged with the Utah Copper. The east hill opposite likewise has 27 miles of subterranean excavations, riddling the monzonite

[9]In the monthly statement of 'Coal Mine Fatalities in the United States for March 1918,' the following description of Explosive Red H No. 4 is given: "Volume of poisonous gases produced per one and one-half pounds, more than 53 to 106 litres. It belongs to the ammonium nitrate class containing a sensitizer that is in itself an explosive. It is permissible only when fired with a detonator (preferably an electric detonator) of not less efficiency than No. 6. The average weight of 1¼ by 8-in. cartridges is 164 grammes. It is permissible in diameters of ⅞ in. or larger. The unit of deflective charge is 248 gm. The rate of detonation in 1¼-in. cartridges is 8980 ft. per sec. (2738 metres per second). It is manufactured by the Hercules Powder Company, Wilmington, Delaware."

[10]Courtenay De Kalb. 'The Utah Copper Mine.' M. & S. P., April 10, 1909. Page 518.

[11]Of which 1,072,921 was mined in the monzonite and 445,770 tons was heavy sulphide ore mined in the limestone and shipped direct to the smelters; most of it being copper ore and the remainder lead ore.

and quartzite, but it still retains its cover of grass and brush, amid which copper-stained croppings can be seen similar to those that incited intelligent interest in the west hill, now completely stripped of its covering of verdure and soil. The first ore broken for the mill at Copperton in 1904 came from development work and from stopes on both sides of the canyon. A caving of the surface on the east side of the canyon shows the results of some of this early stoping. At the foot of the hill, in the bottom of the canyon, was the little Rogers mill where D. C. Jackling made his mill-tests in 1899. Nothing remains of that 'historic' apparatus.

Each level, or terrace, has one line of pipe for compressed air and another for water. The steam-shovel track is laid in 5-ft. sections. Every time the shovel makes a cut, 20 to 25 ft. wide, the track and the two pipelines have to be shifted. The laborers are divided into 30 gangs, of 8 to 20 men. These gangs are not assigned to any particular level, but are shifted as required. The first gang that I saw consisted of Albanians. The steam-shovel moves on a track that is 25 ft. from the one on which are the cars to be loaded. A train of five cars receives the ore as it is dug and loaded by one steam-shovel operating a dipper having a capacity of 3½ cubic yards, equivalent to 7 tons of ore. Above the J level the larger engines are in use; each of these pulls 10 cars. The company uses 21 steam-shovels, of which 15 are Marion model 91, 2 Marion model 60, 2 Bucyrus 95-C, and 2 Atlantic 80. Watching a steam-shovel at work, I recalled similar operations that I saw in the North while the Yukon ditch was being dug in 1908.[12] The 'similarity' began and ended with the machine itself, for the ground and the result of the digging were strangely different. There the moss had to be stripped from the frozen ground before the steam-shovel, burning wood, could set to work. Labor cost $6 per day, including $2 for board, which seemed preposterously high then, but in these War days no longer warrants exclamatory remark. The difficulty of digging the Yukon ditch was not rock but ice, which is harder to conquer. All the steam-shovels at Bingham are covered by a housing consisting of 3-inch planks covered with ⅛-inch sheet-iron to protect the machinery and crew from flying stones. The crew consists of one engineer, one crane-man, and one fireman—all Americans—besides a coal-man and a pit-crew of five men, usually Greeks, whose duty it is to lay the 5-ft. lengths of rail ahead of the shovel, loosen and tighten the jack-screws when the shovel is moved, and keep the loading-track clear of broken ore.

The mixture of races is noteworthy. Seven hundred laborers are employed on the railroad and dumps. I have mentioned the Greeks, Italians, and Albanians; to these, Japanese, Koreans, and even Armenians must be added. The various races are segregated into separate gangs, supervised by a foreman of their own kind, able to speak English, which most of the rank and file cannot do. The better tasks, such as switch-tending, flagging, and bossing are given to those that can speak English, most of these being paid 25 cents and some 85c. per day additional. The regular pay of a laborer is $3.40.[13] The men working underground are mostly Austrians and Mexicans, the latter preferring to call themselves 'Spanish' since our trouble with Mexico. For a similar reason the Austrians usually call themselves 'Serbian', since most of them come not from Austria proper but from Croatia and Dalmatia. The Dalmatians and Croatians that belong to the Greek Catholic church call themselves 'Serbians', whereas the Roman Catholics from the same provinces call themselves 'Austrians'. Greeks, Italians, and Finns also are employed underground. The Austrians and Italians talk about 'going home' after the War. Most of them have left their families in Europe.

The use of coal by the steam-shovels and railroad-engines calls for remark. The coal comes from the Black Hawk and other mines of the United States Fuel Co. at Hiawatha, and also from the Kenilworth mines of the Independent Coal & Coke Co., all of which are in Carbon county, Utah, and about 130 miles by rail from Bingham. The cheapness of the coal explains the use of solid fuel instead of electricity. When the electric shovel has been more nearly perfected and other conditions are favorable, it is anticipated that the present coal-fired shovels will be supplanted. The frequent shifting of tracks and the danger from blasting are held to be unfavorable to electric connections.

After luncheon Mr. Carnahan and I mounted an ore-train and switch-backed along the line from the F level to the K level. These switchbacks have a 4% grade, compensated for curvature on the basis of 0.04 ft. per degree of curve. I noted a dozen Japanese engaged in laying track; I also saw the copper-precipitating plant in the Apex Yard gulch, and the old incline-tramway that served to take the ore from the bins at the mouth of the main haulage-level to the terminal beside the railroad. The abandoned rails and other scrap-iron seemed to invite use, either in a foundry or in a leaching-vat; indeed, I was informed later that this material is collected periodically and such of it as can be used to advantage at the foundry is shipped thither, while the inferior scrap, not saleable or suitable for the foundry, is used as a precipitant at the leaching-plant. I obtained a glimpse of the little settlement tributary to the Utah Apex mine, in which lead ore is being won from the limestone. In this part of the district the fissuring of the limestone has formed lenses of ore containing lead and copper, the latter being the first to 'peter out' in depth.

[12] 'Through the Yukon and Alaska', by T. A. Rickard. Pp. 239-248.

[13] Increased on July 1 to $3.90. The foreign gang-bosses now receive $4.75 per day.

Three years ago the Utah Apex mine was the scene of a story worthy of a moving-picture performance.

On November 21, 1913, Rafael Lopez, a Mexican, shot and killed another Mexican named Juan Valdez while they were on their way to the Highland Boy mine. The two actors in this melodrama had been neighbors in southern Arizona; later, in Mexico, the brother of Lopez had won a girl that Valdez desired; the disappointed lover had revenged himself by killing the successful suitor, burning his body. Rafael Lopez was informed of the deed and waited his chance to get even.

After various wanderings in the mining districts of the South-West Lopez came to Bingham at the time of the strike in 1912. He was one of a number of 'strike-breakers' brought thither to settle an industrial quarrel in a crude way. He met Valdez, but pretended to be ignorant of the circumstances of his brother's murder, until, at last, getting into a political argument with Valdez and another Mexican on the road to the mine, he turned on the latter, whereupon Valdez interfered, drew a knife, and was shot promptly by Lopez. Having done so, he turned back, went to his house in the upper gulch, got his rifle and cartridges, and left town.

Another story is that Lopez was in love with a Spanish girl, called Inez Ocaro (or Ocarriz) living at Bingham, and that he had bought a box of candy to present to her when, on arrival at her home, he found Valdez there. He waited for his rival to come out and then shot him. The first story is the more probable.

Before this event Lopez was known locally as a good-natured fellow, honest in his dealings, and successful in his operations as a lessee in the Apex mine, where he had made $10 to $20 per day. He had been educated in Texas, his mother being English and his father Spanish. He knew that, as he was a 'Mexican', the officers of the law would give him short shrift, so he prepared to resist capture. Moreover, a few months previously, coming to the aid of two girls that were being molested by a couple of Greeks, he had knocked them down, but his part in the affray had been misconstrued by the deputy-sheriff, Julius Sorenson, and he had been sent to jail. This had embittered him against the officers of the law.

As soon as it became known that he had killed Valdez, the officers went to his house and followed his tracks in the snow across country about 40 miles from Bingham to a ranch near Utah Lake owned by a man named Jones. The four officers were searching for him when he opened fire, killing three of them, but missing Sorenson owing to a defective cartridge. The news soon spread and caused intense excitement. Posses were organized and hurried to the scene of the tragedy, where the Sheriff, Andrew J. Smith, took command of the man-hunt. All sorts of wild rumors spread. Meanwhile the leading posse followed the outlaw's trail into the rugged mountains above the lake. Suddenly a shot rang out and the bullet struck a rock close to the pursuers. The posse sought cover and remained in hiding until evening, returning to quarters for the night. Next day, reinforced and led by the Sheriff, they surrounded the spot where Lopez had halted them, but found only a few empty pistol-cartridges. His poor aim was explained by the fact that he had used his revolver instead of his rifle. A day or two later, on November 26, Lopez returned to Bingham at night, going to the house of a friend, Mike Stefano, who lived in one of the Utah Apex company's houses, close to the No. 2 Tunnel on the Minnie claim. His feet had been frost-bitten and he was lame. He took Stefano's rifle and 40 rounds of ammunition; he also asked for some clothes, a couple of quilts, bread and meat, and made Stefano carry the provisions to the entrance of the Minnie workings. Stefano reported these facts to Tom Hoskins, the foreman of the mine. Immediately the officers of the law were notified and guards were placed at the openings of the mine and in the various workings. The bandit was trapped; but he had miles of workings in which to hide. He was familiar with these workings, for he had been 'leasing' on the 400-ft. level, near the shaft, for several months. He could find his way in the dark. To search for an outlaw at bay in the darkness of a mine was foolish, so it was suggested that he be smoked out, although Mr. V. S. Rood, the superintendent, expressed his opinion that the extent of the workings—20 to 25 miles in aggregate length—would nullify the attempt. A dash was made by 15 men in a steel-protected ore-car, but they found no trace. Lopez 'held up' several miners underground, without hurting them, only taking their candles. The deputies were quite unable to track him to his lair. Intense curiosity prevailed. The whole of Utah was aroused. Everybody had his own idea how to catch the desperado, without undertaking to perform the feat. He was known to be a remarkably good hip-shot, that is, he could shoot accurately without sighting.

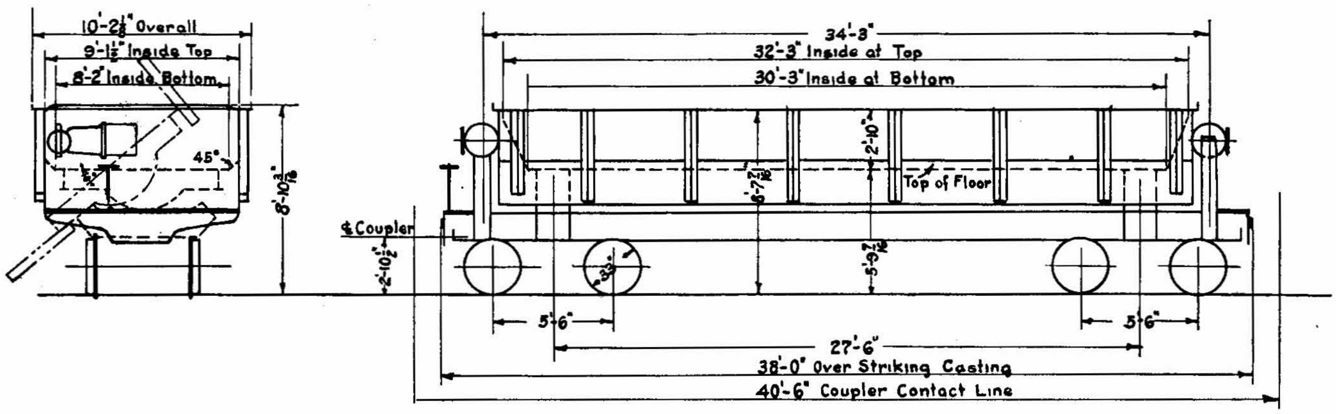
The Sheriff ordered the superintendent of the mine to cease work on the upper levels, but the miners had been scared and operations had been half-hearted since the affair began. At last, unable to track the desperado, it was decided to try smoke. The guards were doubled, some of the outlets of the mine were sealed, others were watched with increased vigilance, and powerful search-lights were placed so as to illumine the surface. Fresh fires were about to be started. Four men climbed with a bale of hay up an incline, called the Andy; two were pulling and two were pushing the bale over a pile of waste at the bottom of a big stope when suddenly three shots broke the silence of the mine. One man was killed at once and his body rolled down the incline; another fell and crawled aside, groaning horribly. The remainder of the posse tried to recover the bodies of their comrades, whereupon Lopez came down the slope, so as to see better, and opened fire again, without causing further casualties. The deputies escaped to daylight. This was on November 30. The next day the Sheriff conferred with Mr. Rood, the superintendent, and Tom Hoskins, the foreman; the result being a decision to bulk-head the levels and fumigate the whole of the upper workings until every part was saturated with gas. They could ascertain when the oxygen in the air of the mine had been consumed by the fact that the fires would refuse to burn further. Among the combustibles used, or suggested, were formaldehyde, potassium permanganate, lump sulphur, green wood, damp gunpowder, oil-soaked cotton-waste, and cayenne pepper. The fires were kept alight for five days. It cost the Utah Apex company several thousand dollars and threw 200 miners out of work for 10 days.

By this time public excitement had reached a climax. The Governor offered a reward of $1000 for the capture of Lopez. The newspapers made the most of the man-hunt. During intervals the daily press gave full space to describing the funerals of the victims of the chase. Betting was lively. When the bulkheads were removed, on December 5, and search for the supposed dead body of Lopez was begun the next day, it became evident that the local betting was lending a gruesome zest to the examination of the mine. Many felt sure that he had escaped; he was reported as seen at several localities; one person acknowledged to having received a letter from him post-marked Salt Lake City. On December 10 the girl Inez Ocarriz stated that she had talked with Lopez in the Andy tunnel; three days later a shift-boss claimed to have been accosted by him underground and he had sworn to die fighting. The officers did not abate their watchfulness, relying upon starvation to finish the pursuit. Stefano's blankets and other traces of Lopez were found in the fifth stope of the Andy incline. Another stope, in which he was imagined to be, was dynamited by the Sheriff, after a melodramatic call to the desperado to surrender. Nothing came of it. Those who saw the dense clouds of smoke that issued from the mine-openings when the bulkheads were removed decided that no human could have survived the effect of them. Days passed; the mine was ransacked from end to end, public interest waned, several guards resigned, and the Sheriff had to bring the fiasco to an impotent conclusion. I am told that there was dispute as to how the desperado managed to get food while in the mine, the most likely suggestion being that "he lived on the milk he got from the Sheriff's goat." Lopez was never seen again. His fate remains unknown. It is probable that he escaped during the confusion and delay following the shooting underground, for 24 hours elapsed before the workings were sealed.[14] Perhaps he became a *revoltoso* in Mexico and under a new name now plays the part of a constitutional reformer south of the Rio Grande.[15]

Returning to the Utah Copper, the block-signal system is not in use on the mine-railway. No less than 120 men are employed as switch-tenders. This will afford suitable employment for crippled soldiers returning from the War.

From the K level I obtained a new view of the surficial workings. We walked along this terrace, which curves south-eastward, to the far end, where I noted the disposal of waste removed in the course of 'stripping'. Here, in Galena gulch, 8,000,000 cubic yards of rock has been stored, leaving room for 15,000,000 more. The waste discarded in the past has been about equal to the ore won, but in the future the proportion of waste to ore will decrease gradually. Last year 4,271,868 cu. yd., or 8,827,400 tons, of cap was set aside while 6,038,900 cu. yd., or 12,542,000 tons, of ore was sent to the mills. The cap-rock of carbonate ore is kept separate by removal over a railroad that crosses Bingham gulch and passes along the east hill to Tiewaukee, McGuire, and Little Eddy gulches, where 20,000,000 tons of it, containing 0.65% copper, is stored until it can be treated at the leaching-plant in the Salt Lake valley. Below the dump of brown monzonite is another consisting of quartzite, from the edges of the orebody. This contains only 0.3% copper.

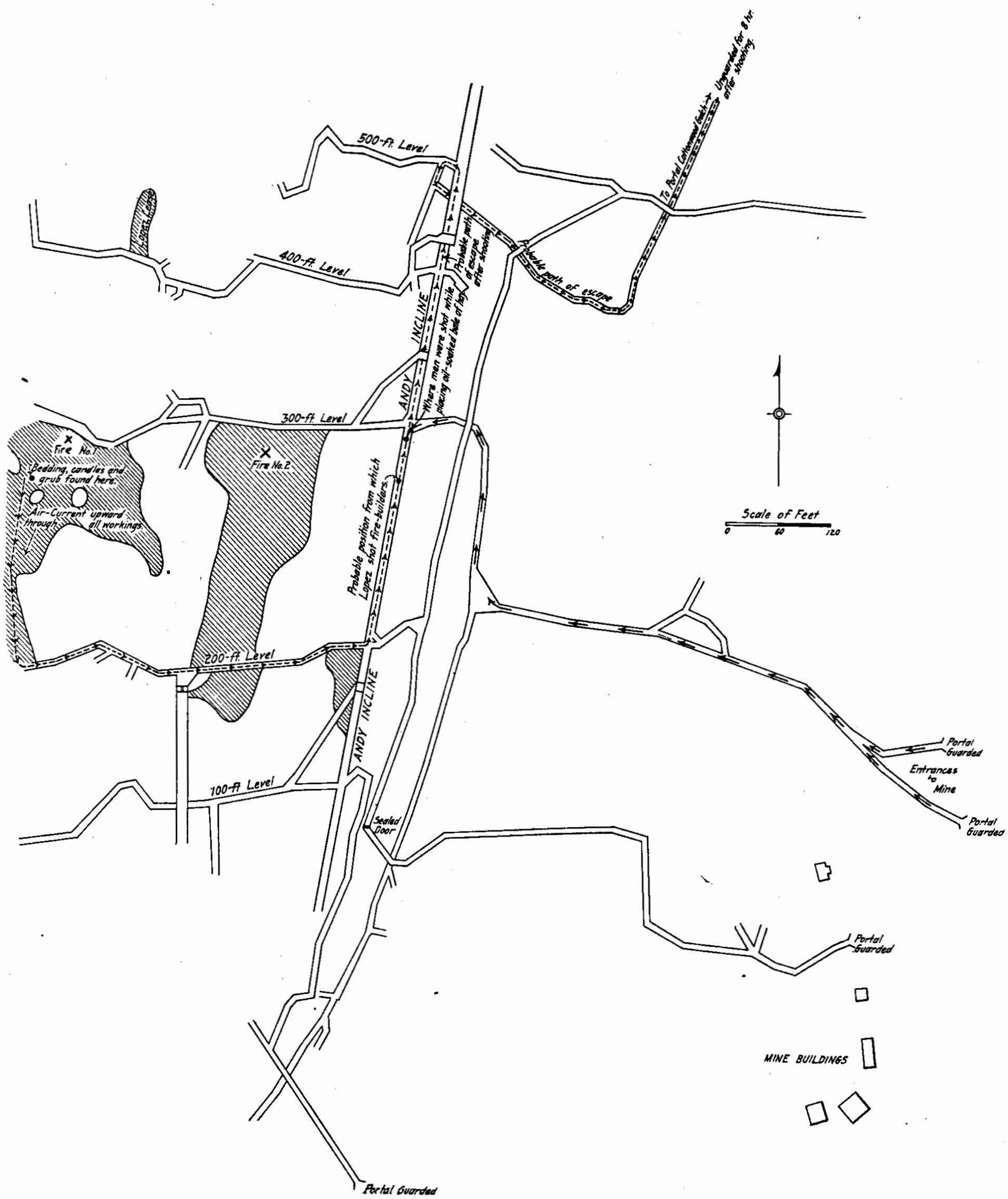
Turning round, so as to face south, toward the dumps of 'stripping', I noticed groups of men on each of the L



THE CLARK 30-YARD STEEL DUMP-CAR

[14]Mr. Rood, in a recent letter to me, says: "Lopez had access to the seven levels on the incline, as well as all the workings in the old York, Petro, and Phoenix mines. If he had desired, he could have held-up the hoisting-engineer on the 700-ft. level and taken a chance to reach the lower levels by climbing down the main shaft."

[15]For my information I am indebted to the files of the 'Deseret News' and to a pamphlet, by one of the hoist-engineers at the Utah Apex mine, entitled 'Utah's Greatest Man-Hunt; by an Eye-Witness'.



THE MINE-WORKINGS IN WHICH LOPEZ SECRETED HIMSELF. THE ARROWS ON THE LEFT SHOW HIS PROBABLE TRAIL; THOSE ON THE RIGHT INDICATE THE MOVEMENTS OF THE POSSE and M levels. These were gangs of 12 to 20 laborers, engaged in dumping cars, moving track, and shoveling rock. Three kinds of cars are in use: the Clark automatic car, of 30 cu. yd. capacity; the Oliver, of 12 cu. yd. capacity, dumped by hand; and the crab-cars, of 6 and 4 cu. yd. capacity, these last being confined to the three uppermost levels. The inferior cars are being replaced gradually by the superior, the poorer type going up-hill. Crab-cars are used only on the two uppermost levels. As an example of the foresight of the Utah Copper management I mention the fact that the south end of the property was tested with churn-drills before it was covered with waste, thus avoiding the blunder, made in other districts, where 'stripping' has been spread on ground that subsequently proved ore-bearing and suitable for steam-shoveling. The surface of Galena gulch, which is at the upper end of the main Bingham canyon, covers the workings of many notable mines, such as the Jordan, Telegraph, Spanish, and Galena, which have yielded many million dollars in lead and silver. Looking across, over the wastage from the big mine at my back, I noted the excavations in the underlying limestone. It was a quiet spot, detached from the intense activity of the mining operations; the tender green of the spring foliage nearly covered the old scars made by the miner; from the farther hillside came the sound of cow-bells, and from the long grass a meadow-lark carolled rapturously.

Taking the train for the return journey, I asked a fellow-passenger about the Japanese. He said they were "right up to the mark on Red Cross and Liberty Bonds." At Magna the mill-hands boarded the train and I found myself in the crowd with which I had come out in the morning. While the train halted, I noticed a stalwart young fellow, a brakeman, shaking hands with several friends in farewell. He had been drafted and was off to the War. It seemed a far cry from that wayside station overlooking the Great Salt Lake to 'somewhere in France,' but the copper that I had seen being mined at Bingham had the same destination, and was to be used for the same purpose, the greatest purpose in the world just now: to stop the march of the goose-step over creation, to frustrate the designs of the Prussian outlaw, and re-establish good faith among the nations.

MINING METHODS

The control of Colonel Wall's property passed to Messrs. Penrose, MacNeill, and Jackling, as already related, in the spring of 1903, and the construction of an experimental concentrating plant—the Copperton mill—was begun in August. The extraction of ore underground began in November. These first mining operations by the Utah Copper company had for their primary object the supplying of 300 tons of ore daily to the Copperton mill. Most of this output was obtained from development work and sub-level drifts. At first no particular system was adopted, the ore being simply stoped from the sub-levels in chambers extending up to the 'cap,' which was supported on stulls. When broken the ore was shoveled into cars and trammed to chutes, through which it was delivered to cars on the haulage-levels and trammed to the bins outside. As the capacity of the Copperton mill was increased and an additional tonnage had to be provided, a plan was devised of digging a number of small raises under each pile of broken ore, which was then withdrawn from below and thus moved without shoveling. The sub-levels were driven 25 ft. apart vertically, and later the ore was withdrawn through a system of small raises with numerous branches, which met those of the adjoining raises. The ground between these 'finger' raises was blasted, causing the ore to cave. The caved ore was withdrawn through chutes at the bottom of each raise until the cap-rock began to appear, indicating that all the overlying ore had been tapped. This method has been elaborated at the Inspiration mine, at Miami, Arizona, as described by me in the Mining and Scientific Press of September 29, 1917. The method is well defined by George R. Lehman,* who says that it consists "essentially, of undercutting the ore (taking out a horizontal slice), allowing the ore above to cave and crush, and drawing off the crushed ore through small inclined raises, driven under the caved ore, into main inclined raises that lead down to the haulage-drift chutes." The aim, of course, is the excavation of ore with the aid of the minimum amount of timbers and explosives, utilizing and controlling the descent of ground that has been induced to cave. The underlying idea is to persuade Nature to do as much of the work of fracturing and moving as is consistent with the safety of the mine and of the men underground.

*'Ore-Drawing Tests.' By George R. Lehman. Trans. A. I. M. E., Vol. LV, p. 225.

At the time this method of mining was used at the Utah Copper it was realized that it would entail some admixture of 'cap' with ore, but this was considered to be relatively unimportant, as it was known that the entire orebody was to be stripped subsequently in preparation for digging with steam-shovels, so that the mixing of 'cap' and ore would not cause any serious loss because these two products could be excavated separately by the steam-shovels.

Such was the first mining. All the ore produced up to June 1907 was obtained in this way. Mr. Jackling was general manager; in January 1906 he appointed Mr. Gemmell as general superintendent. He returned from Mexico to accept the appointment, for which he was peculiarly fitted not only by his wide experience in the work of mining but on account of the six years he had spent as engineer in the service of the Santa Fé railroad. In April 1906 he accompanied Mr. Jackling on a visit to the Mesabi range for the purpose of studying methods of mining, more particularly the use of steam-shovels. Among his class-mates at the University of Michigan was William J. Olcott, distinguished among the engineers of the iron country, and to him Mr. Gemmell turned when the time came to select a man to take charge of the steam-shovel work. Mr. Olcott recommended J. D. Shilling, who was engaged in July 1906. Mr. Shilling is now the superintendent of the mine; Mr. Gemmell is general manager for the company, and Mr. Jackling, who is directing the operations of several other mines in Arizona, Montana, Nevada, and New Mexico, is managing director and first vice-president. Mr. Charles M. MacNeill is the president of the company.

The first steam-shovel went to work in August 1906. The chief duty of the steam-shovels was to strip the cap, or overburden, in preparation for the application of an open-cut or terrace system of mining to the ore underneath. At the end of 1909 the ground stripped amounted to 3,232,000 cubic yards. As the cap was removed, it became possible to quarry the ore. The first production by this method was made in March 1907.

Given unlimited money and time, it might have been possible to start the steam-shovels at the top of the mountain and work downward by successive terraces, but mining engineering is allied to business, that is, to the making of money, and it is doubtful whether it would have been economical to do the apparently obvious thing.

In the first place, such a method of attack would have necessitated a long delay while the requisite amount of stripping was accomplished and large sums of money would have had to be borrowed, by the sale of securities, in order to finance the operations pending the attain-

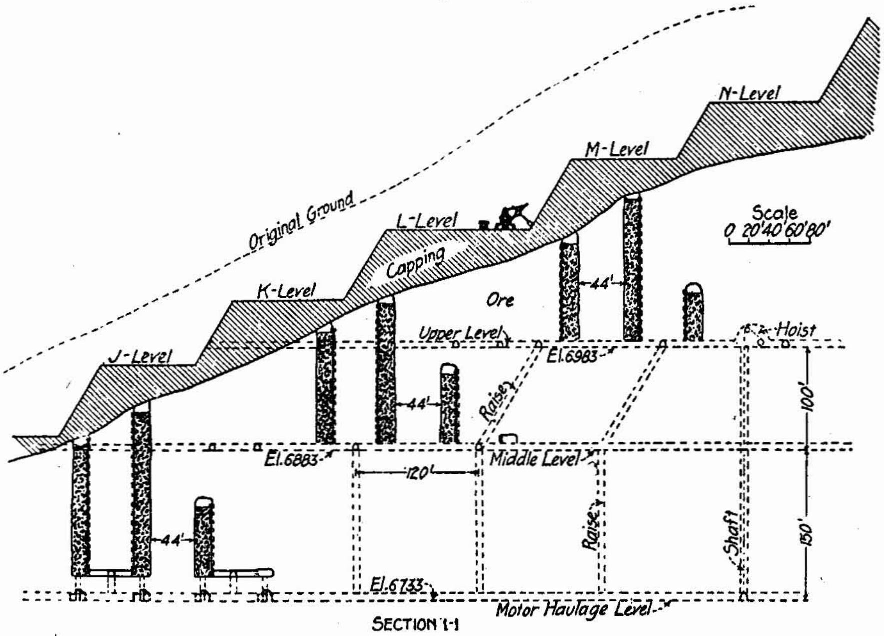


FIG. 1. SECTION SHOWING RELATIVE POSITION OF STOPES TO STEAM-SHOVEL TERRACES

ment of profitable production. By starting underground mining, the company was able not only to extract ore forthwith for the Copperton mill but also to extend exploratory work in such a way as to ascertain the limits of the orebody and to develop the ground in preparation

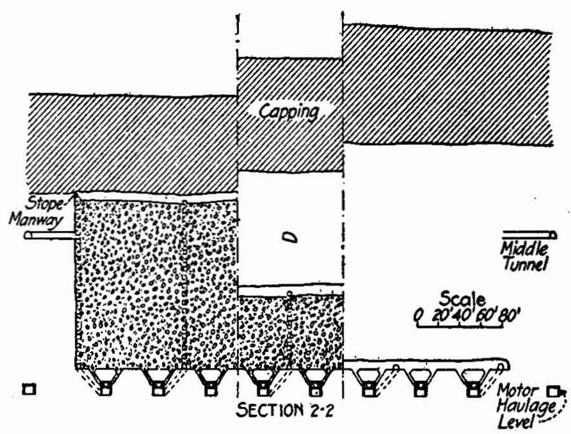


FIG. 2. SECTION SHOWING THE THREE STAGES OF STOPING ABOVE THE HAULAGE-LEVEL

for the enormous exploitation that the manager had in view. Moreover, the Utah Copper Company did not own the top of the mountain until, in February 1910, it acquired the property of the Boston Consolidated Mining Co., thereby increasing its area of lode-mining claims to 554 acres.* The company now owns 830 acres of mining ground. At that time the Arthur mill of the Boston Consolidated also passed into the possession of the Utah Copper, which had previously finished its Magna mill in 1907; so that an enormous tonnage of ore (13,000 tons daily) was demanded. Owing to exigencies of milling capacity it became necessary to continue the shrinkage-caving system of the Boston Consolidated ground, because there was no ore stripped and ready for steam-shoveling in that part of the mountain and steam-shovel operations on the original Utah Copper ground were not sufficiently far advanced to permit of supplying both of the big mills except by aid of underground stoping. Therefore the use of underground mining methods, which had ceased in the original Utah Copper ground on September 17, 1912, was continued in the Boston Consolidated ground until March 31, 1914, by which time the stripping had been so far advanced, and the terraces so well started, as to enable the steam-shovels to supply both mills.

An excellent description of the shrinkage-stoping system as used in the Utah Copper mine is given by T. S. Carnahan in Volume LIV (1916) of the Transactions of the American Institute of Mining Engineers. The system of mining that was adopted was devised so as not to interfere with the steam-shovel operations, which were foreseen from the start. It was essential, therefore,

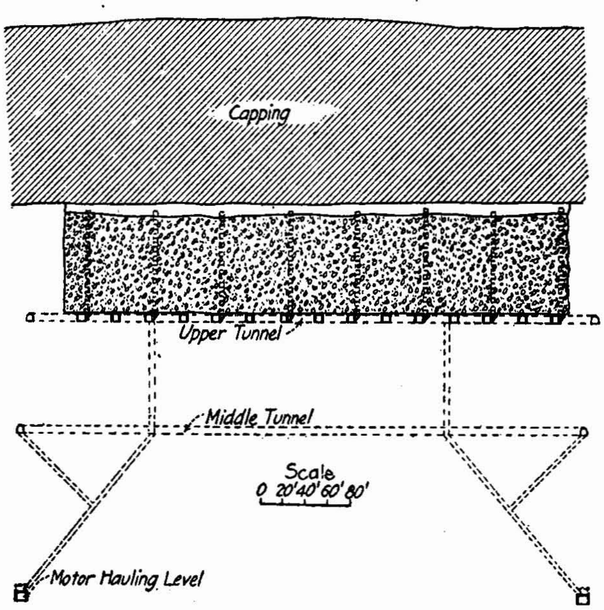


FIG. 3. SECTION SHOWING HOW A STOPE IS STARTED ABOVE THE UPPER LEVEL AND THE RAISES THROUGH WHICH THE ORE IS DROPPED TO THE HAULAGE-LEVEL.

*On December 31, 1918, the area of lode-mining claims was 665.5 acres.

"that the surface should not be caved, that no large openings be left unfilled, and that the capping should not be mixed with the ore." To accomplish this, the stopes were started from three separate levels or tunnels. The lowest of these served for motor-haulage; the second, 150 ft. higher, was equipped for hand-tramming, as was also the third entry, 100 ft. higher than the second. These levels were connected by manways and by raises for dropping the ore from the upper workings to the motor-haulage level. An underground shaft connected the workings and was employed for hoisting supplies, using a compressed-air engine. The connections with the surface were more than sufficient to ensure ventilation. Fig. 1, 2, and 3 are borrowed from Mr. Carnahan's article. He states that it was the custom to divide the orebody in blocks 500 ft. wide and to finish stoping on the top level considerably in advance of the stoping started from the middle level. "In this way the upper stopes were abandoned before the stopes from below began to disturb the tramming-drifts and stope-manways." Likewise the stopes above the middle level were kept well ahead of those started from the main level. During three years, 1911 to 1913 inclusive, 2,824,439 dry tons of ore was drawn from the mine and loaded into railroad-cars at a cost of 56.6 cents per ton. In addition 102,719 ft. of development yielded 247,280 dry tons at a cost of $4.95 per foot of development, or $2.05 per ton of ore.

Before making systematic preparation for the steam-shovel method of mining, it was deemed advisable to obtain expert advice from the iron regions. In the opinion of the experts the canyon was too narrow and the working-area too limited for the best results from steam-shoveling, but the chief obstacle to this method of removing the ore appeared to be the manner in which the ore-bearing rock would be broken in the course of blasting, namely, in lumps so large that they would have to be reduced mechanically before delivery by the steam-shovel to the cars and bins. Therefore they advised the driving of mill-holes to the surface at intervals of 50 or 75 ft., breaking both the cap and the ore into these raises, for discharging into cars. Mr. Jackling, however, demurred to this opinion and after discussing the matter with Seeley W. Mudd and A. C. Beatty, who were then examining the mine for the Guggenheims, preparatory to their underwriting the bond issue, he went to the Mesabi iron region himself, with Mr. Gemmell, as I have previously stated, in order to get the fullest possible light on the subject. The result of this journey of observation was to confirm Messrs. Jackling and Gemmell in their decision to introduce the use of the steam-shovel as rapidly and completely as circumstances would permit. In the report by Mr. Mudd, dated October 14, 1905, he states: "There seems little reason to doubt the feasibility and desirability of mining with the steam-shovel. This would, of course, necessitate the stripping off and removal of the waste and low-grade surface ore, and this would involve a considerable initial expense, but in the end, this expense would be justified."

Mr. Mudd gives 94.43 cents per ton as the cost of mining by the Utah Copper company during the twelve months ending June 30, 1905, and mentions that "the Utah Construction Co., a large firm of railroad contractors, has made a provisional offer to the Utah Copper Co. of 60c. per yard, or 30c. per ton, the contractors agreeing to furnish all equipment and deliver the material not to exceed 2000 ft. from the point at which it is mined. The San Pedro Railroad construction work was done on a basis of 65c. per yard for rock, and most of the material was hauled in carts. Such a price is not abnormal. In the Michigan iron mines, the cost of mining with steam-shovels has been reduced to 12c. per ton or less. . . . Taking all these circumstances into consideration . . . a cost of less than 40c. per yard is probable, but for the purposes of this estimate, we will assume a cost of 50c. per yard, an average thickness of stripping of 80 ft., and an average thickness of pay-ore beneath to be mined of only 160 ft. On this basis it will be necessary to mine three tons of material to secure two tons of payable ore. Assuming a yard to equal about two tons, the cost of mining would be 37½c. per ton, including the cost of stripping. I believe this figure will be found safe, notwithstanding several unfavorable features which should be considered, but the cost of 40c. per ton will be used in the estimates."

The "unfavorable features" include those recognized by some of the steam-shovel experts, namely, the railroad, county road, and dwellings in the gulch, also the steepness of the hillside, causing the stripping to slide when under-cut, and the probable need of a tunnel in order to reach the deeper ore.

In Mr. Jackling's original report, of 1899, there is more about the metallurgy than the mining of the ore, because, among other reasons, the writer of that report was primarily a metallurgist, but it is worth while to quote his reference to the method of mining to be adopted. He said: "For working the property, it is proposed to build a concentrating plant of 2000 tons daily capacity, and a smelter and refinery of 125 tons daily capacity, near the point of the mountain between Salt Lake City and Garfield Beach, where water is plentiful and railroad facilities good. This point would be connected with the mine by 15 miles of standard gauge railroad. The ore would be worked by quarrying or open-pit methods, the overlying deposits of wash and low-grade material being first stripped off the ore and hauled by railroad to the most convenient dumping-ground. Both the ore and waste would be loaded on cars by steam-shovels. Drilling at the mine would be done with air-drills, for which air would be supplied by a small electric compressor plant, centrally located on the ground." He estimated three tons of 'wash' and eight tons of ore per square foot of surface; the cost of stripping to be 75c.

per cubic yard, or 37½c. per ton, so that the cost per ton of ore would be 14 cents. He estimated the cost of mining the ore at 40c. and of hauling it to the mill at Garfield, 15c., making the total cost of stripping, mining, and delivery to the mill 69c. per ton.

Colonel Wall, writing in 1904, in a pamphlet called 'The Mountain Empire, Utah', anticipated that the cost of mining, using the mill-hole method, would be 15c. per ton and that of milling 20c. per ton of ore. He expected the cost of railroad transport, from mine to mill, to be reduced to 5c. per ton. Taking the average copper content of the ore at 2.2%, with 75c. in gold and silver, he anticipated a recovery of 80% and he expected the cost of smelting to be "less than the value of the gold and silver contents."

In 1917, the charge made for stripping against the ore mined during that year was at the rate of 7.5c. per ton of ore; for development, 1.04c. per ton was allowed; while the actual cost of mining was 18.04c. per ton, to which 0.94c. was added for depreciation, 15.99c. for taxes, 1.91c. for insurance, general office expense, legal expense, and salaries. Thus, including stripping, development, and the other necessary charges, the average cost of mining was 45.42 cents per ton of ore. The freightage from the mine to the mills at Garfield represented a cost of 25c. per ton.

THE MILLS

INTRODUCTION. As soon as the mine passed from the ownership of Colonel Wall to the control of the Utah Copper Company in 1903, the company started to build a concentrator having a capacity of 300 tons at Copperton, two miles below Bingham and 2½ miles below the mine itself. The original plant was equipped with one No. 6 and one No. 4 Gates gyratory crushers, one coarse trommel-screen, two pairs of 26 by 15-in. and two pairs of 36 by 15-in. rolls, four fine trommel-screens, four Trent, one Janney, and one Waddell 6-ft. Chilean mills, six 3-spigot hydraulic classifiers, six 3-compartment Harz jigs, 34 Frue and 18 Johnston vanners, 48 Wilfley, two Card, and two Overstrom tables. The flow-sheet of the above equipment, as originally installed, is shown in Fig. 1.

The flow-sheet in this mill was changed from time to time (see Fig. 2), and it was on the information gained at this plant that the design of the later mill was based.

The Copperton mill started to work in August 1904. It made a 25% concentrate on a 2% ore, the recovery being slightly under 70%. The capacity was increased gradually to 1000 tons per day; meanwhile designs were prepared for a larger mill, to be erected at Garfield, near the Lake, four miles east of the smelter belonging to the American Smelting & Refining Co. In 1905 the foundations for a plant of 3000 tons capacity were laid at Garfield, but legal difficulties with Colonel Wall delayed the building of this new mill, and in the interval the development of the mine progressed so favorably that it was decided to make this plant, called the Magna, one of 6000 tons capacity. The first section—500 tons—started to work in June 1907, and the entire mill was in operation by November 1908. By 1908 the mine had been so opened up as to furnish ample ore, but the lack of railroad facilities prevented a regular and adequate supply of ore, so that the mill treated only 77% of its rated capacity and only 55% of what it showed itself capable of treating when fully equipped. At this period the company had constructed and started to operate a power plant at Magna sufficient for the entire enterprise. This proved a great economy and advantage in many ways. The springs at Pleasant Green, near Magna, which the company had purchased from Colonel Wall and others, produced ample water for the tonnage then being treated.

These springs yield, with continuous pumping, 12,500 gal. per minute. The No. 1 pump-station is at the springs, which, to be precise, are half a mile north of Pleasant Green and close to the old Magna steam-power plant. The water is pumped to the Magna reservoir against a head of 225 ft., and to the Arthur reservoir against a head of 325 ft. During recent years the bulk of the water used in the two mills has come from the Jordan river through the Utah & Salt Lake canal, the point of diversion from the river being 5.7 miles north of Utah lake. The No. 2 pumping-station, which is at the lower end of the canal, supplies the Magna reservoir against a head of 30 ft., and from this reservoir the water is pumped to that of the Arthur plant against a head of 100 ft. The Utah Copper Company secured a right-of-way through the canal from its owners, together with the right to enlarge the canal; this work has been in hand for several years and was completed during the current year. The amount of water received through the canal varies with the seasons; any shortage is rectified by pumping from the springs. As nearly as can be determined, none of the tailing-water rejoins the source of the springs, owing probably to the fact that the fine silt in the tailing has sealed the ground. The water from the tailing-dump flows into a large settling-pond and is returned as required to the pump-sump through artificial channels. On average the milling plants use 1440 gal. per ton of ore treated. The average cost of pumping is 8.09 mills per 1000 gallons. In 1910 the company acquired the property of its neighbor, the Boston Consolidated Co.; this included not only the ore-bearing ground adjoining its own mine at Bingham, but also the Arthur mill, about a mile west from the Magna, at Garfield. The Arthur mill, of 3000 tons capacity, was designed by A. J. Bettles in 1905, and was noteworthy for including 312 Nissen stamps in its original equipment. I like to mention this fact because Peter N. Nissen has proved himself much more than a good mill-engineer. A Canadian by birth, he enlisted in the British army in October 1914 and has distinguished himself so greatly by efficiency and gallantry that he is now a D. S. O. and a Lieutenant-Colonel. The Copperton mill was closed-down on August 1, 1910, and most of its equipment of machinery was removed to the Arthur plant. In the following August the Bingham & Garfield railroad, connecting the mine to the mills, was completed by a company controlled by the Utah Copper Co. and by the end of 1912 the two big mills were in a position to treat 21,000 tons of ore per diem.

The Arthur and Magna mills were originally intended for the treatment of 3000 and 6000 tons respectively. They are now treating 16,000 and 20,000 tons. During the third quarter of 1917 they treated a total of 3,439,400 tons, or an average of 37,385 tons per diem.

These two mills stand at the foot of the Oquirrh range, facing the plain from which the Lake has retired. In the mist of morning, as I first viewed the scene, it was easy to imagine that the waters still overspread the plain, which looked like an angry sea, the streaks of saline sand fringing the stretches of marsh like the white crests of green waves.

As the two mills are much alike, except in so far as flotation forms a feature of the Arthur plant, I shall describe only the latter.

Crushing Plant. The ore arrives from the mine in trains of 40 cars. These are of the gondola type, made of steel and emptying at the bottom; on average they hold 68.15 tons wet or 63.82 tons dry. The moisture therefore is 6.35%. The ore is discharged into a primary storage-bin of 10,000 tons capacity, on the top of which grizzley-bars are placed horizontally 12 inches apart. Any pieces too large to pass are broken with sledge-hammers. Below these grizzlies are others set at an angle of 40°, the bars being made of specially rolled steel 2⅜ in. apart. The oversize passes into pockets underneath, and is drawn by steel apron-feeders 48 in. wide onto a steel-pan conveyor 48 in. wide; this carries it to two No. 8 McCully gyratory crushers, each of which serves a unit, but is supplemented by a spare crusher of the same kind, available for either unit whenever required.

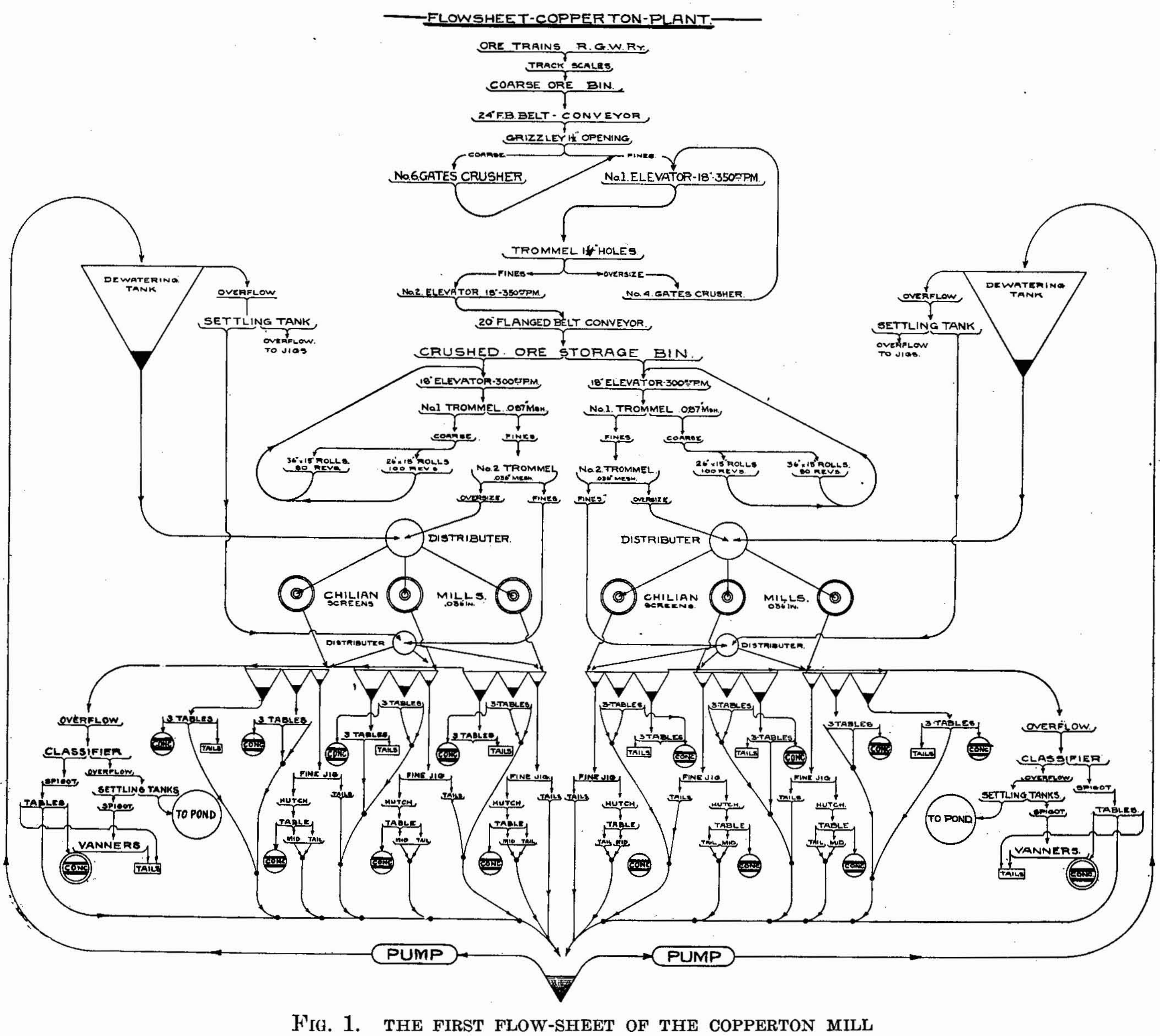


Fig. 1. THE FIRST FLOW-SHEET OF THE COPPERTON MILL

My description will now be confined to one of the two units into which the crushing plant is divided. The undersize from the last-mentioned grizzley drops into pockets, and is drawn from them by steel-apron feeders, 54 in. wide, onto a rubber-belt conveyor, 42 in. wide, that delivers to the first sizing-screen set at 42°, the apertures of which are varied from one to three inches by changing the screen, according to the moisture in the ore. The undersize from the screen is carried by a rubber-belt conveyor (No. 1), 36 in. wide, to the storage-bins. The oversize from the screen joins the product from the crusher, and is conveyed to a belt-bucket elevator (A) that lifts and discharges upon a second screen having apertures that are regulated in the same way as those previously mentioned, that is, in accordance with the moisture. The undersize is conveyed to the storage-bins by rubber-belt conveyor No. 1, while the oversize passes to 72 by 20-in. rolls operated at 100 r.p.m. This pair of rolls is equipped with fly-wheel pulleys, the larger weighing 18 tons and the smaller 5 tons. The product discharges directly to a third wire-mesh screen set at 40°, the apertures ranging from 1 to 1½ in., according to the moisture, which varies from 5 to 13%. The undersize is carried to the conveyor that serves the preceding screens, while the oversize goes to an elevator (B), which takes it to a fourth set of similar screens. The undersize from these goes by a conveyor (No. 1) to the storage-bins, while the oversize is reduced in a second pair of 72 by 20-in. rolls operated at the same speed as the pair previously mentioned. The product of these rolls falls upon a fifth wire-screen set like the others at 40°, with apertures varying from 1 to 1½ in. The undersize goes to the storage-bins, while the oversize passes to the elevator B which puts this material in a closed circuit with the second pair of rolls.

Already work has been started to re-model this plant in order to overcome the difficulties due to snow and ice. For three months in the winter a loss of capacity equal to 25% is caused by frost in two ways:

(1) The ore freezes in the railroad-cars, making it difficult to unload. This trouble will be overcome by the use of a car-tipple, made by the Wellman-Seaver-Morgan company, and able to overturn a car at an angle of 70°.

(2) The ore freezes in the primary storage-bins. This will be remedied by conveying the ore directly from the primary crushing plant to the secondary crushing plant without any interval of rest, so that the ore will be kept constantly in motion.

(3) The ore at the mine appears to break more coarsely in winter than in summer, causing delays in unloading. There follows an excessive breaking of ore by hand in order to admit it through the first grizzley, the one with bars 12 in. apart. This is to be overcome by the use of a No. 27 Allis-Chalmers Gates gyratory crusher, so that the new plant will include the tipple, the new gyratory, and two 60-in. rubber-belt conveyors.

(4) In addition, two pairs of 72 by 20-in. rolls will be placed in the secondary crushing plant, the idea being

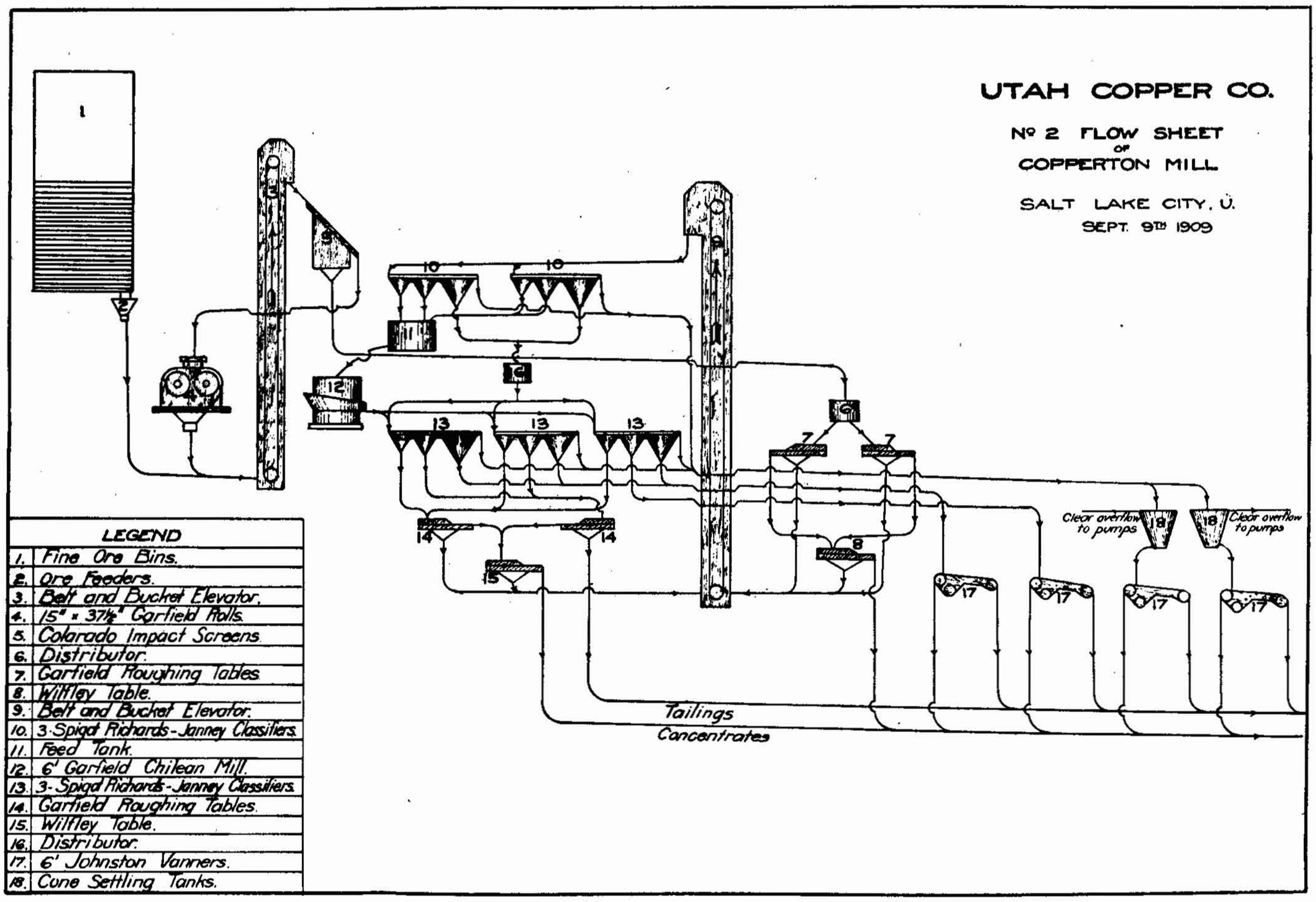


FIG. 2. THE SECOND FLOW-SHEET OF THE COPPERTON MILL

to avoid the use of elevators to obtain a fine reduction, namely, a ½-in. screen-aperture during the whole year, as against a 1-in. aperture in use in the summer and a 1½-in. during the winter. (See Fig. 5.)

The secondary bins receiving the product of the crushing plant have a storage capacity of 12,900 tons. Standing at the top of them the visitor may think for a moment that he has a general view of the mill, but he discovers later that he can only see a fraction of the plant, which extends for 1750 ft., as measured from the railroad-track on which the crude ore arrives to the railroad-track on which the finished product is removed to the smelter. However, one does see the floor below, on which are the classifiers. By leaning over the rail of the ore-bin floor, one can see the Chilean mills at work, and in front of them, under the classifier-floor, are the tanks from which the Chilean mills are fed. The rest of the picture consists of suggestions of further mechanisms with which closer acquaintance will be made forthwith.

Before resuming the description of the mill-movement, or flow-sheet, it is necessary to mention the automatic sampler, which is placed at the discharge of the conveyors feeding the bins. This sampler is based on the principle of taking part of the feed part of the time. A fraction of the flow, equivalent to 1%, is diverted every five minutes. After sampling, the ore is distributed by horizontal tripper-conveyors into the various pockets feeding the sections.

The following is a screen-analysis of the product as it leaves the crushing plant:

	Percentage weight material	Accum. percentage weight material
Opening 1.050 in.	1.17	1.17
" 0.742 in.	8.44	9.61
" 0.525 in.	22.27	31.88
" 0.371 in.	16.30	48.18
3-mesh	12.11	60.29
4 "	7.24	67.53
6 "	4.07	71.60
8 "	4.27	75.87
10 "	3.61	79.48
16 "	2.82	82.30
20 "	2.31	84.61
28 "	1.92	86.53
35 "	1.74	88.27
48 "	1.55	89.82
65 "	1.53	91.35
100 "	1.27	92.62
150 "	0.82	93.44
200 "	1.07	94.51
Pass 200 "	5.49	100.00

From the bins the ore is drawn by steel-apron feeders adjustable to three rates of speed by step-pulleys, to which is attached a mechanical counter, or tachometer, registering the number of revolutions of the head pulley of the feeder, so that it is easy to adjust the proportion of tonnage passing to the 13 sections into which the mill is divided.

We now confine our attention to one of these 13 sections.

CONCENTRATORS. From the two bin-feeders of the section the ore falls upon two Colorado Iron Works shaking-impact screens, 3 by 4 ft., set at an angle of 34°. The undersize, to which water is added, flows to 12 Garfield roughing-tables, these being modified Wilfleys with riffles extending the full length of the deck. The oversize from the impact-screens goes to two pairs of 15 by 37½-in. Garfield rolls of the movable pedestal type, the product from these passing to a bucket-elevator that lifts onto four impact-screens 3 by 4 ft. each, set at an angle of 34°. The undersize, to which water is added, from these screens joins the undersize from the previous screens, while the oversize is returned to the rolls, which puts the material in a closed circuit with the four impact-screens and the two pairs of rolls. The screen-mesh is varied between 6 and 10, depending upon the character of the ore and its moisture.

Various kinds of jigs were tried in the Copperton mill, including those of the Harz type, which was adopted, with trommels, when planning the original flow-sheet of the Magna mill. Later, the Garfield roughing-table was developed, and it proved to be more efficient than the jig, owing to the fineness of the mineral liberated by the crushing and the low ratio of concentration. At the time the Copperton mill was designed, the trommel-screen was in general use in the West and the impact-screen had not as yet been developed satisfactorily; when, at a later date, it was tried in the Magna mill, it proved to be more efficient and economical than the trommel. For these reasons the Garfield roughing-table and the impact-screen were substituted for the jig and trommel as soon as the Copperton mill was overhauled and remodeled; but before these changes had been made in the Copperton mill the Magna plant had been equipped entirely with roughing-tables and impact-screens, and the same practice was adopted subsequently in the Arthur mill.

In one of the sections, I noted a Tyler vibrating screen. This screen is caused to vibrate by means of two hammers that strike pins in the centre of the screen. It is used as an experiment.

Now all the pulp is wet. Reverting to the roughing-tables, these produce a low-grade concentrate containing all the coarse mineral as liberated by the precedent crushing. This concentrate is re-concentrated on four No. 5 Wilfley tables placed on the floor above the roughing-tables, the conveying being done by a bucket-elevator. The Wilfley tables yield three products: concentrate (the first finished product in the mill), middling, and tailing. The roughing-tables yield only middling and tailing. The middling from the finishing tables is returned to the elevator, which puts it in a closed circuit with the Wilfley tables. On a tonnage of 1230 tons per section, the concentrate from the roughing-tables is 208 tons, which is the feed to the Wilfley tables. The latter produce 26 tons of concentrate containing 22.8% copper, 26.8% iron, and 12.6% insoluble. The 20 tons of middling from the Wilfley tables is returned to the elevator, which puts it in a closed circuit with the Wilfley tables.

Looking at the Wilfley tables, I noticed that the pyrite and chalcopyrite hide the finer chalcocite, so that the heading is distinctly pyritic. It does not look like the product from a chalcocite ore. The coarse pyrite forms the top fringe, and includes large grains of chalcocite; then comes the chalcopyrite, with the finer iron pyrite and some of the fine chalcocite. On one table I noted a fringe of fine chalcocite behind the chalcopyrite. The inclination of all the tables, Garfield and Wilfley, is fixed.

The Wilfley concentrate (to which water is added) flows in a launder to the filter plant, while the tailing (which amounts to 182 tons) joins the roughing-tailing,

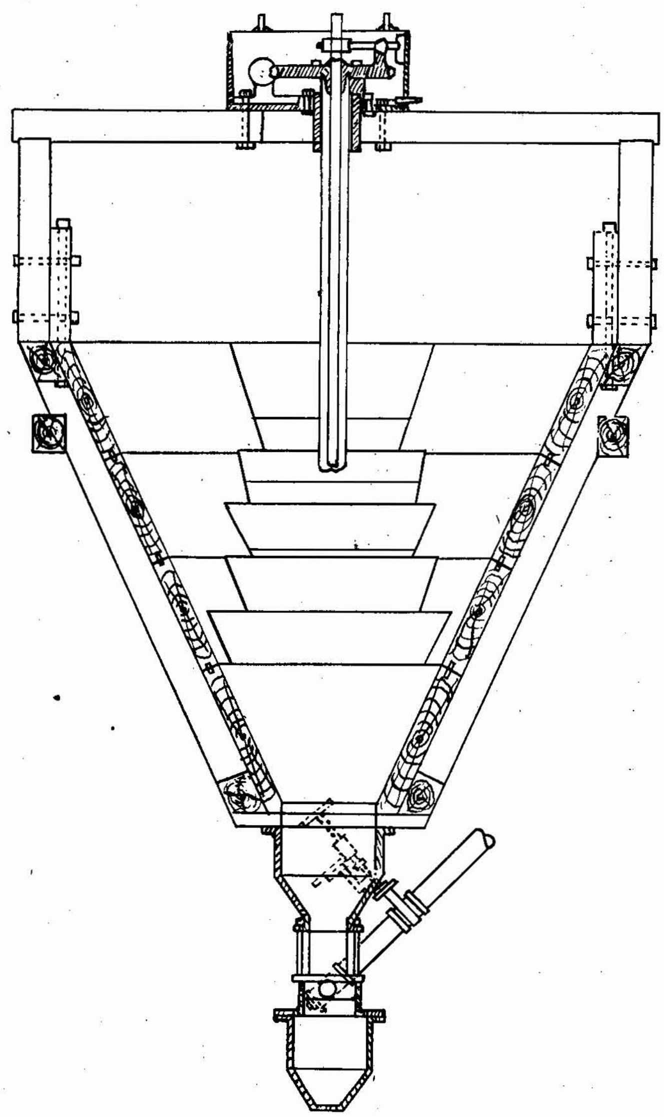


FIG. 3. THE RICHARDS-JANNEY CLASSIFIER

so that united they flow to a 30-in. bucket-elevator running at 390 ft. per minute and discharging into four primary 4-compartment Richards-Janney classifiers. The first and second spigots of the classifiers feed the Chilean mills, the third feeds the tube-mills, the fourth feeds the secondary classifiers, while the overflow goes to the slime-flotation plant. The primary classifiers appear to play a highly important part, but F. G. Janney, Jr., the General Superintendent, informs me that it is intended to replace them with drag-classifiers, because close classification is not essential now that flotation supplements the water-concentrating method, for the reason that the making of slime is no longer a drawback to the successful recovery of the copper sulphide.

THE RICHARDS-JANNEY CLASSIFIER was designed by the late Frank G. Janney, Manager of Mills for the mining companies under Mr. Jackling's direction; it embodies the Richards hindered-settling principle with the practical features of intermittent discharge, retardation of flow, controlled settling, and hydraulic regulation. In classification two general principles are involved, namely, free settling and hindered settling, respectively, according to the action of the material in the settling-column. In free settling, the action takes place in a column so designed as to present a uniform cross-section and consequent constantly rising current of water, so that the particles of material are allowed to settle against the specific gravity of the water alone.

The hindered-settling action, on the other hand, is caused by constriction in the classifying column, causing a difference in the velocity of the water in this constricted cross-section; which results in the formation of a zone of material of increased density in that portion of the column where the water-velocity is lower. This zone consists of particles that settle against the slowly rising flow in the larger column, but will not descend against the rapid flow in the constricted section. This zone of material changes the general density of the settling-medium, and raises the sorting efficiency. Greater success has been attained by hindered settling than by free settling, because the former assists the concentrating machine by furnishing a feed properly sorted according to mass.

When the Magna mill was re-modeled, 76 of the Richards-Janney classifiers were installed as part of the regular equipment. The result was an immediate increase of mill-capacity, together with a decrease in the cost of milling, and an increased recovery. These machines are also a part of the equipment of the mills operated by the Ray Consolidated and Chino Copper companies, in which the use of them has proved similarly advantageous.

The points to be emphasized in the design of a hydraulic classifier are close hindered-settling classification, low water-consumption, flexibility in capacity, non-dilution of slime, low cost of maintenance, and adaptability to change of load. As will be seen from the following description, these objects have been attained in large measure in the Richards-Janney machine.

The classifier consists of a series of rectangular compartments increasing in size and stepping down from the free end, the lower part of each compartment constituting the settling-chamber. A cylindrical sorting-chamber and hydraulic chamber are attached to the bottom of each compartment, the hydraulic chamber being in the form of a tee of rectangular section. The hydraulic inlet is governed by a gate-valve, and the bottom of the hydraulic chamber has a flanged end tapped to receive the inner valve-seat bushing. Attached to this flanged end is the retarding-chamber, provided with an air-cock and

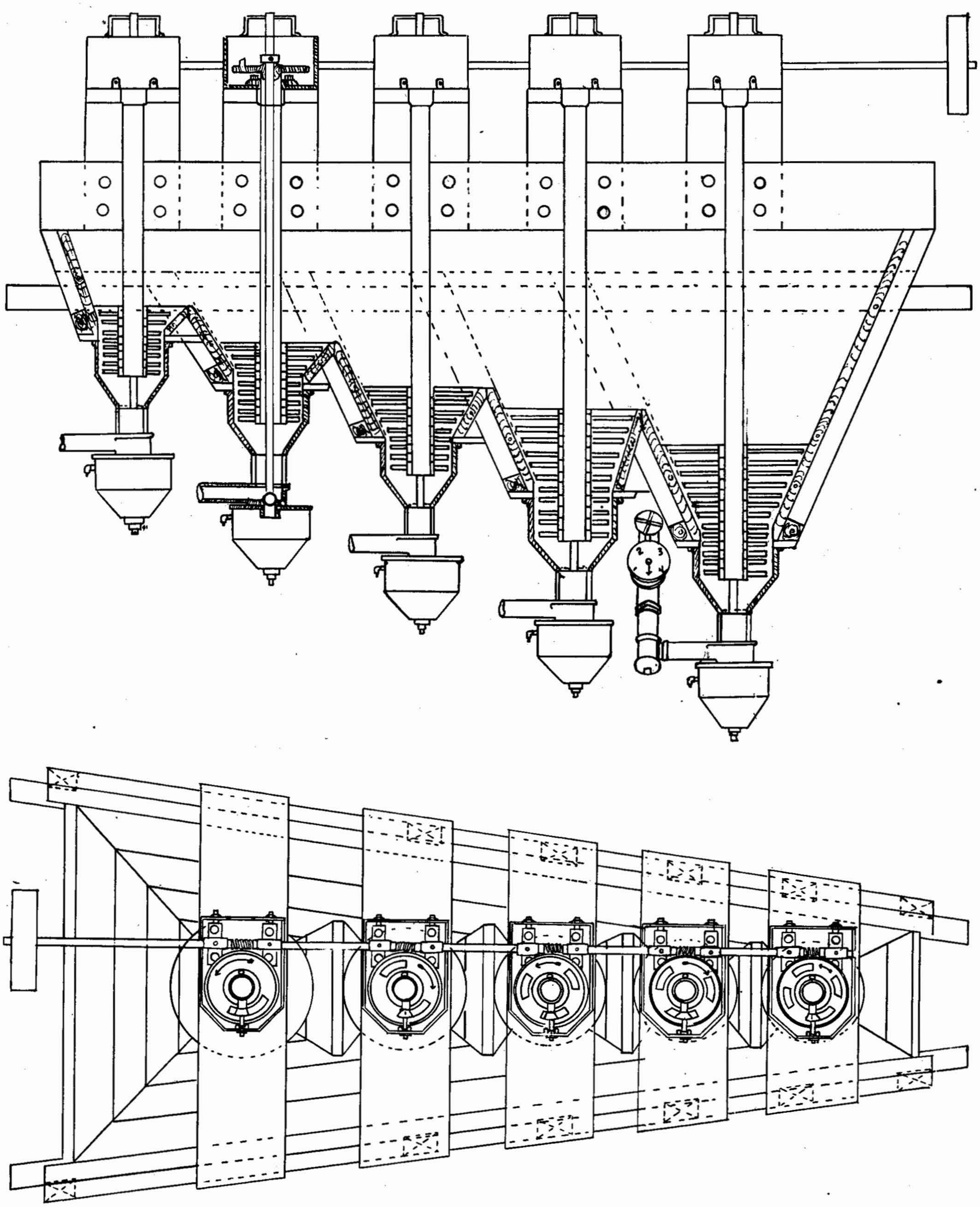


FIG. 4. PLAN AND SECTION OF THE RICHARDS-JANNEY CLASSIFIER AS USED IN THE ARTHUR MILL

tapped at the apex to receive the outer bushing. The feed-inlet and overflow-outlet are a few inches below the top of the sideboard, which extends along the top of the classifier for its full length. A cast-iron box with housing for the driving mechanism is supported above each compartment, the bearing for the spindle extending downward from the base of this box through the supporting frame. This spindle reaches downward into the centre of the sorting-chamber, and carries a series of sorting-blades driven through worm-wheels. Intermittent raising and lowering of the valve-rod are effected by means of cams attached to the worm-wheel. The valve-rod, which extends through the hollow spindle, is fitted at the lower end with a rubber valve, which sits on the inner bushing in the hydraulic chamber, as shown in Fig. 3 and 4. An arm fitted to the upper end of the valve-rod, and engaging with the cams on the worm-wheel, opens and closes the discharge-valve.

When in operation, the shaft revolves at about 80 to 90 r.p.m. Upon the feed entering the classifier the sorting-blades turn and loosen the pulp, without any raising or lowering action. The valve-rod is raised by the cam, opening the valve, which permits the material to pass through the inner bushing into the retarding-chamber. This chamber fills with sand and the valve-rod drops, closing the inner bushing. The material flows from the retarding-chamber through the outer bushing, and when the chamber is empty the classifier is ready for the next cycle.

Some of the main features of this classifier are close classification, economy of water, which ranges from 200 to 500 gal. per ton of ore, and large capacity. The standard 5-compartment classifier, from actual mill-tests, treats as much as 750 dry tons of ore per 24 hours, on material up to 7 mm. in size.

STAGE-GRINDING. Each section of the mill contains two Chilean mills, of 6-ft. diameter and of the Garfield type, the mullers making the circuit at the rate of 33½ per minute, and discharging through a No. 89 Tyler Ton-Cap screen having long narrow slots equivalent to 14 mesh. These grinding-machines are fed from equalizing-tanks, that is, storage-tanks intended to overcome the effects of a variable feed; they treat 432 dry tons of material per 24 hours. The product from the Chilean mill joins the rolls-product and goes to the roughing-tables, forming a closed circuit with the primary Richards-Janney classifiers. The feed and product of the Chilean mills are shown in the following screen-analysis:

	Feed		Product	
Mesh	Weight of material, %	Accum. weight of material, %	Weight of material, %	Accum. weight of material, %
On 10	3.46	3.46	...	...
14	24.21	27.67	1.28	1.28
20	23.27	50.94	4.24	5.52
28	17.02	67.96	9.43	14.95
35	12.80	80.76	12.38	27.33
48	8.07	88.83	11.37	38.70
65	4.20	93.03	9.19	47.89
100	2.14	95.17	7.18	55.07
150	0.99	96.16	4.84	59.91
200	1.12	97.28	6.21	66.12
Pass 200	2.72	100.00	33.88	100.00

The two drag-classifiers, which are fed from the third spigot of the Richards-Janney classifiers, are of the Dorr, or interrupted eccentric, type. Each of them delivers to a tube-mill, 7 ft. diam. by 10 ft. long, also home-made, and differing from other types in being equipped with ring-liners that are self-locking, thereby obviating the use of bolts. The absence of bolts is noticed by the visitor, but the chief feature is a shell that does not leak. Not only is the nuisance of loose liners avoided, but, it is claimed, an increase of grinding efficiency is caused by the anvil action afforded by the solidity of the lining. Each tube grinds 350 tons in 24 hours, the reduction being such that the proportion of material larger than 48-mesh in the general mill-tailing has been lowered from 35 to 10%. These mills revolve at a speed of 20 r.p.m. and are driven by 75-hp. motors.

In some of the mills Californian beach-pebbles of No. 3 and No. 4 size are used as a grinding medium; in others, Adamant cubes; and in others, Manganoid steel balls. The average consumption of the Californian pebbles is 3.16, of the Adamant cubes 2.03, and of the Manganoid balls 1.23, pounds per ton of material passing through the mill. The horse-power is 61.3 for the mills using the Californian pebbles, 60.09 for those using Adamant cubes, and 142 for those using Manganoid steel balls. The following screen analysis shows the feed and the product of the mills using Adamant cubes, which have been adopted as they have proved to be the most economical grinding medium.

	Feed		Product	
Mesh	Weight of material, %	Accum. weight of material, %	Weight of material, %	Accum. weight of material, %
On 10	1.03	1.03	...	...
14	11.26	12.29	0.24	0.24
20	16.89	29.18	1.60	1.84
28	19.29	48.47	4.83	6.67
35	22.19	70.66	12.82	19.49
48	15.93	86.59	18.31	37.80
65	7.66	94.25	15.62	53.42
100	3.58	97.83	8.51	61.93
150	1.09	98.92	6.32	68.25
200	0.47	99.39	6.90	75.15
Pass 200	0.61	100.00	24.85	100.00

The product from the tubes returns to the elevator that lifts the tailings from the roughing and Wilfley tables, and returns with them to the primary Richards-Janney classifiers, establishing a closed circuit. The overflow from the drag-classifiers feeding the tube-mills joins the overflows from the primary and secondary classifiers, going with them to the thickening-tanks that supply the slime-flotation plant.

Before leaving the fine-crushing department it might be well to state that the rolls and Chilean mills are each attended by a 25-ton crane, while one of 50-ton capacity is required for the tube-mills. All of the big repairs are made in a rigger-shed, where a gang of mechanics and laborers is constantly employed. One complete set of rolls, a Chilean mill, and a tube-mill are kept on hand, so that the time lost in repairing these machines is confined to the period required for removing and putting them back in place.

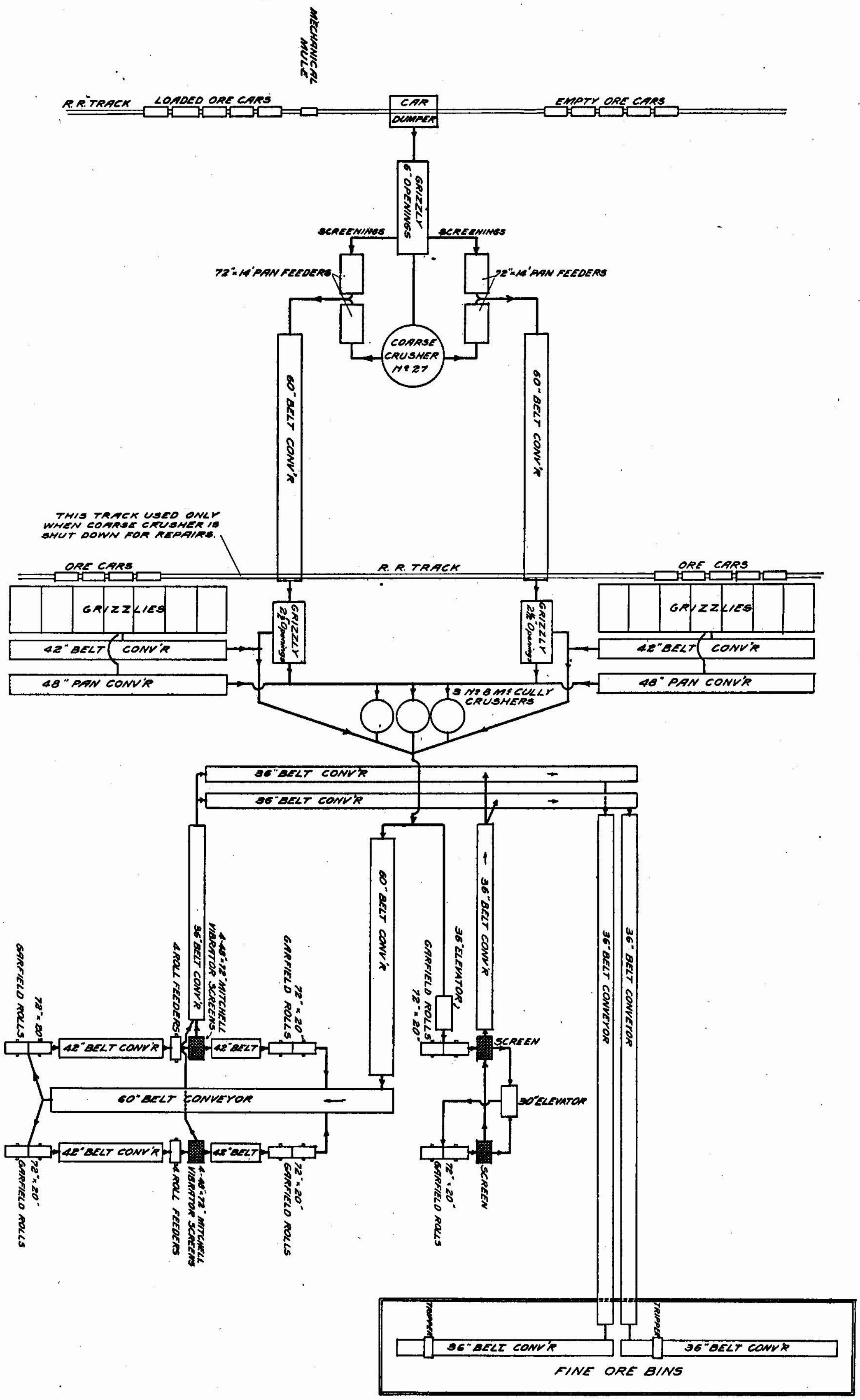


FIG. 5. FLOW-SHEET OF PRIMARY AND SECONDARY CRUSHING PLANTS IN THE ARTHUR MILL

VANNERS. The product (460 tons) from the fourth spigot of the primary classifiers passes to four secondary 5-spigot classifiers, also of the Richards-Janney type, where it is further classified by hindered settling so as to yield five sandy products and a slime which goes to flotation. Each of the five sandy products goes to a group of Isbell and Johnston vanners, thus:

First spigot to 8 Isbell vanners
Second spigot to 8 Isbell vanners
Third spigot to 8 Johnston vanners
Fourth spigot to 8 Johnston vanners
Fifth spigot to 8 Isbell vanners.

These vanners are provided with corrugated belts, 6 ft. wide and 10 ft. long; the Isbell is operated at a speed of 76 side-shakes of 1-in. amplitude per minute, and the Johnston at a speed of 132 side-shakes of 2-in. amplitude per minute. The slope of the vanner is 7° from horizontal and the travel of the belt varies from 60 to 120 in. per minute. The average load per vanner is 7.05 dry tons. The belts seem to give remarkably good service; one that I noticed had been in use for over two years, and yet the corrugations were perfectly sharp. This belt was made by the Republic Rubber Co., I was informed.

The vanner-tailing goes to waste, this being the first waste-product to be recorded. The concentrate is collected in a box and flushed with fresh water at intervals so as to push it into an elevator that lifts it to two hydraulic classifiers, each of five spigots, the products of the first four spigots of each classifier going to two Wilfley tables, while the product from the fifth, with the overflow, goes to Dorr thickening-tanks and thence to the flotation re-treatment plant. This subsidiary table-concentration plant, therefore, consists of eight Wilfley tables, each of which has a corresponding vanner. Each of the Wilfley tables produces a finished concentrate, a middling, and a tailing; the middling is returned to the elevator, and is in a closed circuit with the classifiers and Wilfley tables, while the tailing is re-concentrated on its corresponding vanner, which in turn yields a middling that is returned to the elevator and is in a closed circuit, while the tailing goes to waste. This department treats 359 dry tons of material containing 2.483% copper, and recovers 37 dry tons of concentrate assaying 22.35% copper, 20.1% iron, and 22.9% insoluble. It produces a tailing containing 0.2% copper, which goes to waste. The concentrate flows to the dewatering plant. The overflow from this last set of classifiers, with the product from their fifth spigot, goes to flotation, as I have stated, so that the water-concentrating operations end here.

On the first vanner-floor one detects the aromatic smell of flotation-oil; this becomes more intense as the flotation cells are approached, until in the midst of the flotation annex it has a lachrymose effect on the eyes. The spread of vanners on the lower floor is impressive owing to its extent, but I noticed that 20 out of 60 vanners in each section were idle, the slime that formerly went to them being now subjected to flotation, because as yet these vanners have not been equipped, as they will be, with the corrugated belts required to enable them to treat the sandy portion of the pulp. All the wooden vanner-floors have been propped, the ground underneath having been excavated previous to replacing it with concrete.

A laudable effort is being made at the two mills to erect attractive office-buildings and houses for members of the staff. The lawn and trees are pleasing to the eye, and must be much appreciated during the hot summer. A club-house and dormitory are to be erected at Magna, the dormitory to be open not only to the unmarried members of the staff but to any men of decent habits. In the centre of the group of buildings is a fountain erected as a memorial to Frank G. Janney. It reads:

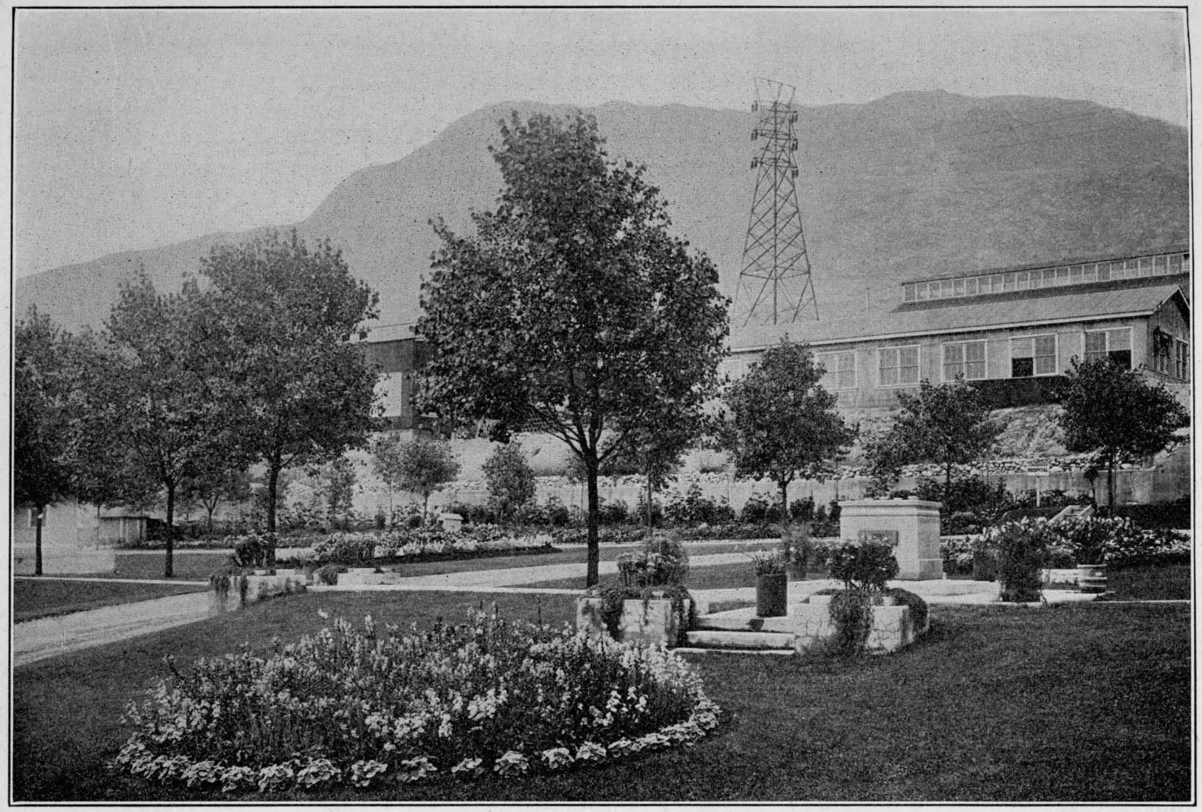
In Memoriam
1866 F. G. JANNEY 1916
Manager of Mills

This memorial erected by the employees of the Magna Plant in recognition of his impartial leadership and as a token of their undying esteem.

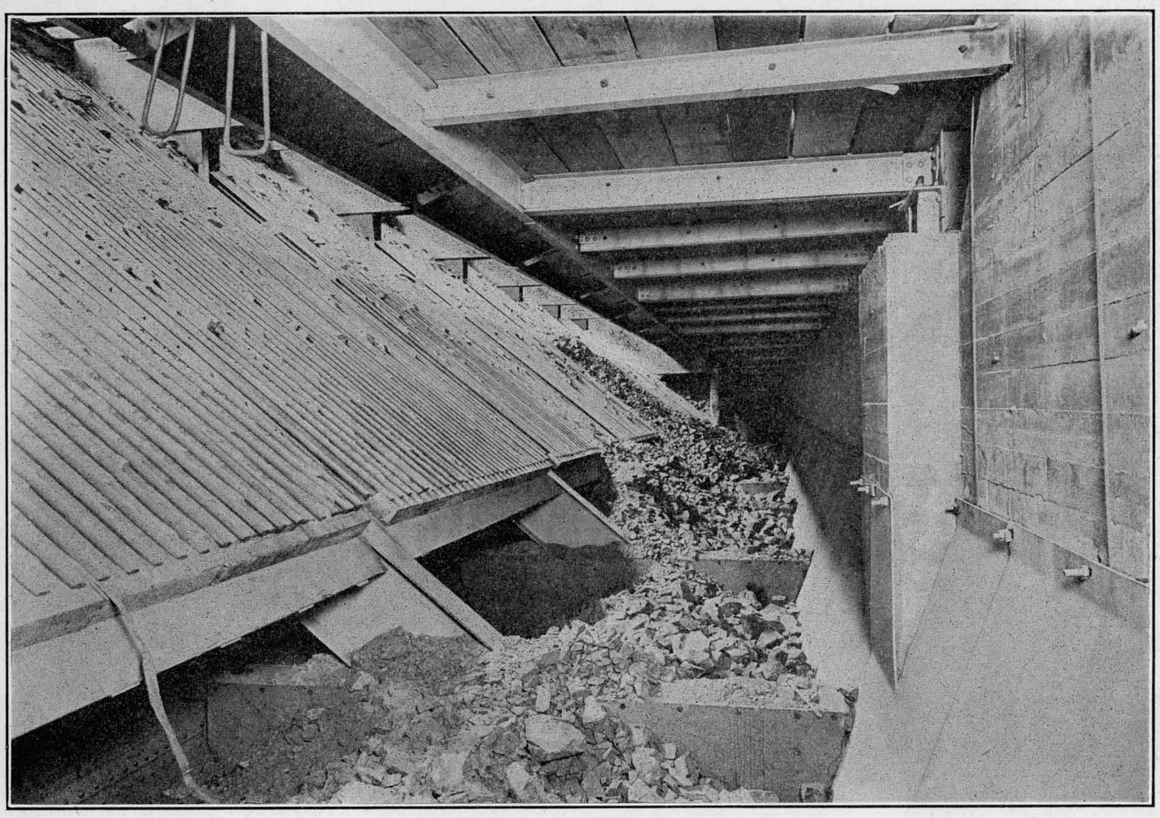
To live in hearts we leave behind is not to die.
Dedicated May 12, 1917.

This is a thoroughly American tribute to a good worker and captain of workers. The man so memorialized has left not only an honorable memory, but two worthy sons. Frank G. Janney, Jr., and T. A. Janney, the elder being now general superintendent of both mills and the younger the superintendent of the Arthur plant.

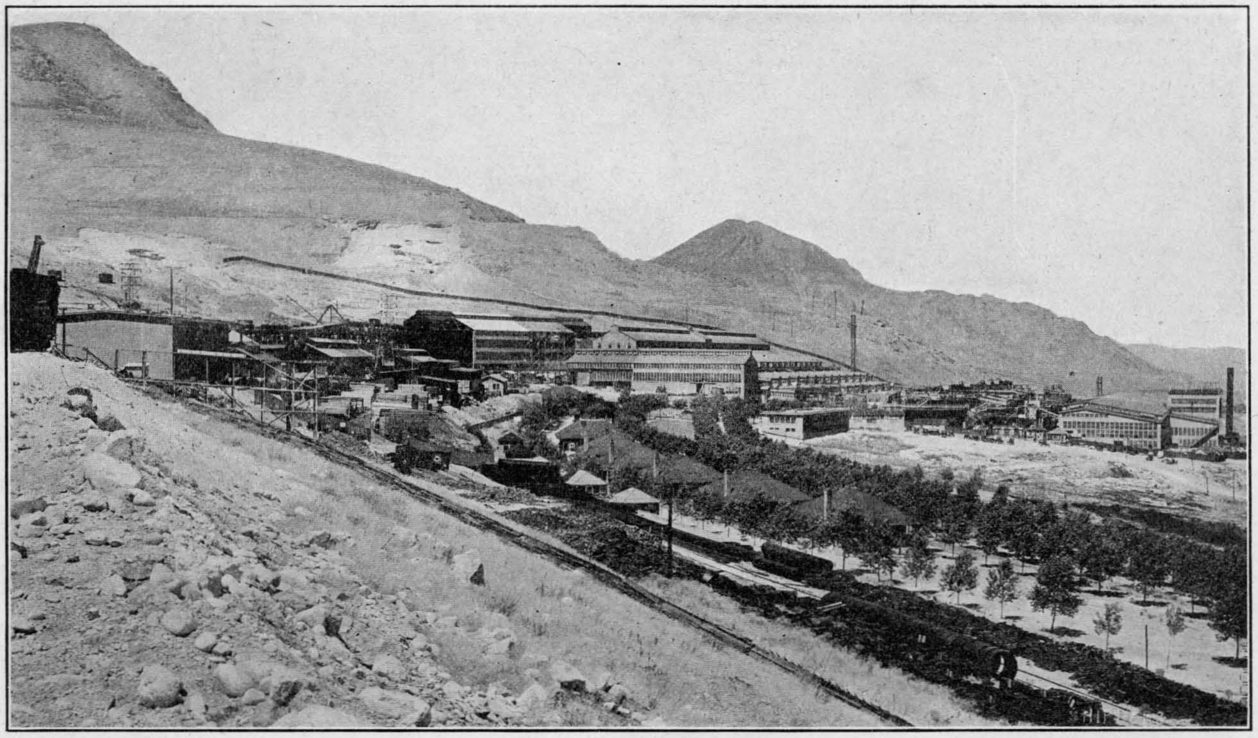
THE LATE FRANK G. JANNEY, MANAGER OF MILLS



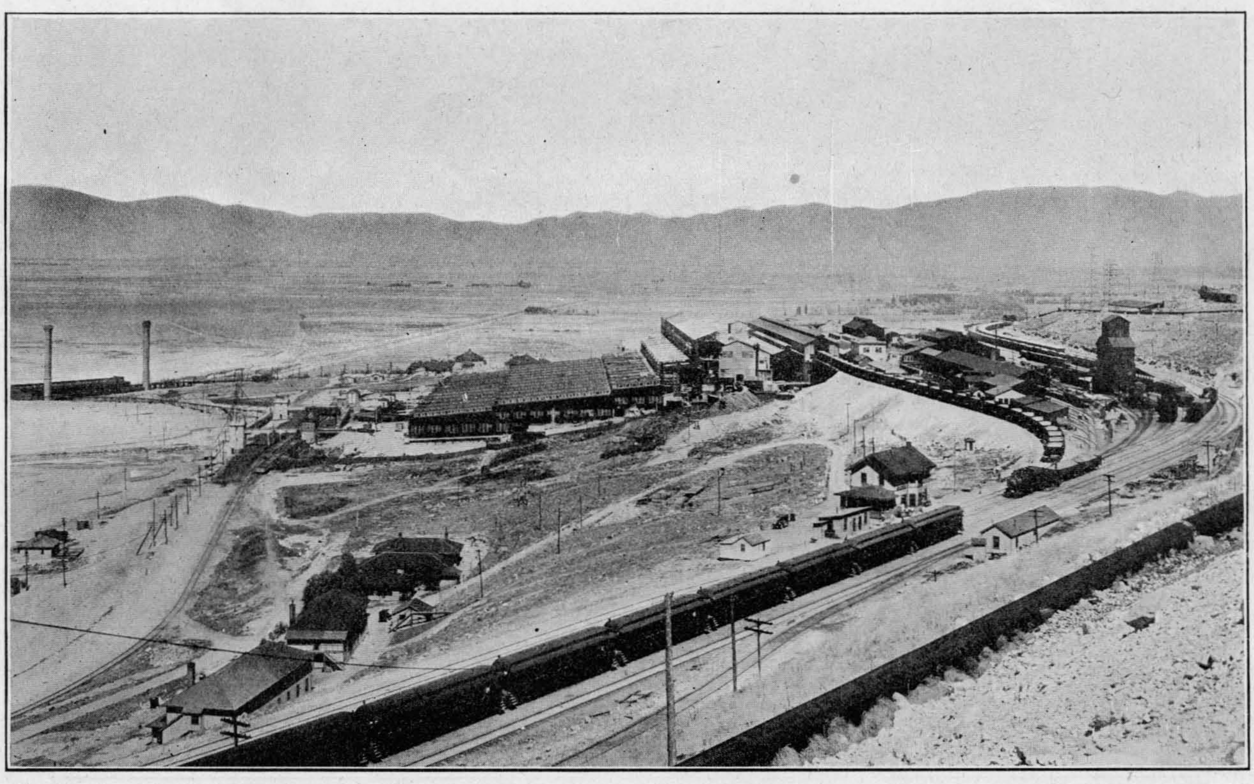
MEMORIAL TO THE LATE FRANK G. JANNEY AT THE ARTHUR MILL



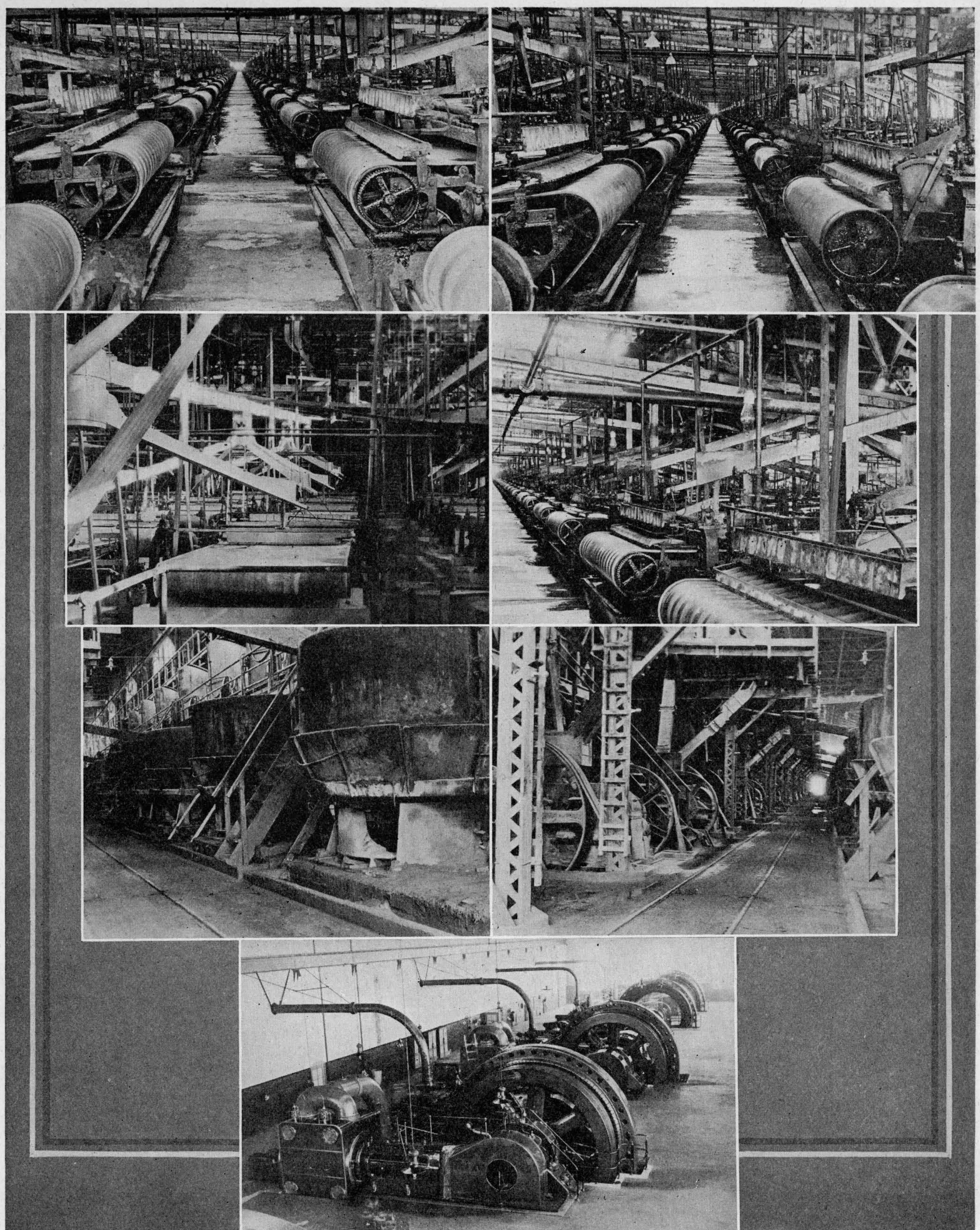
GRIZZLIES AND OVERSIZE HOPPERS



THE ARTHUR MILL, AT GARFIELD, LOOKING WESTWARD. RESIDENCES OF STAFF IN THE CENTRAL FOREGROUND



THE MAGNA MILL, AT GARFIELD, LOOKING TOWARD SALT LAKE. BINGHAM & GARFIELD RAILWAY ON THE RIGHT



VIEWS INSIDE THE MAGNA MILL

Second vanner-floor, looking west from Section No 1
Second vanner-floor, looking east from Section No. 1.
Lower vanner-floor
First vanner-floor, looking toward classifiers.
Chilean mills in fine-crushing department
15 fine-crushing rolls
Power-plant, looking west. Two 1250-kva. Allis-Chalmers units and two of the three 2000-kva. Nordberg engines.

FLOTATION PRACTICE

Treatment of Vanner Concentrate. This goes first to two 44 by 20 ft. Dorr thickeners, which increase the density of the feed from 6 to 24. The thickened product passes to two emulsifiers and 13 Janney cells of the mechanical type. They are placed in series, stepping six inches. All that is visible of this machine is the hood, which contains the motor. The vent-holes for the field-cores make it look like a beehive, the emblem of Utah.

This machine for froth-flotation was designed by T. A. Janney at the Arthur mill, and is the result of systematic experimental work in which the inventor was assisted by his late father, Frank G. Janney, and his brother, Frank G. Janney, Jr. Two types of machine have been developed. One is the 'mechanical', this term being used to designate the kind of agitation it produces, in contradistinction to the other type, which is based upon the pneumatic, as well as the mechanical, principle.

The 'mechanical' Janney consists of a chamber (22 in. diam. by 26 in. high) in which revolves a shaft carrying two impellers, each having four blades set at an angle of 45° and revolving at 570 r.p.m. This stirrer is actuated by an electric motor, to which the shaft is directly connected. When ready for work the hood containing the motor is the only part of the machine that is visible. Four baffles are fixed around the periphery of the chamber, the lower impeller moving inside these baffles on a level about one inch from the bottom, while the upper impeller moves one inch above the baffles, the effect being to ensure a complete and violent agitation of the pulp. Besides stirring the pulp, the impellers induce a circulation from the chamber to the spitzkasten and return. The spitzkasten, a V-shaped base attached to each side of the principal mechanism, receives the pulp after it has undergone such stirring as will create the froth essential to the process. The froth rises to the surface of the pulp in the spitzkasten and is skimmed into a launder. The skimmer consists of a blade attached to an interrupted eccentric that while revolving drops the blade into the froth so as to give first a vertical and then a horizontal motion, thereby avoiding the breaking of the froth. The amplitude of the sweep of this skimmer is regulated by the throw of the eccentric; the depth of the skimming is controlled by a bent rod to which the eccentric is attached and by which it is maintained in the desired position. An important feature of the invention is the circulation of the pulp. The feed enters through a gate at the bottom of the 'spitz' and is drawn, by displacement, into a pipe that takes it directly to the agitating-chamber; it is then thrown upward and outward near the top of the 'spitz'. Thus a constant circulation is maintained, the volume of pulp undergoing treatment being five or six times as much as the volume of feed or of discharge.

The 'mechanical-air' machine, as it is called, has the same kind of agitating-chamber, but the spitzkasten is provided with a porous bottom inclined at an angle of 35°. This porous bottom, or 'air-mat', is made of four plies of canvas through which air under a pressure of 3½ to 5 pounds is admitted. The air-pan is divided into three compartments, fed from corresponding pipes. The division is intended to compensate for the varying hydraulic head, thereby ensuring a uniform admission of air into the pulp.

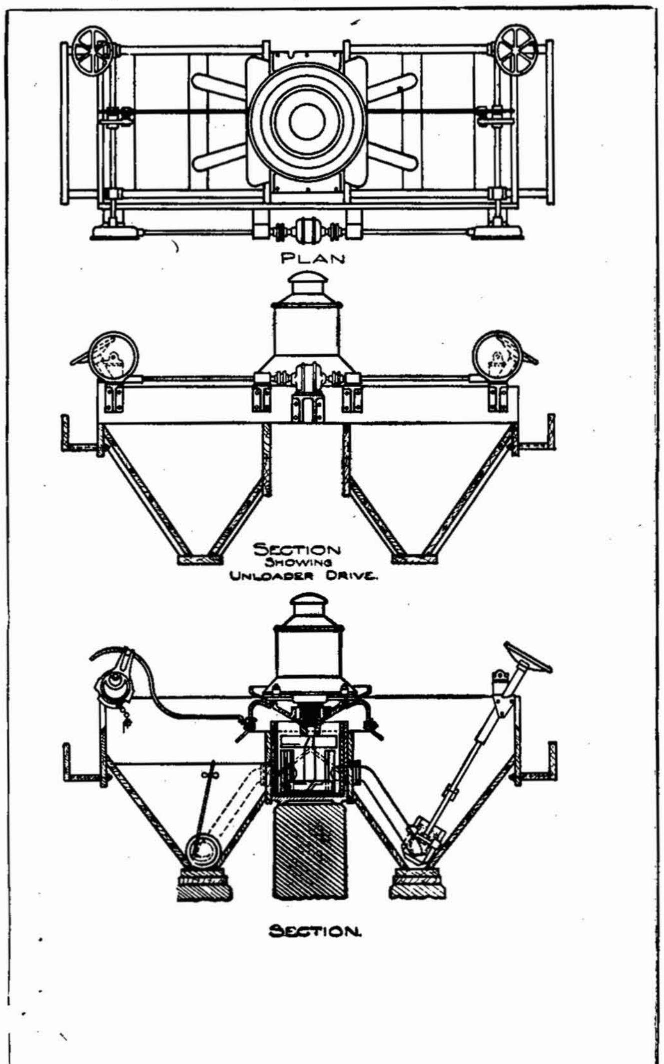
The first machine in a series of either type of Janney equipment is used as a preliminary mixer, or 'emulsifier'. At this stage no metallurgical product is forthcoming. Here oil or other modifying agents are added just before the feed enters the mixer, and the pulp is prepared for the actual froth-making that follows in the second machine of the series, whatever the number of machines. The oil is added by means of a special device, designed to secure the constant feed necessary for the operations that are to follow.

From this mixer, or 'emulsifier', the pulp enters the 'mechanical-air' machine at the side of the agitating-chamber (above the lower impeller); thence it is ejected into the spitzkasten at a level higher than the inlet. In this chamber, which is open under the motor, there is an indrawing of air, as shown by a lighted match. In the 'spitz' the further aeration of the pulp is effected by the air that enters through the porous bottom, causing bubbles to be generated freely in the pulp, so as to buoy the particles of mineral to the surface.

The froth produced by the two types of machines differs notably, that of the 'mechanical' being denser and tougher than the froth in the 'mechanical-air', the product of which is more readily broken after it has been discharged, because, of course, once the froth has done its duty, it becomes a nuisance. The sooner it differentiates into liquid and solid the better. The reject of this operation descends over the air-mat to be returned by an interior pipe from each 'spitz' back to the agitating-chamber, the excess escaping over a weir into a launder or pipe that takes it to the next machine in the series.

The froth from the 'roughing' machines is re-treated in the 'cleaners', to which it is pumped. The tailing from the 'cleaners' is pumped back and rejoins the feed to the 'roughers'. On the other hand, the reject from the 'mechanical' Janney goes direct to waste.

Turning to the drawing of the 'mechanical-air' machine, let us follow the flow of the pulp. It is fed at *A* and enters the opening *B,* part of it being thrown out at *C,* to rejoin *A,* producing a half-circulation at this stage. The other half is ejected at *D,* where the pulp drops into the box *E,* from which a portion is again drawn into the



JANNEY MECHANICAL FLOTATION MACHINE.

mixing-chamber through the pipe *F,* producing the other half of the circulation. The excess, equal to the feed, escapes through the pipe *G* into the next machine, which it enters at *H,* where it is ejected, at the points *KK,* into the spitzkasten, passing over the air-mats *LL* to the box at *M,* where part of the pulp passes back through the pipe *N,* joining the original feed of this machine at *OO,* while the excess escapes at *M,* the sand dropping through the spigot *S,* while the slime rises over the weir *P* and falls into the box, where it joins the sand, the joint products passing through *T* into the next machine.

Returning to the flotation of the vanner-concentrate; a sample of the froth when shaken in a pan and patted with the hand, so as to break the bubbles, shows the black copper sulphide that has settled underneath. The first six cells produce a concentrate containing 25% copper, 21% iron, and 20% insoluble. The number of cells in use to produce a finished product is regulated by the character of the feed, any cell being made to yield either a middling or a concentrate by shifting the baffle that divides the launder lengthwise into two compartments. The product of the last seven cells is a middling, which is returned for re-treatment.

The oil-mixture consists of mineral, pine, and creosote oils. The feeding of the oil is effected by a special machine of water-wheel type; it is a wheel having small buckets on its periphery dipping into a tank of oil that is kept at a constant level by a constant inflow, the excess returning to a storage-tank, to be returned by a centrifugal pump. The tank to which it returns is equipped with a steam-coil so as to maintain the oil at a uniform consistence. The speed of the feed-wheel is controlled by a friction-disc that regulates the speed as required, maintaining the feed of oil in excess of 10 lb. per ton of ore. A sulphonated alkali, called 'calura', is added to the pulp before it enters the two emulsifiers in order to act, like acid, as an electrolyte; and thereby increase the efficiency of the process, it having been found that the feed requires an alkali instead of an acid. The slimed feed requires an acid; the granular feed an alkali. The amount of vanner-concentrate sent to flotation is 466 tons daily; that of slime is 12,000 tons.

TREATMENT OF SLIME. The slime from the water-concentrating mill runs into ten Dorr thickeners, six of which are 75 by 12 ft. and four 75 by 18 ft. Four more, 75 by 18 ft., are being added. The density of the inflow is 8, and the density of the outflow 24.

From the thickeners the pulp passes to three small equalizing-tanks (8 by 12 ft.). Eventually six equalizers will serve 14 thickeners. The purpose is to correct any fluctuation of feed to the flotation machines, and also to mix the feed with the middling that is returned from the cleaner-cells of the flotation department. Acid is added to the equalizers at the rate of three pounds of 60° B. sulphuric acid per ton of ore.

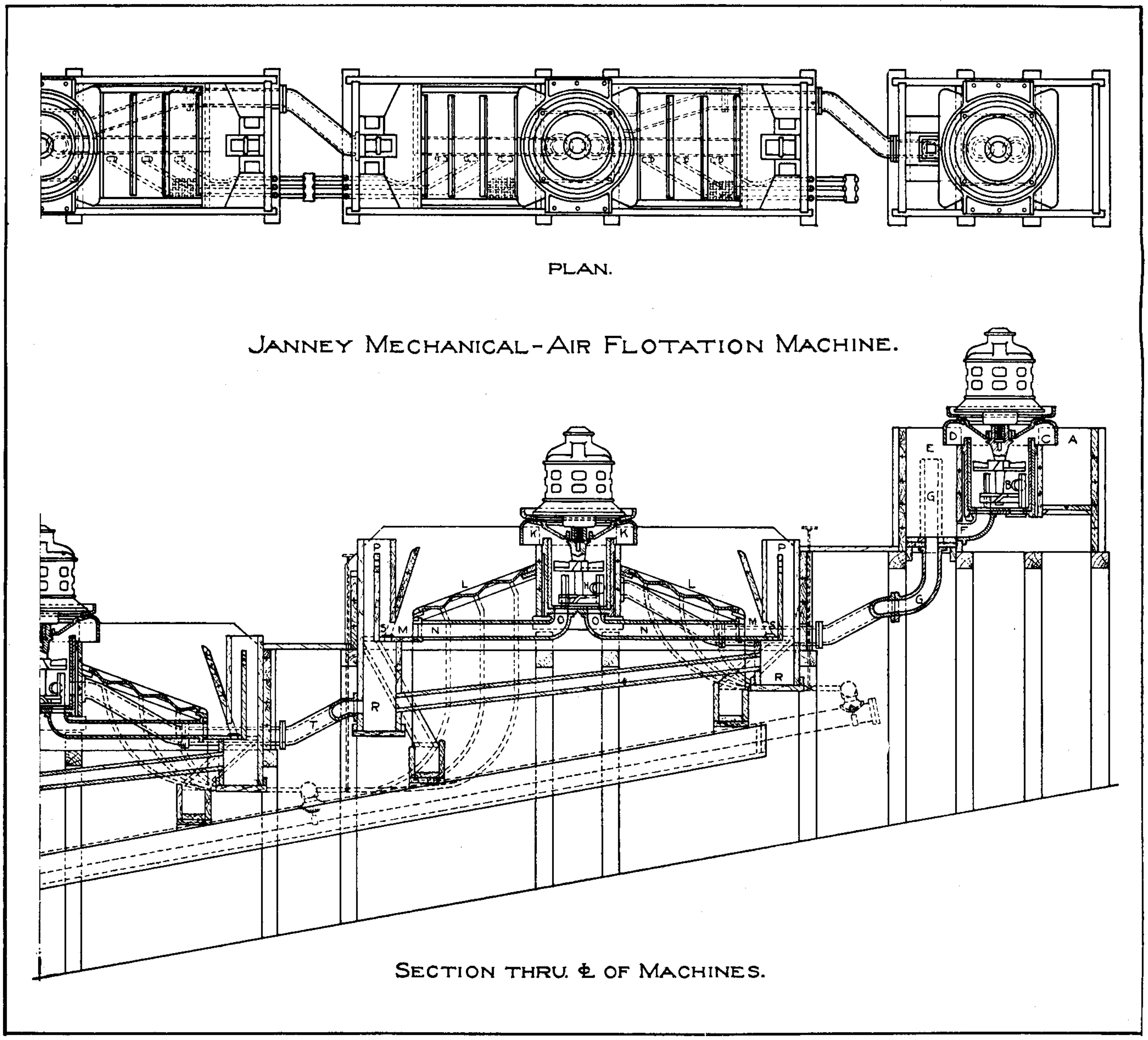
From the equalizers the pulp is conducted through 12-in. wooden pipes into a 24-in. concrete pipe that extends the entire width, 583 ft., of the flotation plant. From this conduit the pulp is distributed by 4-in. pipe to each row of cells—each row consisting of one emulsifier and five cells. There are 11 such rows to each unit of the plant, which when complete will consist of 11 units, 8 of them roughers and 2 cleaners, and one a recleaner. Thus there are 726 machines, of which 605 are working-cells and 121 emulsifiers. The emulsifier serves to distribute the oil equally throughout the pulp before it enters the working-cell.

In the roughing-cell a low-grade concentrate is produced and a clean tailing, the concentrate itself being re-treated in the 'cleaner', while the tailing goes to waste. The tailing from the cleaner is returned to equalizing-tanks, in which it is mixed with fresh feed. The oil, at the rate of more than 10 lb. per ton of ore, is added by means of a special feeder, which consists of a flat-faced pulley that dips into a small tank of oil and thereby collects a film, to be removed by a scraper impinging on the face of the pulley just above the point where it enters the oil. This feeder is adjustable by the width of the overlap of the scraper. See next page.

No skimming device is used in the air-cells. The froth rises sufficiently fast, owing to the large volume of air forced through the porous bottom, to push the mineral-laden froth over the edge of the cell.

The air-machine in use at the time of my visit consisted of five twin-cells, back to back. This entire plant was made of concrete and tile, the latter for piping. Each cell was 9 ft. 4 in. by 4 ft. 3 in. The porous medium was a four-ply canvas stretched crosswise doubly on half-inch centres. Experiments were being made with concrete mats, because the canvas wore out after being in use for eight weeks. Air was admitted at the rate of 150 cubic feet per minute under $4\frac{1}{2}$ lb. pressure. No air-machines are now in use.

The low-grade concentrate from the roughers is returned to two equalizing-tanks, 20 ft. diam. by 30 ft. high, from which it flows into a 24-in. concrete pipe and is distributed to each cleaner-cell through a 4-in. pipe made of concrete and tile. No oil is added to the cleaner, nor to the cleaner-tailing only at the head of the roughing-cell. The mixing of the oil for the mechanical-air machines is done by the emulsifier, the oil being added through valves on a 6-in. header, or pipe, through which runs a steam-pipe, maintaining the oil at a uniform consistence, the oil running from this pipe through $\frac{1}{2}$-in. globe-valves directly to the emulsifier. On account



PLAN.

JANNEY MECHANICAL-AIR FLOTATION MACHINE.

SECTION THRU. ℄ OF MACHINES.

of the large quantity of oil used it is not necessary to employ a mechanical feeder, the volume of oil sufficing to induce a continuous flow that is readily controlled by the valve.

The composition of concentrate and of tailing is as follows:

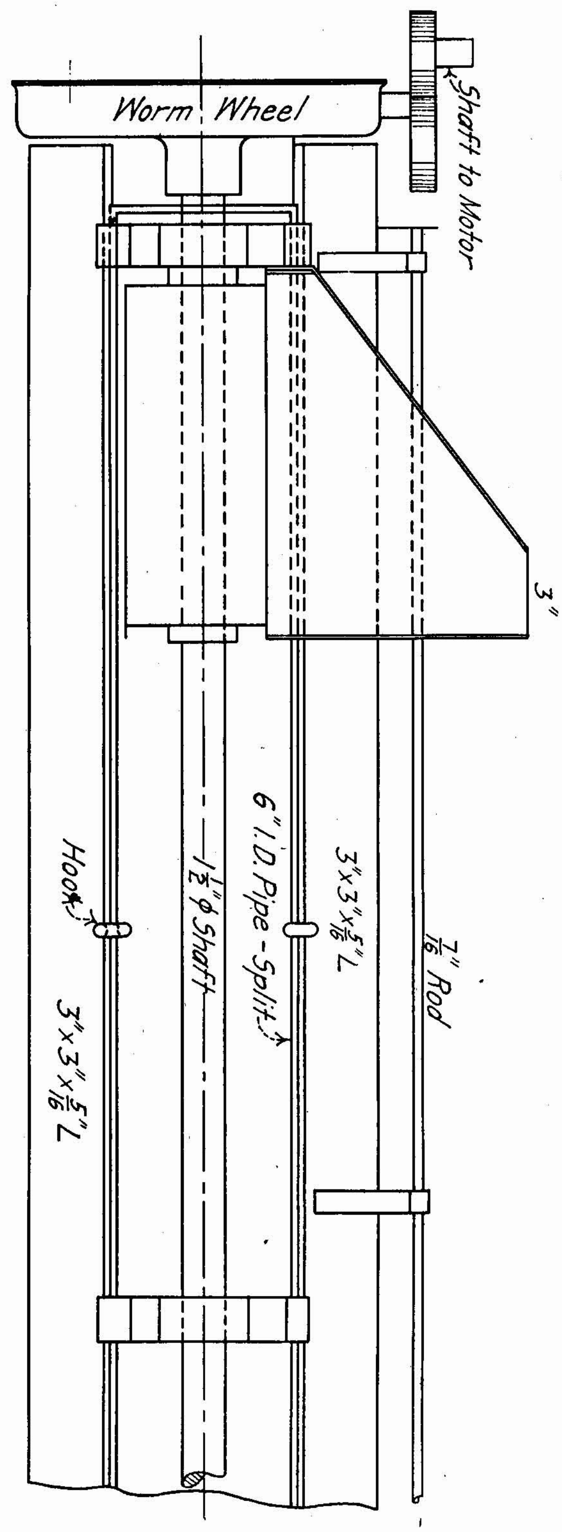
	%
Concentrate:	
Copper	18
Iron	17
Insoluble	34
Tailing:	
Copper	0.25

From five units the products run into wooden launders; from three units the rougher product and tailing, as well as the cleaner-tailing, drop in a cascade directly into separate concrete launders, the tailing-launder being 6 ft. wide and the concentrate-launder 10 ft. 6 in. wide. These deliver their contents to the return-pumps and main tailing-launders are required. The effect is to produce a small metallurgical Niagara, the function of which is to break the bubbles and produce a product more readily pumped.

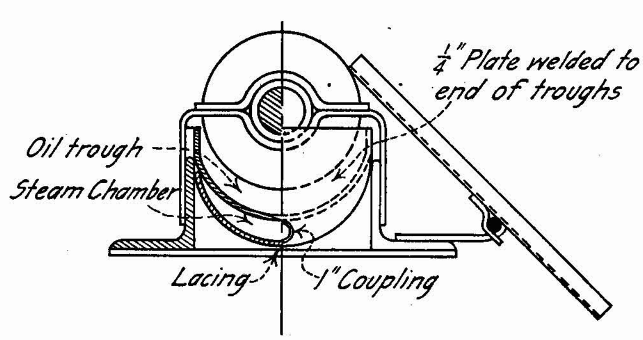
At the far end of the annex, looking down over a series of five double-cells, there is visible a big surface of froth —a veritable boiling sea—the five units occupying a space 100 by 265 feet.

Looking at a series of machines one can note differences in the froth due to causes recognized by the experienced millman. The best type of froth consists of light-gray bubbles, fairly large (two or three inches), of an evanescent character, that is, they burst as soon as they have done their duty of buoying the valuable mineral. A bad froth is lighter in color, the bubbles smaller, and not bursting sufficiently soon. Here I may mention that manipulation is an important factor in flotation. Much of it is empirical rather than scientific, it is not patentable nor is it well adapted to description, therefore the importance of it has not been emphasized either in litigation or in technical literature, but it has played and will continue to play a decisive part in successful flotation.

The two kinds of froth-concentrate are conducted by concrete launders to elevators that discharge into Dorr thickening-tanks, 75 ft. diam. by 20 ft. high, of which eight are in operation and one is being constructed. Here the density is increased from 9 to 51. These tanks are operated in pairs and in series, also intermittently, the idea being to operate a pair until the overflow from the second tank indicates that the suspended solid has accumulated to a level near the discharge, then the feed is diverted to another pair of tanks, until the same stage of repletion is reached. Experience shows that these tanks cannot be used as efficiently in parallel continuously as in series intermittently, because the range of the settling-rate is so great in the froth-concentrate that the colloids accumulate and remain in suspension within the tank.



OIL-FEEDING DEVICE



FILTRATION. The concentrate produced by water, in the upper mill, flows in concrete launders to four drag-classifiers, which yield two products: (1) the slime, which carries the bulk of the water, joins the froth-concentrate, and flows to a bucket-elevator and is raised to a Dorr thickener (75 ft. diam. by 20 ft. high); (2) the coarse portion, which contains about 10% moisture, is discharged upôn a belt-conveyor (leading to elevators) that unites this product with the thickened flotation product coming from the other Dorr thickeners. Thus the water-concentrate is mixed with the froth-concentrate before undergoing filtration.

The filter-plant consists of eleven 14 by 14 ft. Portland filters. Another is to be added shortly. Of these 8 to 11 are in constant use. They consist of revolving drums on which a porous medium of canvas is stretched; this is immersed in the concentrate during a part of its revolution, the pulp being drawn to the canvas by the vacuum induced within the drum. The pulp is sucked, and becoming dewatered, is detached by compressed air, which replaces the vacuum. Three Ingersoll-Rand vacuum-pumps of 9900 cu. ft. capacity per minute, serve to operate the filters, which make one revolution every 15 minutes. The vacuum capacity is equal to 1.5 cu. ft. of displacement per square foot of filtering area. The filter-cloth is No. 3 Oakdale canvas. Live steam is injected at the bottom of the hopper in order to heat the pulp and also to agitate it, keeping the coarse particles in suspension. These filters are not equipped with any mechanical device for producing agitation, because experience has shown that no agitation is required other than that caused by the introduction of the steam, as described. The hotter the pulp, the dryer the cake. As the vacuum withdraws the moisture the pulp forms a cake that is loosened by compressed air and removed from the surface of the revolving drum by a scraper that rides on the wires that wrap the filter. The scraper consists of steel plates, which warp and have to be replaced frequently. Experiments are being made with a scraper made in sections, so as to allow closer and more frequent adjustment, and thereby induce a clean removal of the cake.

From the filter the concentrate drops upon a belt-conveyor leading to a short inclined conveyor that discharges directly into a railroad-car. One man, with a hoe, is employed to supervise the loading. Five men are employed in the filter-plant per shift.

THE LEACHING PLANT

The ore to be leached is the oxidized cap that is stripped from above the main mass of sulphide ore. It averages 0.65% copper in the form of the carbonates (malachite and azurite) with a minute proportion of the silicate (chrysocolla), and contains an additional amount of copper, 0.1 to 0.2%, in the form of chalcopyrite and chalcocite. The principle underlying the metallurgical process is borrowed from nature, for sulphuric acid, derived from the decomposition of sulphide mineral, is used to dissolve the copper, which is then precipitated upon scrap-iron. The plant has a capacity of 2000 tons of ore per day.

The railroad-cars, containing 64 dry tons each, are discharged over grizzlies made of I-beams covered with a mushroom top of manganese-steel. Pieces of ore too large for the openings, which are nine inches wide, are broken with sledge-hammers. The ore then falls on a second set of grizzley-bars, three inches apart, the oversize going to steel bins that discharge upon steel caterpillar-apron feeders delivering to 5-ft. Stevens-Adamson pan-conveyors, by which it is discharged to screens, with one-inch openings, ahead of two gyratory crushers, No. 6 Gates, style K. From these crushers the ore passes to a belt-conveyor leading to a sizing-screen, inclined at 45°, and made of ⅝-in. wire with 1-in. openings. The oversize from the screen goes to Garfield rolls, 72 by 20 in., while the undersize passes direct to a hopper discharging upon the belt-conveyor that feeds the vats. The undersize from the 3-in. grizzley passes to another sizing-screen, similar to the one already described, from which the undersize falls upon the hopper discharging upon the belt-conveyor feeding the leaching-vats, while the oversize joins the crusher product on the conveyor ahead of the primary rolls. The product from these rolls is conveyed to a screen ahead of the secondary rolls, the oversize being returned by a conveyor after passing through the secondary rolls to the screen ahead of the primary rolls, while the undersize is delivered to the vats. The belt-conveyor carrying all the crushed ore discharges upon a 4-ft. pan-conveyor provided with a transverse slot through which one part in 400 is removed as a head sample. This pan-conveyor delivers to a hopper that feeds a 42-in. rubber-belt conveyor discharging by means of a tripper traveling with a loading-bridge equipped with a Robins tripper-conveyor, discharging into and filling the leaching-vats. The product from the crushing department shows about 10% coarser than half an inch, more than 40% coarser than 3-mesh, and only 20% finer than 20-mesh. About 39% of the copper rests with the material between ½-inch size and 4-mesh. Evidently this is the critical point in the crushing; 58% of the weight of the ore containing 48% of the copper in the ore has been separated at this stage. A typical screen-analysis is given herewith:

Screen-Analysis

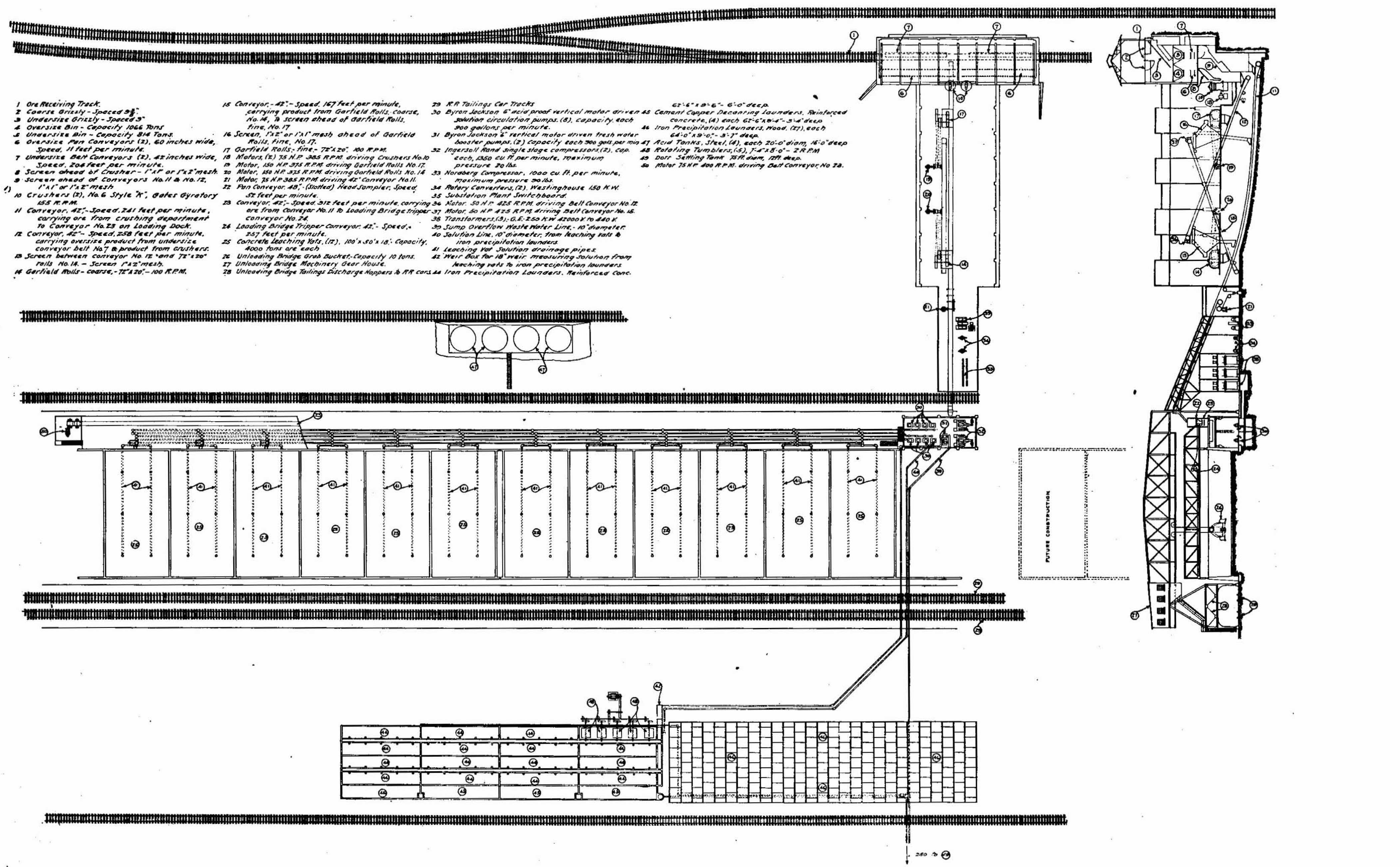
Assay of tank No. 9 heading; October 2, 1918; run No. 87
Original weight 3335 grammes:

Mesh, inch	Material by weight %	Cumulative material by weight %	Copper by assay %	Weight copper %	Cumulative weight copper %
1½	0.00	....	0.71	...	....
1¼	0.00	....	...	...	....
1	0.00	....	...	...	....
¾	0.57	0.57	0.26	0.21	0.21
½	8.75	9.32	0.76	9.49	9.70
3	35.14	44.46	0.59	29.58	39.28
4	11.48	55.94	0.58	9.50	48.78
6	7.38	63.32	0.64	6.74	55.52
8	5.76	69.08	0.60	4.94	60.46
10	4.89	73.97	0.63	4.39	64.85
14	4.32	78.29	0.71	4.38	69.23
20	2.78	81.07	0.72	2.85	72.08
28	2.13	83.20	0.70	2.13	74.21
35	3.18	86.38	0.79	3.58	77.79
48	2.10	88.48	0.83	2.48	80.27
65	2.55	91.03	0.96	3.50	83.77
100	1.65	92.68	1.08	2.54	86.31
150	1.17	93.85	1.21	2.03	88.34
200	1.83	95.68	1.30	3.40	91.74
Through 200	4.32	100.00	1.34	8.26	100.00
	100.00		0.70	100.00	

The 12 leaching-vats are 100 ft. long, 50 ft. wide, and 17½ ft. deep. They are arranged in two contiguous groups of six each, so as to neutralize the expansion. They are made of concrete, heavily reinforced. The walls are lined with mastic, a mixture of

	%
Trinidad asphalt	25
Sarco putty No. 4	4
Gilsonite	4
Cement	5
Sand (minus ¼)	67

The mastic is 1½ in. thick and is reinforced with 2 by 2 by 0.162 in. diam. crimped wire. The floors of the vats are coated with mastic half an inch thick. Upon this floor come 6 by 6 in. stringers, spaced 10 in. apart, laid lengthwise of the vat. The stringers are 9 ft. 6 in. long; their ends are beveled 3 in., and they are 3 in. apart. Across them are placed wooden stringers 2 by 8 in. and $\frac{5}{16}$ in. apart. On these is spread cocoa-matting ¼ in. thick; then more 2 by 8 pieces, $\frac{5}{16}$ in. apart, so that the two-by-eights lie across each other, separated by the mat.



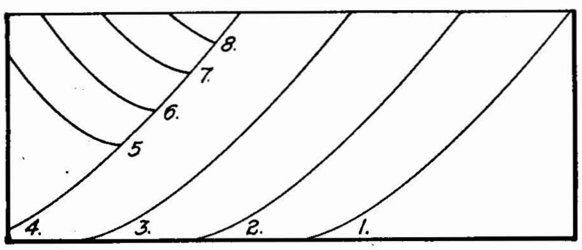
THE LEACHING-PLANT OF THE UTAH COPPER CO., AT GARFIELD, UTAH

Ample pumping-plant capacity has been provided for the transfer of the solution as required. The intention is to resort to a cyclic advance of solution. All the valves are of the disc-and-gate type, lined with lead, whereas the pipes are made of redwood staves that have been immersed in paraffine, wound with hard-drawn copper wire. When received from the Redwood Manufacturers Co. they are immersed in hot asphaltum paint. The cast-iron fittings and connections are lined with mastic. The mastic mixtures used for different purposes are as follows:

	Floors %	Walls %	Fittings %	Weight per cu. ft. lb.
Trinidad asphalt	25	25	25	89
Sarco No. 4	..	4	5	62
Sarco putty	1½	..	..	109
Gilsonite	..	4	5	63
Cement	..	5	7	83½
Sand (minus ¼)	73½	67	58	*96 †85
Pioneer paving cement	..	..	..	53½
Floor mixture	..	..	..	120
Wall mixture	..	..	..	134

*Dry. †Damp.

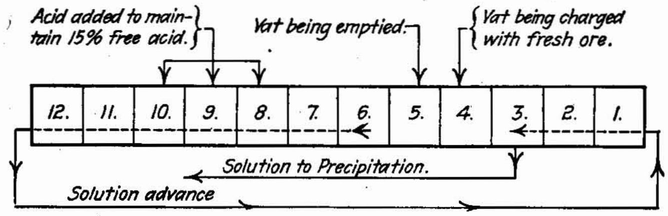
When charging, the tripper-conveyor drops the ore at one side of the vat and continues to feed on the same side until the toe of the slope reaches the opposite side; then the filling re-commences from the opposite wall into the V-shaped space that has been left by the preceding operation. This is the method adopted at Chuquicamata.



METHOD OF FILLING THE TANKS

The idea is that the coarser particles of ore run to the bottom of the vat; if the filling were to advance continuously from one side, the coarse ore would extend up the opposite side, producing conditions unsuitable for the uniform percolation of the acid solution.

Fresh ore is fed into the vats in sequence, the freshest ore being subjected to the oldest solution, that is, the one containing the most copper and the least free acid. At the time of my visit, No. 5 vat was being emptied of its tailing, while No. 4 was being filled with fresh ore. The last stage of solution was taking place in No. 6, thus:



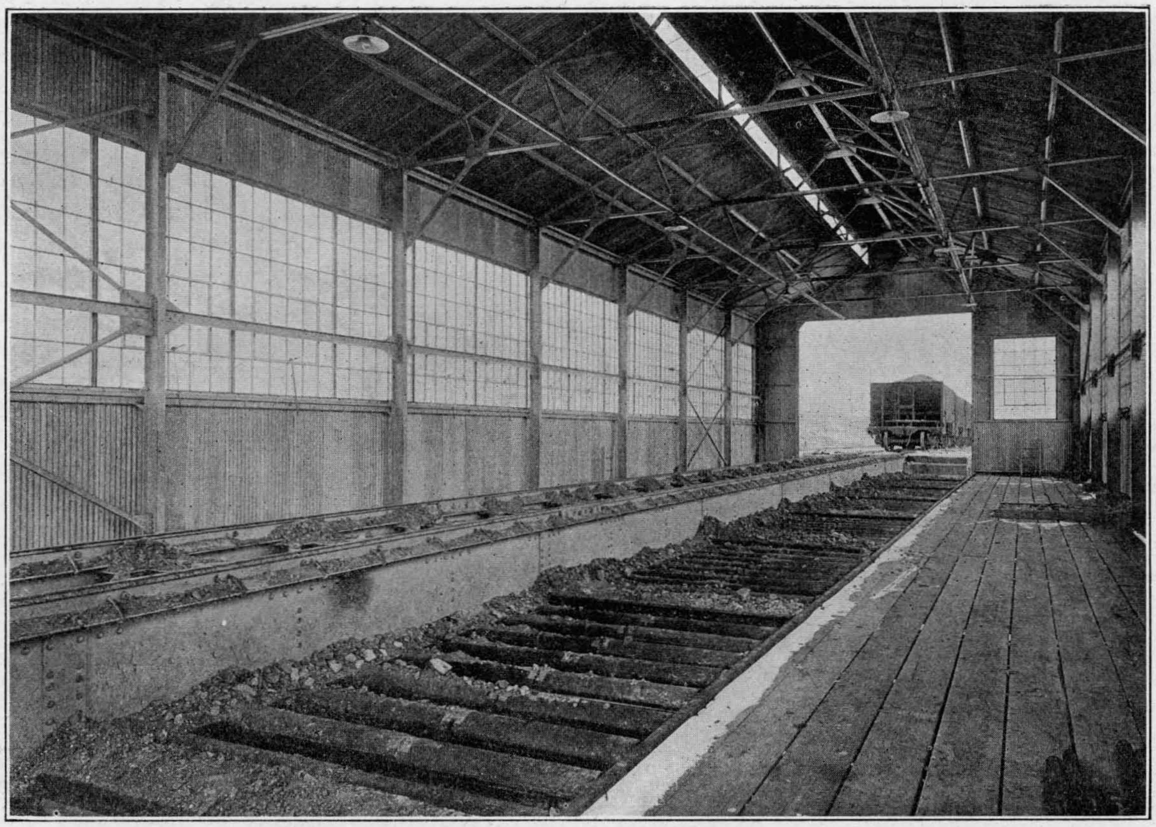
CYCLE OF LEACHING

The solution is advanced at the rate of approximately 80 gallons per minute through the vats in series from 6 to 7-8-9-10-11-12-1-2-3. After all the solution in No. 6 has been advanced to No. 7, wash-water is added until the effluent from the tank contains from 0.08 to 0.15% copper. The wash-water advancing in the cycle is gradually enriched with free acid and copper, the first addition of acid having been made to a selected number of the vats, No. 8, 9, and 10, in the cycle. From this point no more acid is added, the free acid being gradually neutralized by dissolving the copper, iron, aluminum, etc., content in the ore until the solution on the newest ore, or that last charged into the leaching-vats, contains 3.16% copper, 0.37% iron as ferrous, 0.95% as ferric iron, and 0.24% free sulphuric acid. The intention is not to add acid at any stage in such measure as to cause the proportion at the close to exceed 0.2%. Thus, when the cycle of operations brings the solution to the fresh ore, the acidity is at its minimum and when the copper-bearing solution (the product of the leaching process) is drawn away from precipitation, it contains not more than 0.2% free acid. To express the idea in another way, the acidity, or strength of the active solution, is brought to a peak in the middle of the series of operations, so that when it is applied to the new ore and likewise when it is withdrawn, enriched by 2½% copper, it contains not more than 0.2% acid. The rich solution is drawn from the vat containing fresh ore, because, owing to the soluble components, it can neutralize the free acid quickest at this stage.

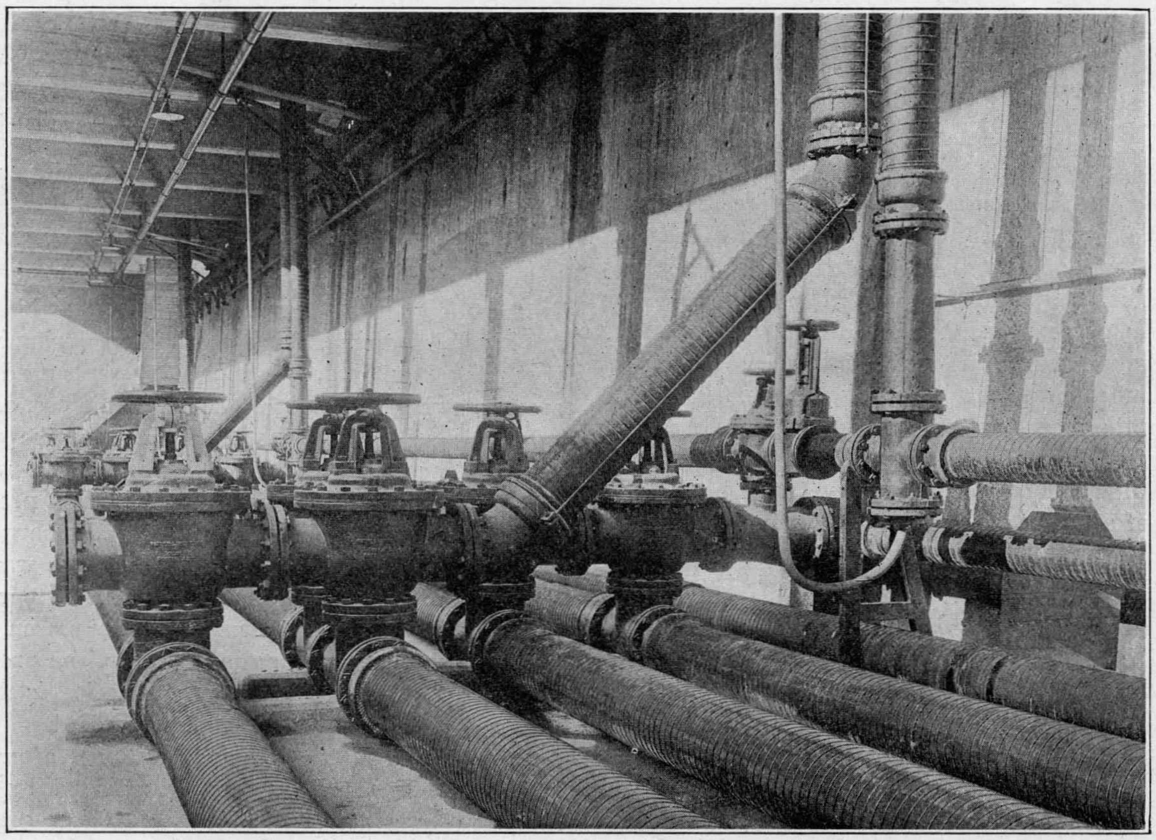
Ascending the stairway to the top level of the vats, I watched the solution (1.3% ferric, 0.3% ferrous iron, with 2% copper, and 2% free acid) being circulated by an air-lift after withdrawal from the bottom of the same vat, to induce circulation. The yellowish green liquor was covered with a creamy foam, except where clear patches appeared indigo-blue, owing to the reflection of a perfect sky. As one proceeds to the end of the cycle, the solution circulating in each successive vat gradually changes in color from a greenish-straw to a greenish-blue color until finally the solution circulating on the freshest (or newest) ore is of a fairly clear blue color.

The explanation of this variation in coloring is that the percentage of ferric phosphate present gradually decreases, owing to its precipitation in the ore, as the free sulphuric acid present is neutralized in the cyclic advance. The ore contains 0.2% phosphorus, which evidently first goes into solution, and is then precipitated, the greater portion being eventually discharged in the tailing. Until the presence of phosphorus was noticed, it was thought the precipitate of cement copper was contaminated with basic ferric sulphate by reason of atmospheric oxidation occurring in the scrap-iron precipitation launders. Apparently this contamination is due to the presence of basic ferric phosphate instead.

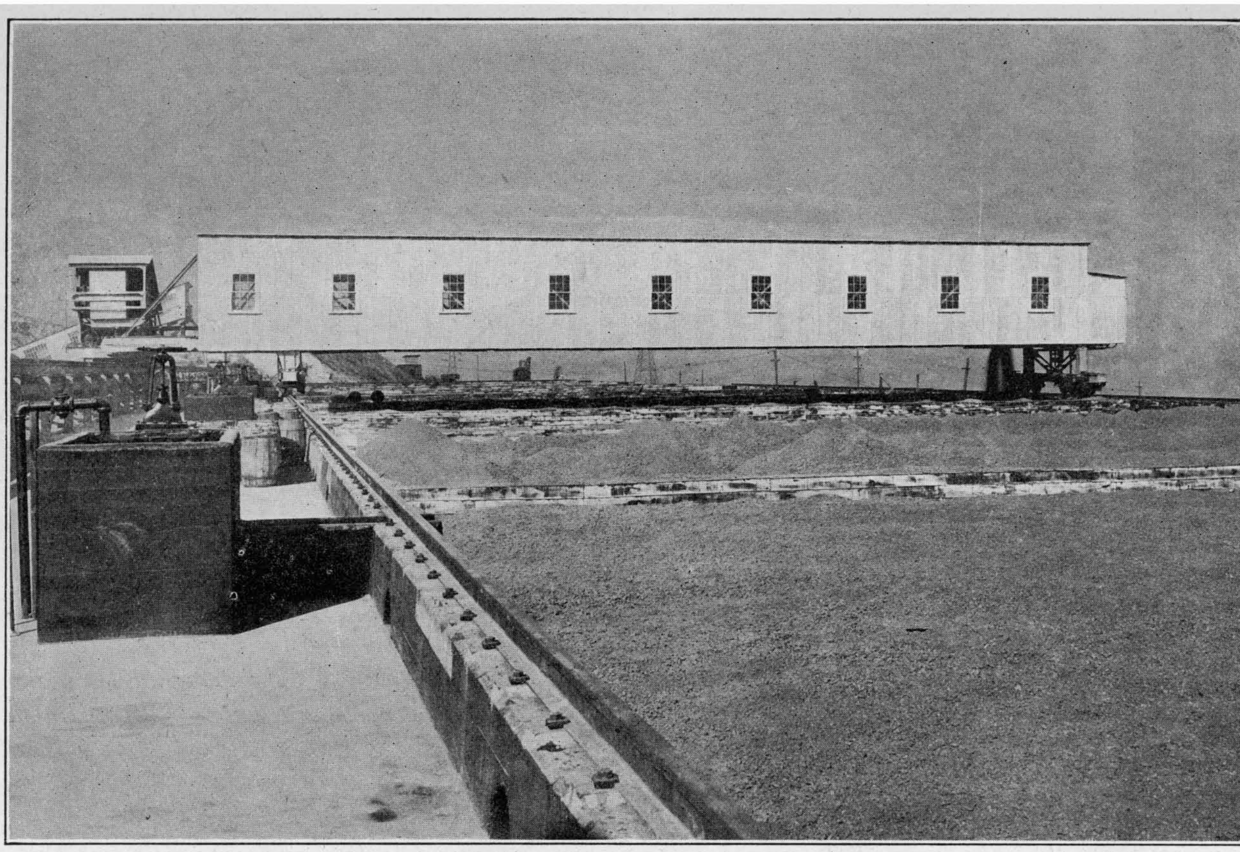
From No. 5 vat the residue (tailing), after leaching had been completed, was being removed by a Mead-Morrison excavator, which has a 10-ton grab-bucket, enabling 4000 tons to be removed in an eight-hour shift. The grab takes the tailing from the floor cleanly, planing



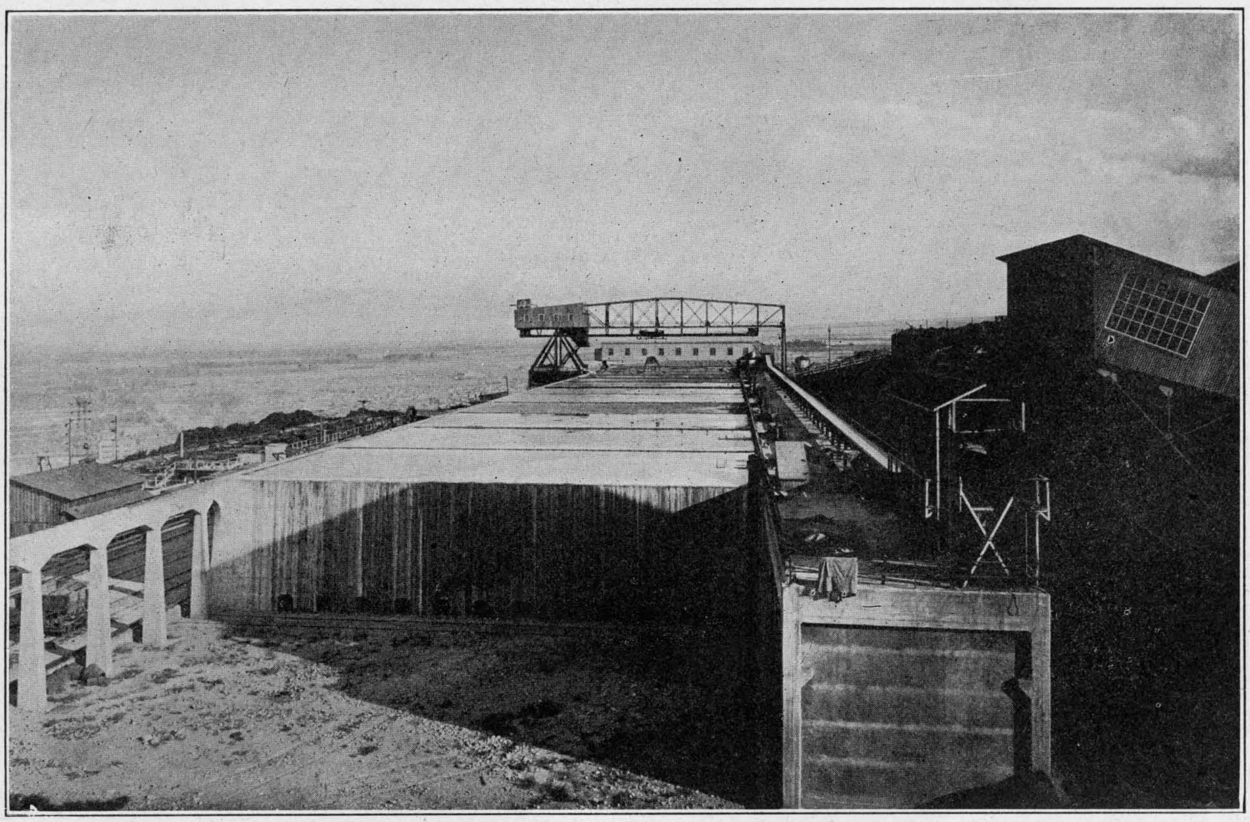
ORE-BINS, DUMP-TRACK, AND COARSE GRIZZLIES



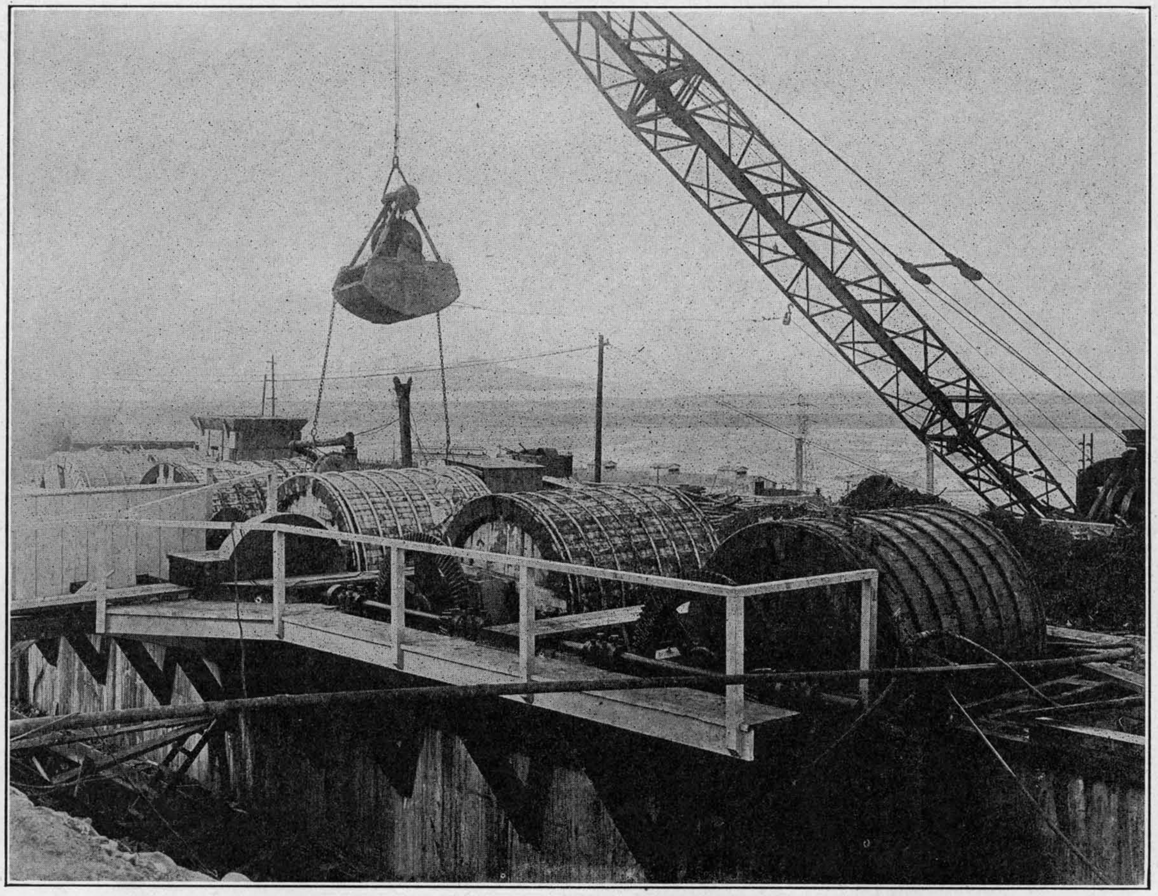
ARRANGEMENT OF VALVES AND REDWOOD-STAVE PIPE LEADING TO LEACHING-VATS



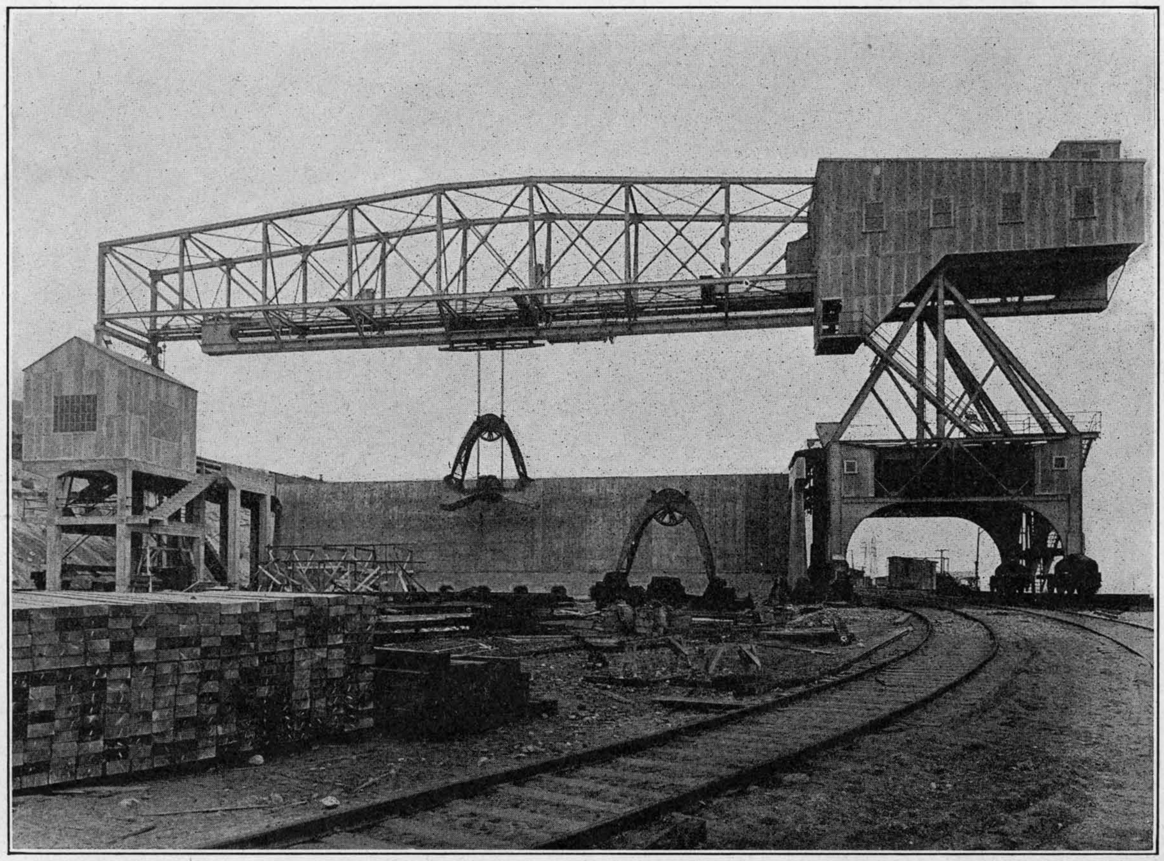
THE LEACHING-VATS, WITH THE LOADING-BRIDGE



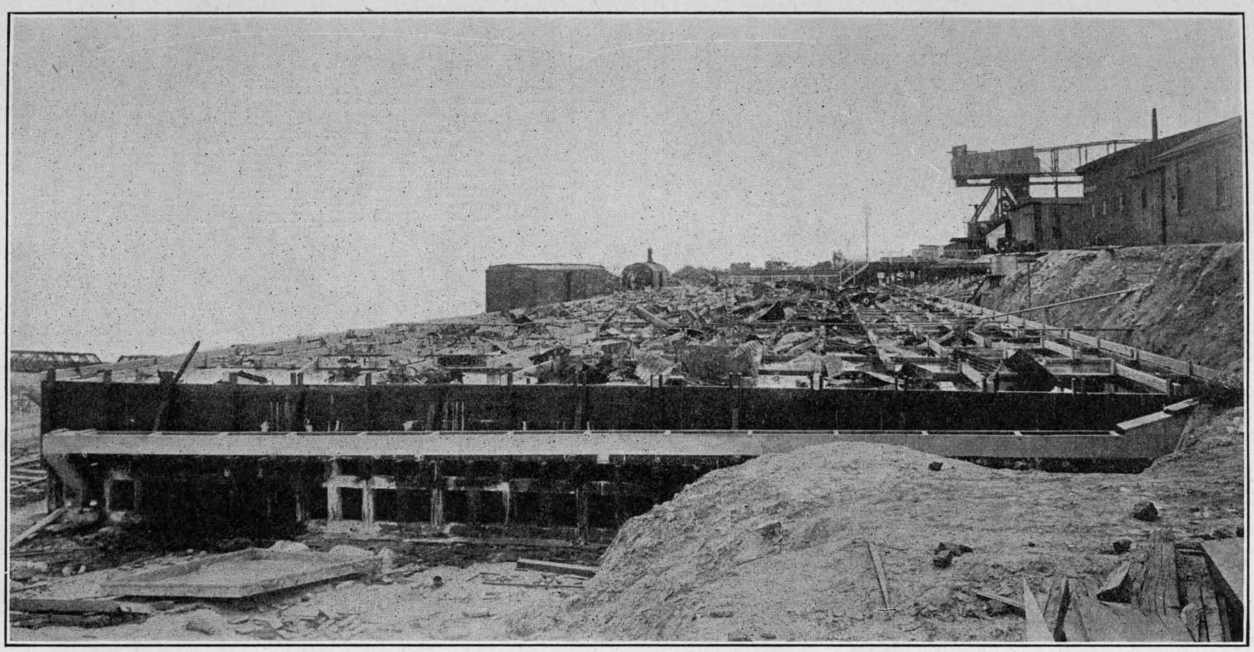
THE LEACHING-VATS, WITH THE LOADING AND UNLOADING BRIDGE IN THE BACKGROUND



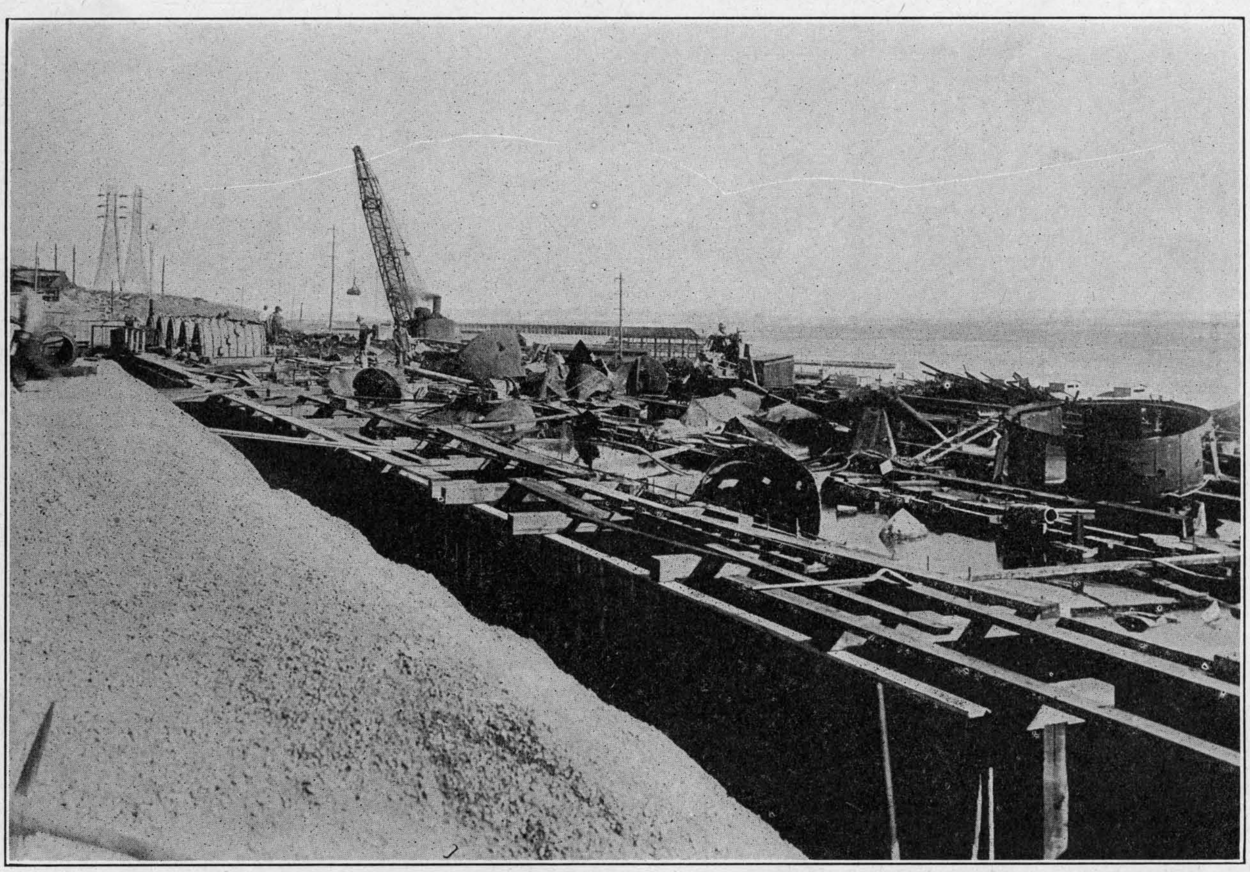
THE PRECIPITATION DRUMS



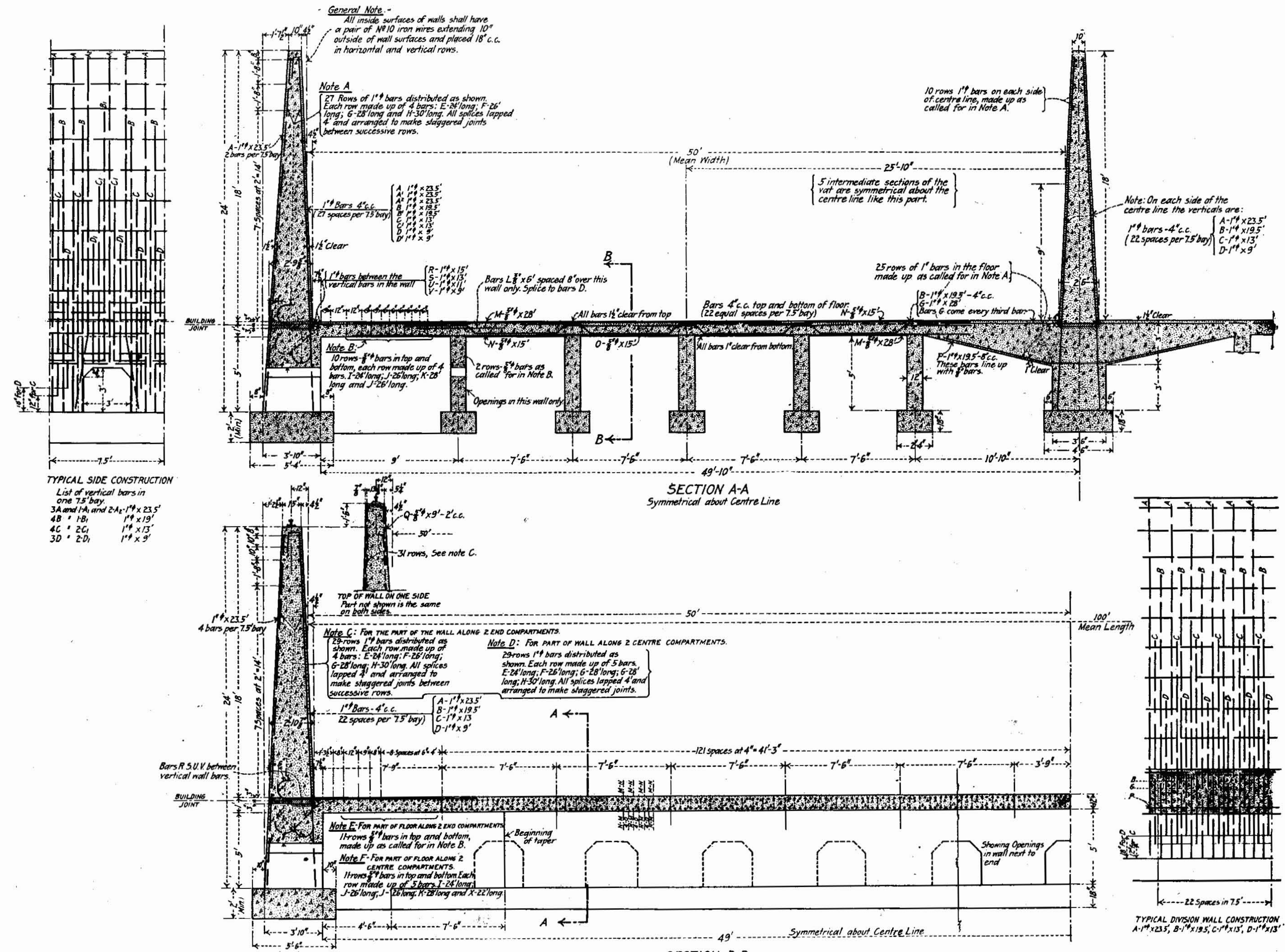
UNLOADING-BRIDGE AT THE SOUTH END OF LEACHING-VAT



THE WOODEN LAUNDERS FOR PRECIPITATING COPPER UPON IRON SCRAP



THE CONCRETE LAUNDERS, SHOWING THE IRON SCRAP ON WHICH THE COPPER IS PRECIPITATED



SECTIONS SHOWING THE CONSTRUCTION OF THE LEACHING-VATS any boards that rise above the general level. This excavator discharges into a hopper having a motor-controlled gate that empties into 45-ton Clark air-dump cars. The tailing shows only a trace of soluble copper, the greater loss being in undissolved sulphide. It would not be fair to quote the exact assay, because operations are still in the experimental stage.

Acid is obtained from the Garfield Chemical Co., five miles distant and adjoining the Garfield Smelting Co.'s (A. S. & R. Co.) plant. Fresh acid is added in the middle of the leaching advance. Very little (0.06 to 0.09%) ferric sulphate passes from the precipitation vats. All of this solution is wasted. The specific gravity and assay of the solution in cycle is as follows:

	Specific gravity	Free acid %	Copper %	Iron — Ferrous %	Iron — Ferric %
To precipitation	1.1390	0.24	3.16	0.37	0.95
In centre of cycle	1.1880	1.24	2.58	0.20	0.75
At end of cycle	1.1100	1.23	1.83	0.18	0.68
Last wash-water before tailing	.....	...	0.089	...	...

The copper-bearing solution flows through wooden-stave pipes of 10 in. diameter to the precipitation vats. The first series consists of four rows of four mastic-lined vats, each 9½ by 62½ ft. and 6 ft. deep, made of reinforced concrete, the walls being 6 to 10 in. thick, with a suitable batter. The flow of the solution continues up and down the total length of 1000 ft. In the second series likewise, the flow is continuous through 27 wooden vats, 9 by 64 ft. each, and 3 ft. 7 in. deep, a total length of 1728 feet.

In the first vat (62½ by 9½ ft.) of the first series are five barrels, 7 ft. 4 in. diam. and 8 ft. long, which are turned at the rate of two revolutions per minute. One of these is made of maple and four are made of fir. The wooden staves (4 by 8 in.) of the barrels are perforated with ⅝-in. holes placed 8 in. apart, permitting the solution to enter. The barrels are loaded with tin-cans and other small scrap-iron, which, as it becomes corroded by the solution and replaced by the copper, disintegrates. When small enough, the particles of iron, with the cement copper, escape from the barrel through the perforations into the vat. These products of the process are discharged by pulling a wooden plug in the bottom of the vat, from which the thin mud of cement copper and iron runs over a screen of ½ in. openings by which the coarser particles of undissolved iron are held. The same screen is used to clean up the residue from the other vats, and it serves to catch any leaf or plate copper that might interfere with the accurate sampling of the 'cement', or copper mud. All the vats, except the first, are filled with scrap, the heavier pieces being placed in the concrete vats and the lighter ones in the wooden vats. When cleaning up, the solution is withdrawn through a hole in the bottom, and a hose is played on the scrap so that the water removes the cement, which runs into one of the four settling-vats covered with a screen. The wash-water is decanted and the copper precipitate is removed by a clam-shell excavator into cars that transport it to the drying-vats. The decanted water flows into another vat, 75 ft. diam. and 12 ft. deep, where any remaining particles of copper settle. To this vat comes the waste solution from the entire series of vats, in order to arrest any copper that otherwise would be lost.

At the south end of the leaching-vats are three shallow wooden vats, 65 by 70 ft., full of red clay apparently. That is the cement copper undergoing atmospheric drying, or evaporation of its excess water. The method seems ill-adapted to winter or to the rainy season. The yellowish green slime of basic ferric phosphate discolors the copper. The solution coming to the precipitation vats from the leaching-vats is vividly bluish-green, the color of copper sulphate mixed with ferrous and ferric sulphates.

The precipitation plant looks like a junk-store in chemical liquidation. One regrets that the scrap cannot be macerated or otherwise reduced to small and nearly uniform particles so as to expose a large surface to corrosion and precipitation. Sponge-iron is being tried for this purpose at Ajo successfully, but the use of it depends upon the cost of production.

The accompanying photographs illustrate the various stages of the metallurgic operation.

THE SMELTING OF THE CONCENTRATE

The products of the Arthur mill, as we have seen, consist of two kinds of concentrate, one made by water and the other made by flotation. These concentrates are mixed before being delivered to the filter-plant, so that the final output of the mill is a product containing, on average, during 1917 for example, 14.82% copper, 26.79% iron, and 21.69% insoluble, together with 13.6% moisture. The last item is of greatest importance to the smelter, as we shall see. Formerly the table concentrate and the flotation concentrate were shipped separately, the first containing about 8% and the second from 20 to 25% moisture. They used to be loaded into the cars together, not properly mixed, the consequence being that accurate sampling was rendered impracticable, for the flotation concentrate was much richer in copper, containing 20% as against 12% in the table concentrate. So it was decided to ship these products separately, in order to facilitate sampling, but the cars containing the flotation concentrate had to be left on a siding for a week or more in order to allow the water to drain. Finally the management at the mill devised the scheme of mixing the table concentrate with the flotation concentrate in the Dorr thickener before filtering, thereby yielding a product containing 16 to 17% moisture. This was material that, it was thought, could be sampled fairly well.

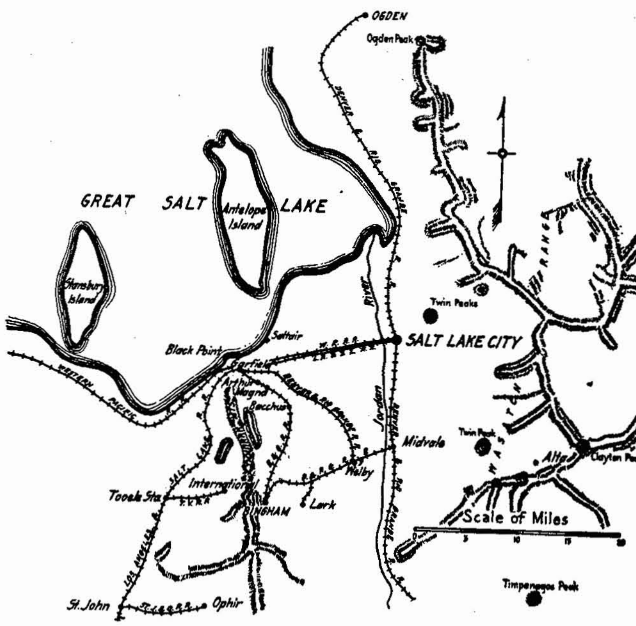
However, before discussing this tender subject further, let us go to the Garfield smelter, to which the concentrate from the Arthur mill is consigned under contract. This smelter belongs to the American Smelting & Refining Co., it is only two miles from the Arthur mill, and stands on the shore of Salt Lake, near Black Rock point. The Garfield smelter treats only custom ores. At the present time 60% of the gross tonnage of ores comes from the Utah Copper company. Only a small proportion of precious-metal ore, averaging 10 oz. silver, is included in the receipts, the richest containing 32 oz. silver per ton and being used in the converter owing to its high content of silica. The 24.6% insoluble in the flotation concentrate from the Utah Copper contains 6.7% alumina and 16.9% silica.

The manager of the plant, R. F. McElvenny, was most courteous, and A. H. Richards, the assistant-superintendent, was kind enough to conduct me through the works.

A car of flotation concentrate from the Ohio Copper company was being unloaded. It contained about 25% moisture, and while watching the shoveling one could realize what trouble the Utah Copper product used to give to the smelter people. The flotation concentrate arrives in flat-bottom cars having a floor of wooden slats on which cocoa matting is spread so as to serve as a filter through which the water can drain. The car arrives with a pool of water on top; in winter this takes the form of a cake of ice two or three inches thick. From November to March it is customary to run the cars into a thawing-house. This is a brick building provided with four tracks giving room for 12 cars apiece, so that 48 cars can be thawed concurrently. In this building is a pit covered with cast-iron (one inch thick), under which is a big brick flue through which hot air passes from a furnace fired with coal. A temperature of 110° is maintained. Usually 24 hours suffices to thaw the concentrate, but if it is frozen solid, the period of thawing may be lengthened to two whole days.

The unloading is performed under contract with gangs of Greeks, also a few Japanese. These men expect to earn from $6 to $7 per day, for they work hard. The price for unloading concentrates from the Utah Copper mills varies with the moisture; for table-concentrate it is $9.60; for mixed table and flotation concentrates it is $11.50; and for straight flotation concentrate, $13.35 per railroad-car. The price of $11.50 for the mixed product was maintained only during a short period, because this material was so sticky, the unloaders insisting that it was as bad as the unmixed flotation concentrate, so that recently $13.35 has been paid for unloading the mixed concentrates. Since my visit a general rise in wages has lifted the contract prices to $10.60, $12.50, and $14.35 for the three classes of material; and, as a matter of fact, it has been found necessary to pay $14.35 on all the concentrates. A car holds 70 tons and the crew that does the shoveling consists of 8 men. The price is determined by agreement before the job is tackled. In order to ascertain the proportion of moisture, holes are dug into the concentrate at two points, each one-third of the distance between the centre and the ends of the car. Both holes are dug right down to the bottom of the car and a sample is taken from each hole by means of a spoon or scoop, which is an inch wide, three inches long, and is provided with vertical sides that give a depth of one inch. In taking this moisture-sample a uniform slot is cut on the face of the hole from the top of the load of concentrate to the very bottom of the car. While visiting the unloading-shed, above the receiving-bins, I found that a gang of shovelers had struck, demanding $13.35 for unloading a car of Utah Copper concentrate that happened to be unusually wet. It was, they claimed, a load of flotation concentrate unmixed with table concentrate. These men knew the difference. They stopped work, but the difficulty was adjusted later.

Every tenth shovel from the railroad-car is thrown onto a platform where it is half-shoveled twice, one quarter being shoveled into a car that delivers it at the sampling-room. From a small bin it is drawn onto a steel floor, upon which it is reduced by successive quartering to a sample of 200 pounds. The operator throws every other shovelful into a wheelbarrow in order to decrease the labor involved, the stickiness of the concentrate rendering it impracticable to make a regular cone. He simply shovels toward the centre, makes a heap, and then flattens the mass, using an iron sector to mark the quarters.



MAP OF PART OF UTAH, SHOWING THE POSITION OF GARFIELD, ALSO THE MAGNA AND ARTHUR MILLS

From the receiving-bins the ore is conveyed on a belt to the bedding-bins, whence it is discharged into lorries. Each of the four bins is 20 ft. wide and holds 7200 tons, occupying a building that is 80 ft. wide by 540 ft. long. The discharge is done by using a pick to move the boards on the bottom of the bin, which are made hopper-bottom by allowing the ore to collect at the sides. Five men are employed to empty the bin and load the lorry.

A 10% addition of silicious oxidized ore, containing 50% SiO_2, is added to the concentrate in these bins. Lime, in the proportion of 10%, is added on the fifth hearth of the roasting-furnace, so as to obtain an intimate mixture. The lime used at this smelter is in the form of sand from the shore of the big lake, a deposit of it having been discovered and first used by W. H. Howard in 1909. It contains 48% CaO and 9% insoluble.

In approaching the roaster I heard a noise that furnished further evidence of the difficulties made by the millman for the smelter. Three men were engaged in hammering the lorry, one poking with a crowbar from above, while two were hitting the sides with a special implement, namely a bar having a knob at each end. Thus even in summer the stickiness of the concentrate gives this further trouble. In winter the difficulty of unloading the electric lorry is increased, of course. It appeared a pathetic performance in this age of proud efficiency, but I confess that by this time my sympathy had veered to the smelter-people, who, it seemed to me, were called upon to overcome difficulties that had been passed to them from the mill. Mr. Richards informed me that his operations were curtailed frequently, especially during winter, by the inability to get the concentrate out of the railroad-cars and then by the difficulty of feeding it to the roasters; even after it has been mixed with 10% of silicious ore, it is difficult to ensure a uniform feed to the furnace. This is true of operations during the winter, but in summer also there is trouble in feeding concentrate to the roasters. In a recent letter from Mr. Richards he says: "I was over to the roasters this morning and found that the product we are now receiving is 'bridging' in the hoppers of the Call-Wagstaff furnaces. You will recall that this set of roasters has the apron-feeder underneath. I counted no less than 12 men poking the charge out of the hoppers, so as to get it into the roasters, and even at that our reverberatories were curtailed owing to not being able to obtain sufficient calcine." No metallurgist can refuse his sympathy to a superintendent compelled to roast undried flotation concentrate.

New cars are to replace the lorries now in use; three of them will constitute the train to be pulled by the electric locomotive; they will carry 20 tons divided in two compartments, having vertical sides, so that when the bottom drops the load will fall promptly.

Of the various feeders, the Stevens-Adamson apron appears to work best on wet material. This feeder has a storage-hopper, two sides of which are vertical, whereas the other two slope at 70°. A 40-in. belt of overlapping corrugated steel slats 4 in. wide moves the ore, the flow of which is regulated by a gate, giving an opening 12 in. high and 36 in. wide. I saw what an inefficient feeder, of another type, will do. The concentrate stuck so as to bridge the opening and stop the feeding.

The roasters at Garfield are of the McDougall type, with rabbles having straight blades set at 45°, except on the top hearth, where the angle is 60°. The blades have a 3-in. sweep and are 6 in. apart. In order to prevent clogging by the flotation concentrate, it is proposed to widen the blades from 7 in. to 11 in. on the topmost hearth, where four arms with rabbles spaced 9 in. apart are in use. The roasting-plant contains 48 furnaces, divided as follows:

10 Herreshoff furnaces, with seven hearths of 19½ ft. diameter.

8 Call-Wagstaff furnaces, with eight hearths, also 19½ ft. diameter. This furnace was designed by R. A. Wagstaff and B. G. Call, members of the staff.

16 McDougall furnaces, of six hearths, 18 ft. diameter.

14 McDougalls, six hearths, of 19½ ft. diameter.

The cars that take the roasted ore to the reverberatory furnaces are not covered. Hereafter, while being loaded they will come under a hood that has an opening, with a slight suction into a flue, for the collection of dust. This arrangement is not yet in use, but it has proved successful at the Tacoma smelter. Hoods have been built above eight new roasters and provision made to connect these hoods with the flue, but this improvement is being delayed by the need of covered calcine-cars. It is intended to cover the cars and to double their size; also to provide a tight connection between the hopper of the reverberatory and the car. All such betterments are delayed by the inability to obtain quick delivery of material. In these matters of dust-control the International smelter at Miami is so much better equipped that I venture to express surprise at the fact that the experience obtained there had not been utilized at Garfield. In my article describing the International smelter I gave details and also drawings of the appliances there in use.[1]

The feed to the roasters contains 14 to 16% copper, with 29 to 30% sulphur. After roasting it contains 16 to 18% copper, with 10 to 12% sulphur, having lost 12% moisture. The roasted ore is smelted in reverberatory furnaces, fired with pulverized coal, yielding a matte containing 42% copper and 25% sulphur. About 10% of the sulphur is eliminated in the reverberatory. The matte is treated in converters that yield blister copper 98% fine. This goes to the refinery at Baltimore, Maryland.

A question arose among our party as to the proportion of power used in the various stages of treatment to which the Utah Copper is subjected. It was ascertained that the refinery produced 170 lb. copper, the smelter 80 lb., and the concentrator 34 lb. per kw-day.

Returning to the question of moisture in the concentrate, it used to be the practice at Garfield to unload it upon belt-conveyors and bed it directly with the crude ore going to the blast-furnaces. At times as much as 10% of the blast-furnace charge was flotation concentrate, but the loss in dust was excessive. At the time of my visit shipments of flotation concentrate were being received from the Engels Copper Mining Co. in California. In order to minimize the cost of transport, this company reduces the moisture as much as is practicable, bringing it down to about 9%, which makes it slightly dusty. Apparently a 10% moisture is low enough for operations at the smelter.

On enquiry I was informed by Mr. Elmer E. Paxton, manager for the Engels company, that two Oliver filters are in use in his mill and that the dewatering of the concentrate is facilitated by a liberal use of steam, in three ways: (1) Steam is introduced through the emergency pipes, the air being excluded occasionally while the steam is admitted to the lifts. (2) Steam is forced through the feed-pipe to the bottom of the settling-tank, so as to warm, but not agitate, the concentrate just before a charge is drawn from the settling-tank into the filter-tank. While one charge is filtering, the next is being warmed. (3) A strong jet of steam is applied, through a pipe and hose at the back of the filter, so as to loosen

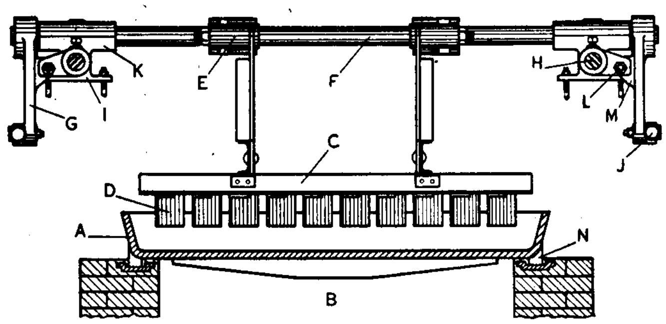


FIG. 1

and warm the bank of concentrate that forms in the bottom of the filter-tank. Thus the moisture in a two-inch cake is reduced as low as 10%. The average moisture in the concentrate shipped during May was 11%, this being reduced, by evaporation, to 9.5% by the time the concentrate reached Garfield.

The Ohio Copper company, at Bingham, has adopted the Lowden dryer. This machine is manufactured by the Colorado Iron Works, at Denver. It is designed to treat

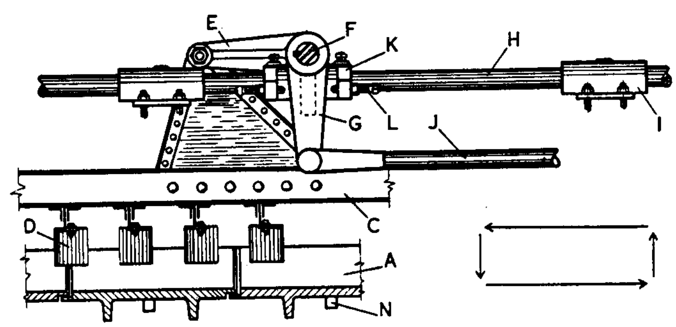


FIG. 2

plastic material, such as flotation concentrate, without excessive loss in dust. The dryer consists (See Fig. 1, 2, and 3) of a single hearth made of cast-iron plates *A*, beneath which is the flue *B*, through which the gases of combustion pass. The feed is moved forward by means of rakes *C*, bearing rabbles *D*. These are right and left hand, in order to impart a ploughing action and distribute the material over the hearth. After the forward stroke, the rakes lift and clear the top of the material for the return stroke. The rakes move close to the hearth, preventing any caking, which would hinder the transmission of heat. The 'balling' of the plastic concentrate is minimized by the fact that the length of the stroke is less than the distance between the transverse rows of rabbles, so that, in descending, they enter the material where it has been left in ridges by the preceding stroke.

[1]See M. & S. P., December 1, 1917.

Any lumps are broken by a roller placed between two rows of rabbles at the point where such lumps are in the most friable condition. Ample provision is made for expansion and contraction wherever parts are exposed to heat. The following data show the results obtained in practice at three plants:

Company	Moisture in feed	Moisture in product	Coal per ton of dry solid
	%	%	lb.
Federal Lead	15	5	117
St. Joseph Lead.........	16	5	125
Vindicator Consolidated..	29	13	128

In each case the temperature of the feed was 55°, and that of the product 212° F., or the boiling-point of water. The concentrate from the Ohio Copper mill contains 12% moisture, with which the superintendent of the Garfield smelter is quite satisfied. On the other hand, Mr. Janney tells me that the moisture in his concentrate during April averaged 12.83%, in 425 cars containing 30,898 tons, each car carrying 72.7 tons wet or 63.36 tons dry. Apparently the stickiness of the Utah Copper product is due to the large proportion of insoluble matter in the concentrate produced by the pneumatic flotation-cells.

This problem of drying flotation concentrate, so as to deliver a product suitable for unloading, conveying, and sampling at smelters, must be faced. So far, whether at Miami, at Ray, or at Garfield, the mill-people have 'passed the buck' to the smelter-people, who are tied by contracts that were signed before the true character of flotation concentrate was appreciated. I find that as between the managers of mills and smelters, the former always deliver a product containing less moisture than the same product contains on arrival at the smelter, despite such drainage and evaporation as may ensue in course of transit. This is a discrepancy that may be due more to a human factor than to one that is technical. It would be to the advantage of all concerned if the concentrate could be turned out in a form convenient to handle. Meanwhile the experiment being made with the dryer at the Ohio Copper is most interesting; there the dryer is placed at the discharge from the filter, as it should be, so that the concentrate drops from the filter upon a hearth of the same width as the filter. I may add that a rake-conveyor was used to cool and convey hot material from the Argall roaster at the Portland mill near Colorado Springs, in 1902.

Would nodulizing be advantageous? That would be the next step after drying. Nodules of, say, half-inch size, would be most convenient to handle and would give a feed suitable for the reverberatory or even the blast-furnace. At Braden the flotation concentrate is adapted for the blast-furnace by nodulizing in rotating kilns,[2] heated by oil-burners to a temperature of 1750°F. The sandy concentrate, containing 17% silica, is heated to a sticky consistence, due partly to the combustion of the part of its sulphur content, so that the rolling motion causes it to ball into nodules. The kilns are sloped an inch per foot toward the discharge, from which the red-hot nodules drop upon an endless chain of cast-iron pans, which convey the product to hoppers that feed the blast-furnaces. In the course of this process the sulphur is reduced from 28% to 18%, and, of course, the moisture has been evaporated.

Nodulizing has been tried also by the Canada Copper Corporation at Princeton, British Columbia, by R. M. Draper.[3] A rotary kiln borrowed from a cement plant was used. The kiln was 7 ft. wide and 125 ft. long, or about 25 ft. longer than was required for the nodulizing of the copper concentrate. About 82% of the sulphur was retained in the nodules, this being important in a region where the sulphur is required for smelting. Pulverized coal from a neighboring mine was the fuel used, the kiln itself being operated by electric power. It was estimated that a 100-ft. kiln of this type would nodulize 100 to 125 tons of flotation concentrate in 24 hours. At Chrome, New Jersey, a similar experiment was made, by Mr. Draper, on flotation concentrate from the El Cobre mine, in Cuba. The consumption of oil, as fuel, was 6 to 7 gallons per ton of concentrate. The nodules proved satisfactory for blast-furnace smelting when charged cold, but there was trouble with crusts on the furnace when too much of the hot nodules was introduced, owing probably to the fact that "they were near the smelting-point when charged into the furnace and naturally smelted much higher up in the furnace." I quote Mr. Draper. The concentrate averaged 30% sulphur, whereas the nodules averaged 14%. One sample of nodules chilled immediately in water had a sulphur content of 20%. A screen-test of the nodules showed that all the product was under half-inch, but only 4.78% was finer than 16-mesh.

Desulphurization would accompany nodulization. A product containing 10 to 11% sulphur and about 16% copper would be most suitable for making a 40% matte. Unless desulphurized sufficiently the delivery of nodulized concentrate at the smelter would conduce to a low temperature in the roaster, so that the smelter-people would not regard this step as an advantage. In short, if any desulphurization is to be done at the mill, the operation must be completed to the point of rendering unnecessary any further roasting at the smelter itself. Moreover, the smelter company, in making a rate for treatment, includes a charge for expelling both the water and the sulphur in custom ores; it would lose this source of profit if the preliminary operations were performed at the mill. On a long haul, like that of the Engels product, the removal of both moisture and sulphur would effect a considerable economy. Thus there are various conditions, technical and commercial, to be considered.

One would expect to find the Cottrell precipitator attached to the roaster, but at Garfield this precaution

[2]M. & S. P., February 12, 1916.

[3]'Nodulizing Flotation Concentrates'. 'The Canadian Mining Journal', August 1, 1918.

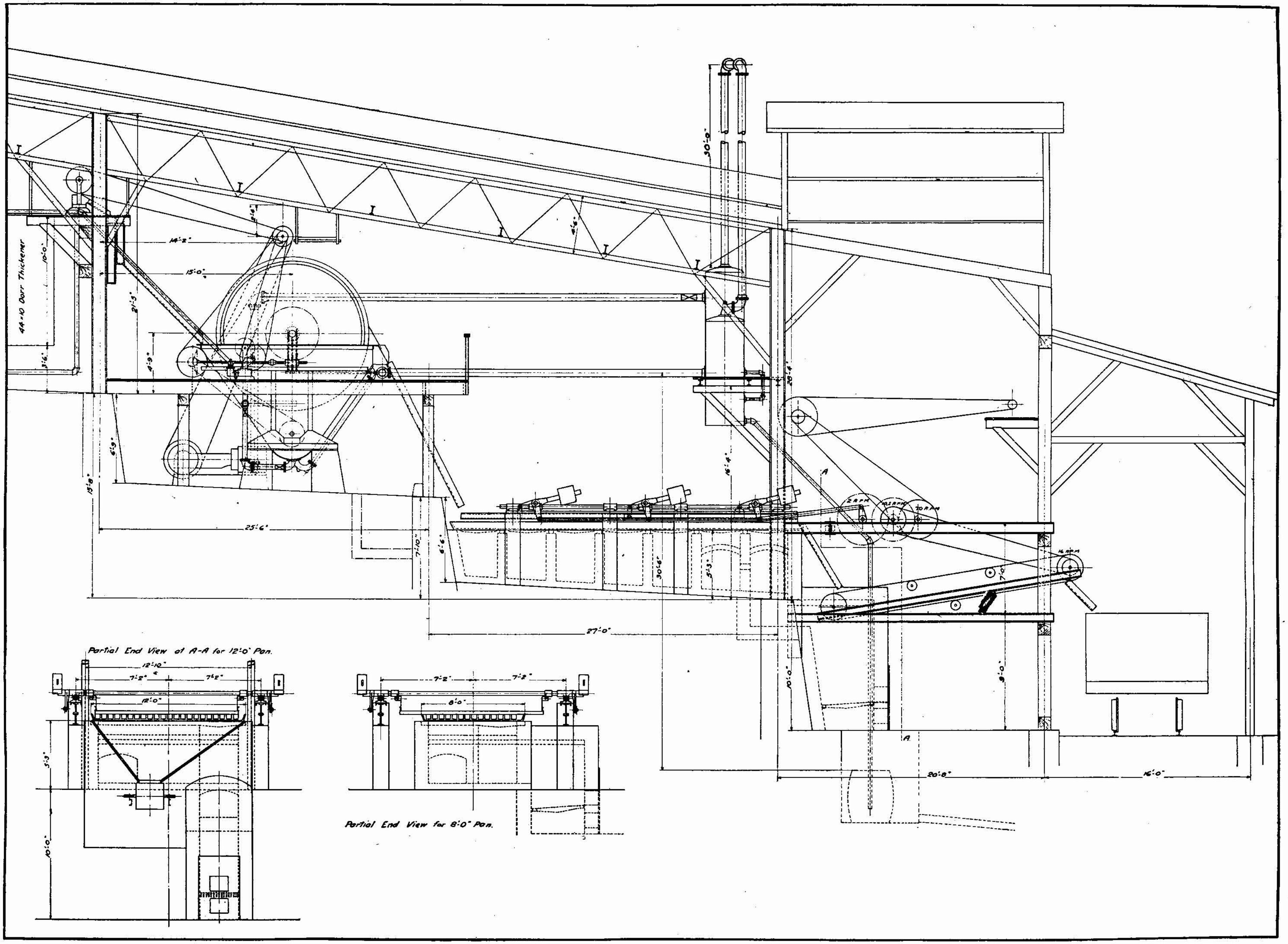


Fig. 3. THE LOWDEN DRYER, AS ARRANGED FOR THE TREATMENT OF FLOTATION CONCENTRATE

is only applied to the converters. How great the loss of copper in dust from the dryers would be, for example at Miami, if the Cottrell tubes were not used, was shown to me by L. O. Howard, the superintendent, who had the electric current turned off for a minute so that I might see the effect, which was to change a nearly colorless smoke into one heavily laden with dust. At Miami the flotation concentrate is not desulphurized, only dried, in the roasting-furnace.

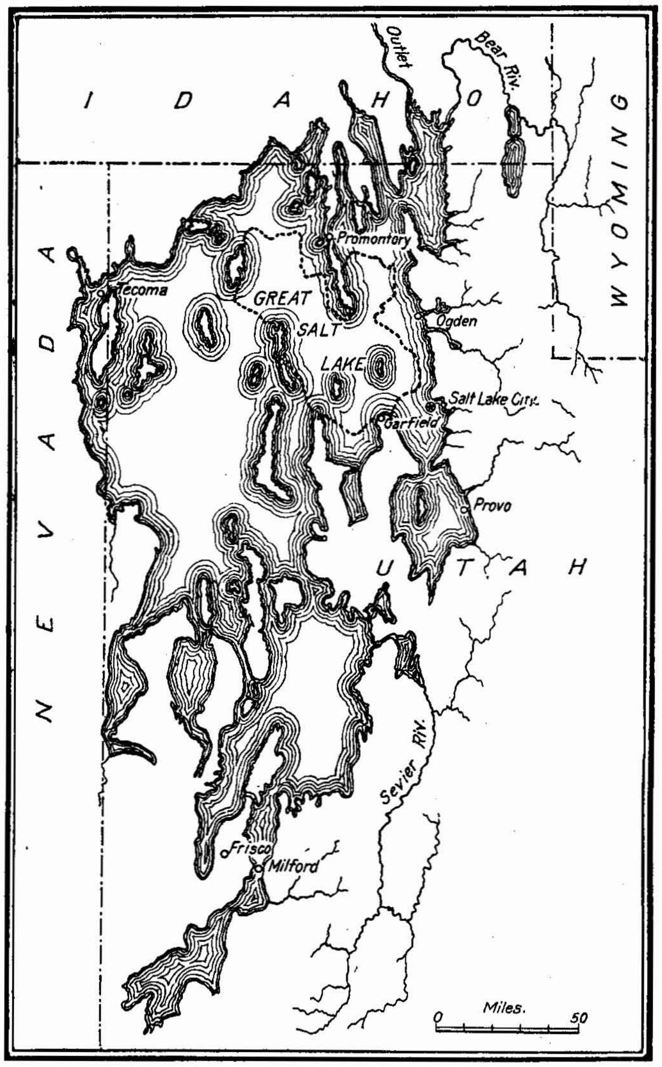
At the Nevada Consolidated smelter, at McGill, Nevada, the gases from the roaster contain such an excess of water, owing to the high proportion of moisture in the concentrate, that it tends to condense with the sulphur tri-oxide, forming sulphuric acid, which attacks the brick of the flues and the iron of the jack-arches. At Garfield such trouble is avoided by causing the roaster gases to join the reverberatory gases, thereby maintaining a high temperature. The withdrawal of too much heat from the stack affects the dispersion of the fume and is likely to cause the destruction of vegetation. The wise smelter-manager avoids trouble in that direction.

Evidently the technical staffs of the mill and smelter should co-operate to solve this problem of drying the concentrate. As the proportion of table concentrate decreases, and more slimy flotation concentrate is produced, this phase of flotation practice will increase in importance.

The smelter occupies a picturesque site. Looking northward, across the marshes, the pavilion and long trestle of Saltair can be seen projecting into the lake, and beyond them the mountainous mass of Antelope island, now remarkable for its herd of buffalo. Turning around to the overlooking Oquirrh range, the eye rests on the level contour of a bench of gravel and other parallel terraces, which suggest old shore-lines. Such indeed they are, harking back to a time humanly remote but geologically recent.

The greater Salt Lake of the olden time is known to geologists as Lake Bonneville, that being the name given to it by G. K. Gilbert,[4] who published a monograph on the subject in 1890, under the auspices of the U. S. Geological Survey. These former shores were noticed by the explorers of 1849 and 1850; they surmised the previous existence of a lake vastly larger than the present one. The drift-wood found on beaches five feet above the lake indicated the recent recession of the waters and suggested that a recession on a larger scale had caused a greater shrinkage of the lake in a former geologic period.

The Lake Bonneville of geology was named by Gilbert after Capt. B. L. E. Bonneville, an Army officer who explored this region in 1833 and made notes that were published, with a map, by Washington Irving, in 1842. Lake Bonneville flourished in Pleistocene time; it had a length of 346 miles, an extreme width of 154 miles, and a depth of 1050 feet; it covered 19,150 square miles, as against the 2170 square miles of the existing lake. For comparison, I may mention that Lake Superior covers 31,500 square miles and Lake Ontario 7250. The eastern edge of the old lake is now the granary of Utah. If the water were to rise to its former level, the Mormon temple in Salt Lake City would be submerged 850 ft. and Fort Russell would be 150 ft. below the surface. The area of

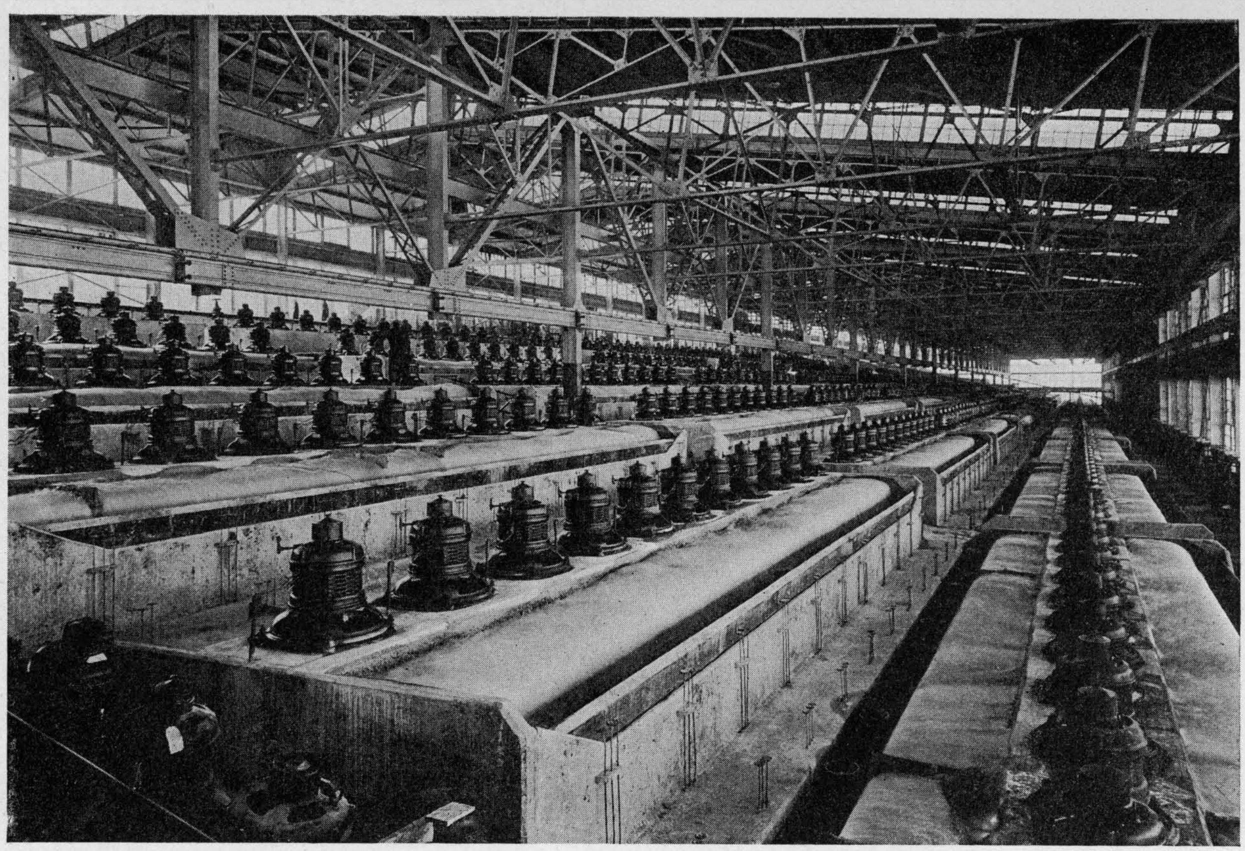


SHOWING THE EXTENT OF LAKE BONNEVILLE AS COMPARED WITH THAT OF THE EXISTING SALT LAKE

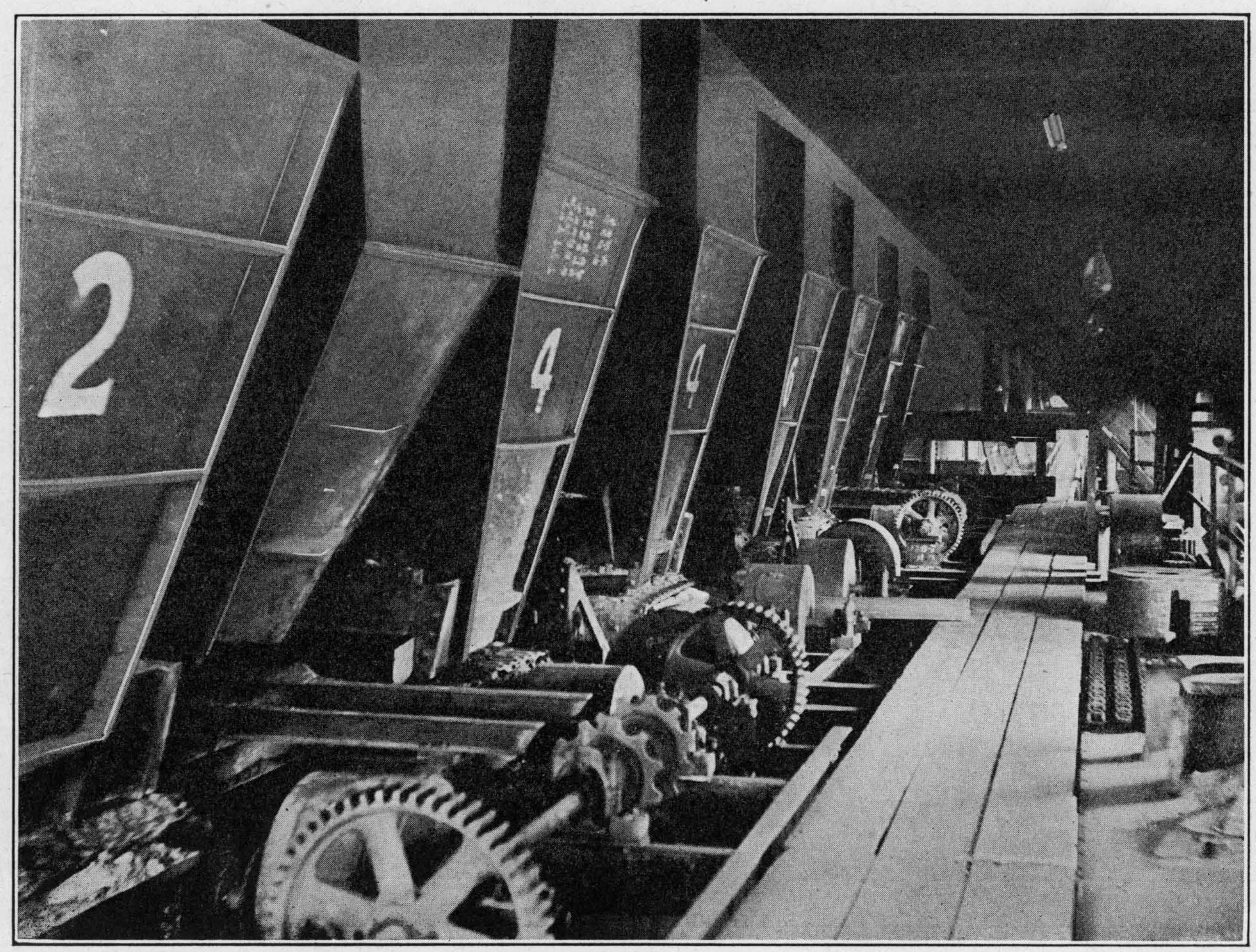
Lake Bonneville was diminished suddenly by an overflow that eroded an outlet through its rim at Red Rock pass at the north end of Cache valley. There the rim was a barrier of alluvium, instead of solid rock; the flood-water overflowed, a break was made, and a mighty river soon issued, causing a geologic debacle.

NOTES. It is interesting to note that the first contract with the American Smelting & Refining Co., for the

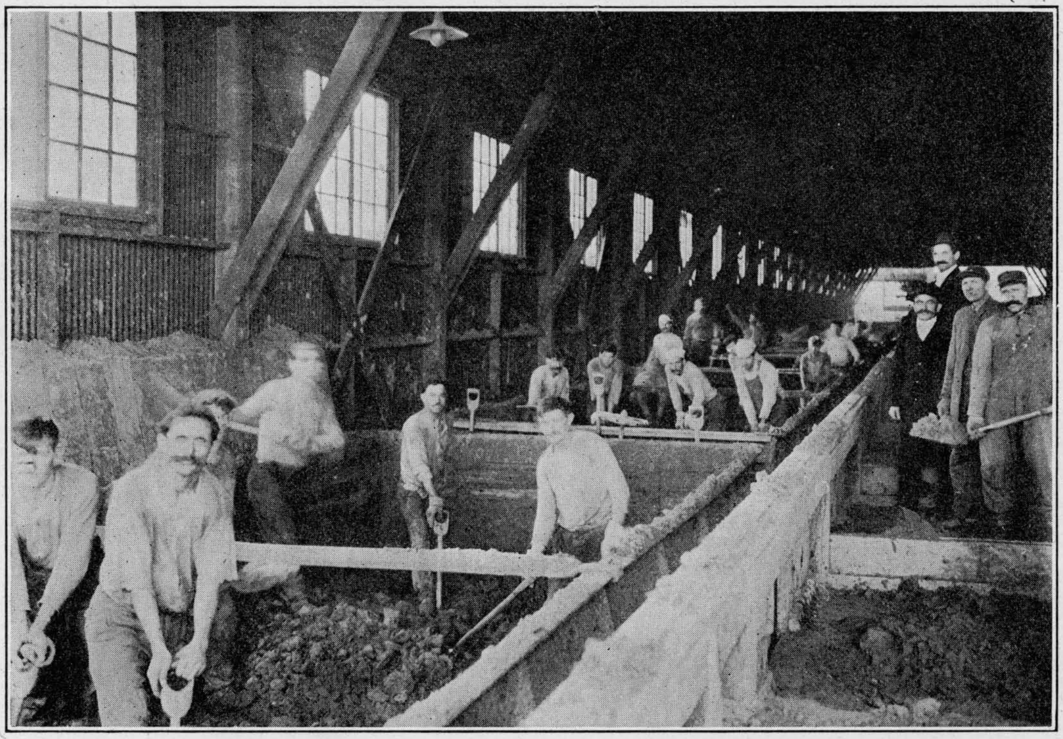
[4]He died, I am sorry to add, on May 1, 1918. In anticipation of his 75th birthday, on May 6, a number of his friends had arranged by correspondence to celebrate the event by a presentation. The 'Lake Bonneville' monograph is No. 1 in that series of U. S. Geological Survey publications, because No. 1 had been set aside for a treatise by Clarence King, who failed to write it, so 'Lake Bonneville', which was intended to be No. 9 or No. 10, was given the place of honor.



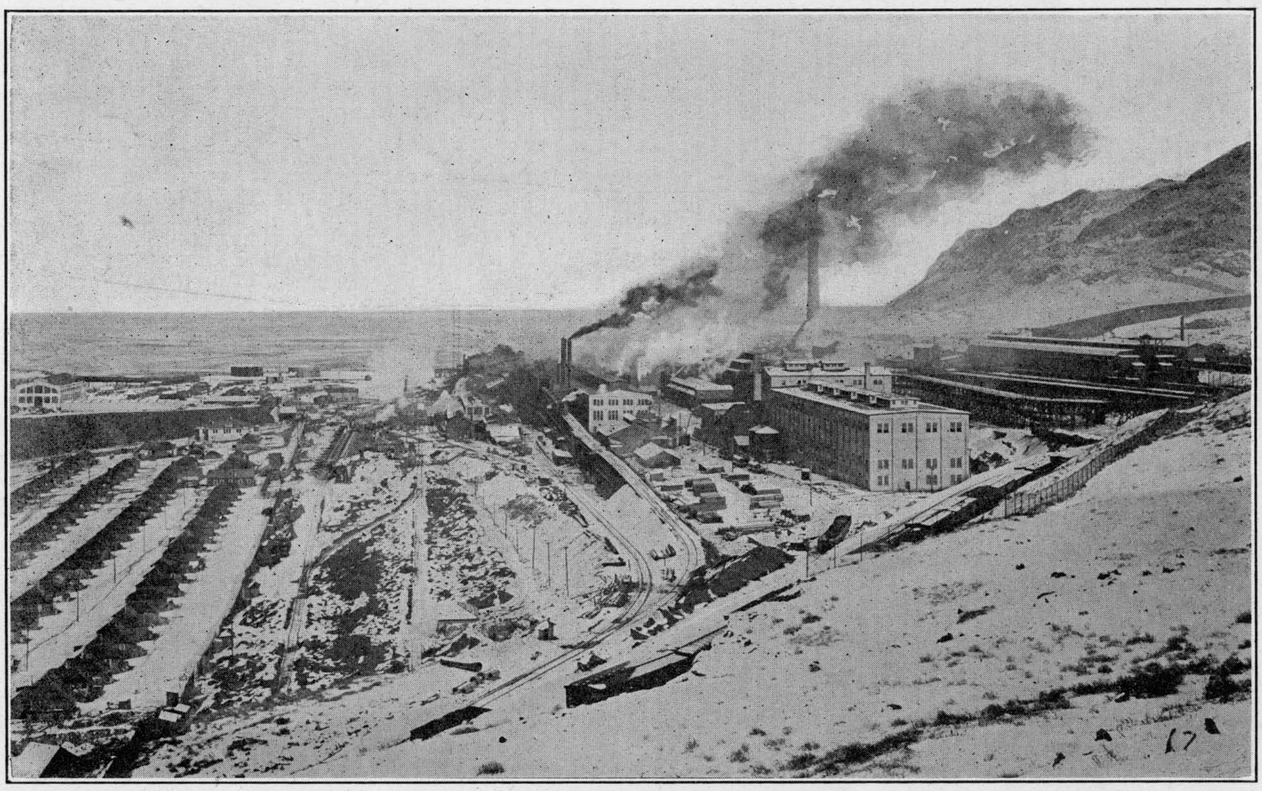
THE JANNEY FLOTATION MACHINES IN THE ARTHUR MILL



A BATTERY OF STEVENS-ADAMSON FEEDERS



UNLOADING FLOTATION CONCENTRATE AT THE GARFIELD SMELTER



THE GARFIELD SMELTER OF THE AMERICAN SMELTING & REFINING CO. AT GARFIELD, UTAH smelting of the concentrate, called for 95% of the copper contents, as determined by wet assay; for 90% of the silver at the New York quotation; and payment for all gold contents of 0.03 oz. or over at $20 per ounce (that is, 67 cents less than its market-value). The base charge for treatment was $6 per ton, a minimum of $5.50 and a maximum of $7.50 being stipulated. At least 10% of iron over insoluble was guaranteed by the Utah Copper Company, "such excess to be considered neutral, and an allowance of 10 cents per unit for each percentage of excess over the guarantee." All insoluble in excess of iron was charged for at the rate of 10c. per unit. The charges for refining the blister copper were $30 per ton, besides a 2% allowance for loss in refining of the silver and a deduction of 13 lb. per ton for similar loss in refining of the blister copper.

In 1917 the two mills treated 12,542,000 tons at a cost of 69.3 cents per ton. The Arthur mill treated 5,464,800 tons, or an average of 14,972 tons per diem. The average grade of the ore milled at both plants was 1.337% copper and the average recovery was 61.1%, equivalent to 16.33 pounds of copper per ton of ore. In 1916 the average recovery was 62.34%, the poorer showing made in 1917 being due to unsettled operating conditions, the various improvements under way, and the lower grade of the ore. In the company's annual report, from which I get these data, it is stated that "the average normal and metallurgically economical capacity of the two plants during the year was about 24,000 tons per day, whereas in order to maintain maximum production of copper a daily average of 34,362 tons was milled." The cost of milling in 1917 was abnormal, owing to the higher costs of labor and supplies and to the large charges made for State and Federal taxes, which amounted to $4,381,205. At the Magna mill the cost was 62.28c. and at the Arthur mill it was 78.40c., as compared with 35.35 and 40.94c., respectively, in 1916.

UNLOADING, CRUSHING AND SCREENING AT THE ARTHUR MILL

By FRANK G. JANNEY

DELIVERY OF ORE. The ore comes from the mine in trains of 40 cars, and at a point 15.3 miles from Bingham, or 2 miles above the Magna plant and 2.6 miles above the Arthur plant, the ore is diverted from the Bingham & Garfield main line to the Arthur high-line tracks. From here the train descends on a 0.4% grade over a 150-ton Strait scale, equipped with a Streeter-Amet automatic weighing and recording device, on which the ore is weighed in the train, while moving at the rate of two miles per hour. From the scale the train descends on an average 0.4% grade to the Arthur load-yard, which has a capacity of 102 cars. From here the loads are delivered by a switching crew to two dumper-load tracks, each having a capacity of 21 cars. These tracks are laid on a 1.5% descending grade, and the loads are fed one at a time by gravity to the car-dumped mule-pit. From the pit they are elevated 21 ft. up an 11.5% grade by means of an electric 'mule', and spotted on the car-dumper platen.

After the car has delivered its load it is bumped off the platen by the following load, and runs by gravity down a trestle track on a 5% grade, up a kick-back, and into either of the three empty-yard tracks, one of which is provided for cutting out all 'bad-order' cars. From the car-dumper empty-yard, the capacity of which is 90 cars, the empties are either switched to the general empty-yard, having a capacity of 136 cars, or picked up by the road-engines in making up their trains.

UNLOADING PLANT. The haulage and dumper mechanisms are electrically operated, and were designed, fabricated, and erected by the Wellman-Seaver-Morgan Co., in conjunction with the Utah Copper company's engineers, to handle the cars of ore delivered by the Bingham & Garfield railroad.

The haulage mechanism consists of one Esselius patented 'mule' car, with necessary haulage and tail ropes for operating it, upper and lower by-pass gates to allow the 'mule' to return to the pit when the car is standing between the mule-pit and the car-dumper. It is operated by a 300-kw. hoist, and the controlling mechanism is automatic. This mechanism is designed to handle cars having a total load of 215,000 pounds up an 11% grade at a rate of not less than 30 cars per hour, which is the rated capacity of the dumper. See Fig. 1, 2, and 3.

The car-dumper consists of a heavy steel framework supporting a rotating cradle upon which is mounted a movable platen, which carries the rails for the ore-cars. The cradle is supported from one side by heavy pivot-pins secured to the framework; the cradle-nose is rounded and provided with heavy wearing-plates. The 300-kw. hoisting mechanism for rotating the cradle is placed in a housing at the top of the framework. At the rear of the structure are the counter-guides and weights, with their cables passing over sheaves at the top of the framework, returning to the cradle and the three car-clamps. After the incoming car has been pushed onto the platen by the 'mule' and properly spotted on the platen by the car-rider, the cradle-hoisting machinery is started and the first motion of rotation releases the platen, which then moves sidewise toward the dumping side of the cradle until the side of the car rests against the cradle-blocking. Continued rotation of the cradle turns the car over so that it is tilted at an angle of 75° from the horizontal, thus discharging the contents of the car onto a grizzley. In winter when frozen ore is being unloaded and when a full load clings to the car, the cradle is not allowed to rotate beyond an angle of 45° from the horizontal. To take care of this extra weight additional counterweights are provided; they are readily attached at the will of the operator. During the motion of rotation the clamps which hold the car securely to the platen are automatically set by means of heavy counterweights which travel in guides at the rear of the main frame, these counterweights being operated by means of ropes so arranged that the pressure of the clamps is equally divided on the two sides of the car. This arrangement of clamps is a distinctive feature of the machine and causes less damage to cars than any other type constructed. The motion is entirely automatic, being controlled by counterweights and not dependent in any way on the operator. After the contents of the car have been discharged, the cradle returns to its original position, and the platen moves out so that its rails come into alignment with the approach and outgoing rails. The machine is now ready for another loaded car, which, on entering the cradle, displaces the empty car.

This entire unloading plant is operated by three car-riders, one car inspector, and one dumper operator, who, from his cab on the car-dumper, hauls the car up the incline and onto the dumper, where it is discharged.

As a reserve in case it becomes necessary to shut-down the car-dumper, the cars may be delivered over an auxiliary track to primary-storage bins of 10,000 tons capacity, where they are dumped from the bottom by hand. On top of these bins are grizzley-bars spaced 12 in. apart. Any pieces too large to pass are broken by sledge-hammers. Below these grizzlies are others, set at an angle of 40°, spaced 2⅜ in. apart, the bars being made of special

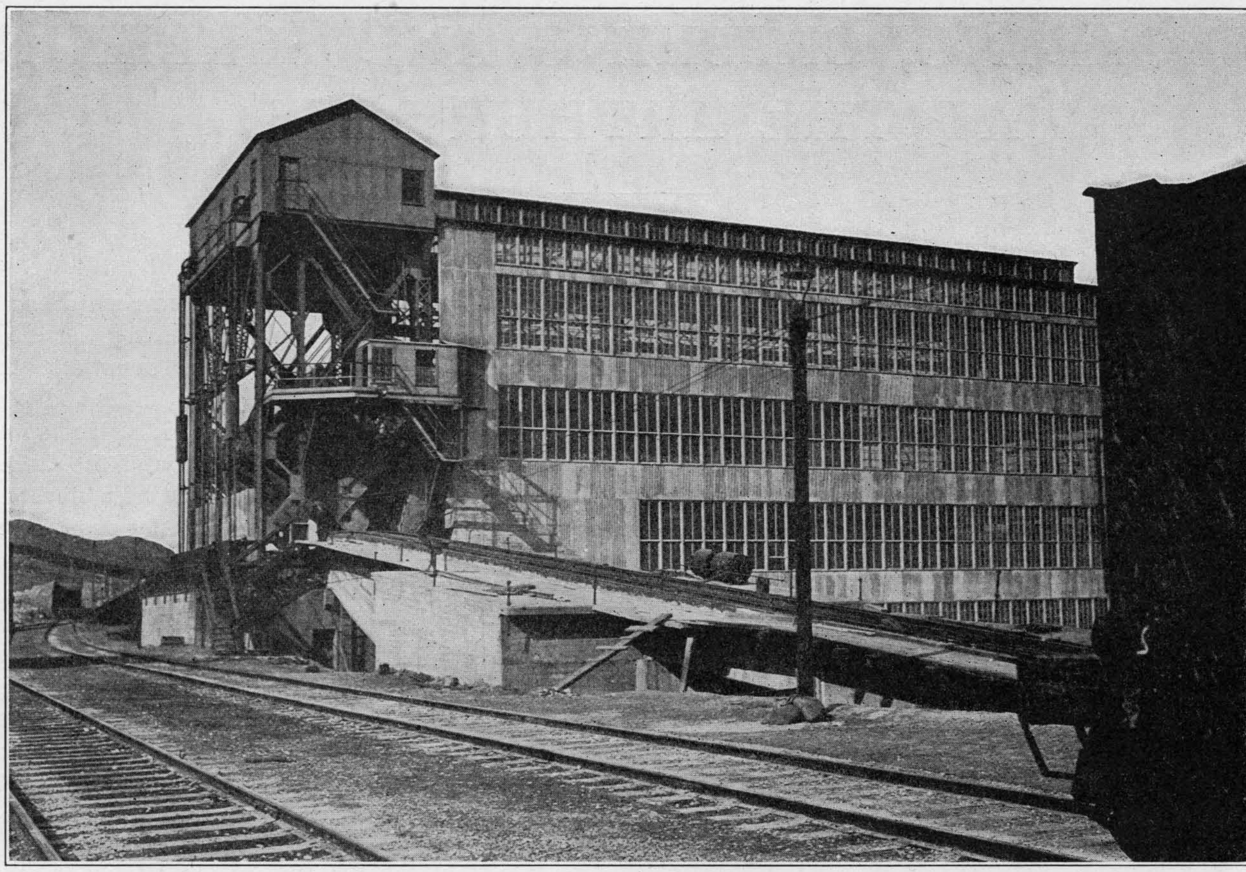


FIG. 1. CAR ON PLATEN OF CAR-DUMPER AT BEGINNING OF ELEVATION; ALSO EMPTY CAR RETURNING THROUGH KICK-BACK TO EMPTY-YARD

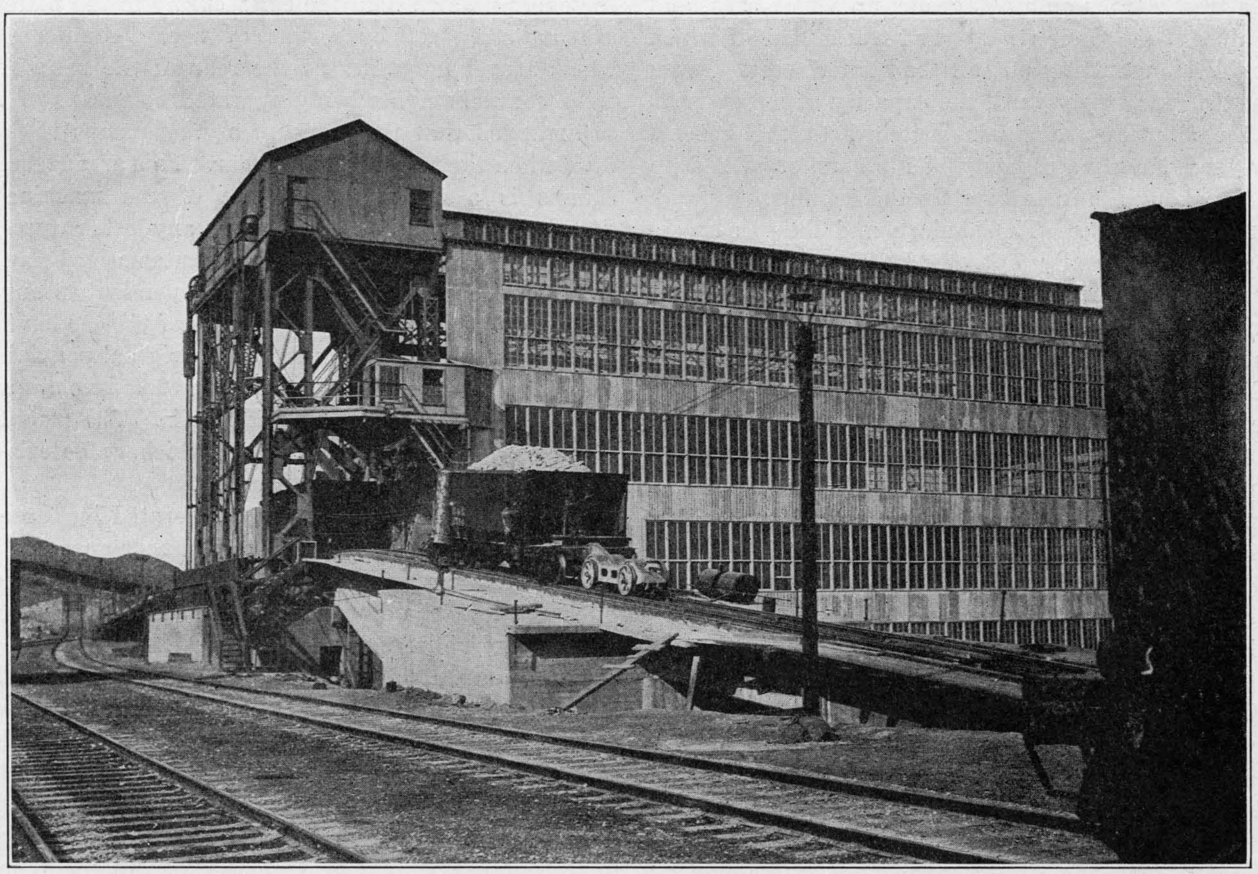


FIG. 2. CAR BEING PUSHED UP 11.5% GRADE BY ELECTRIC MULE

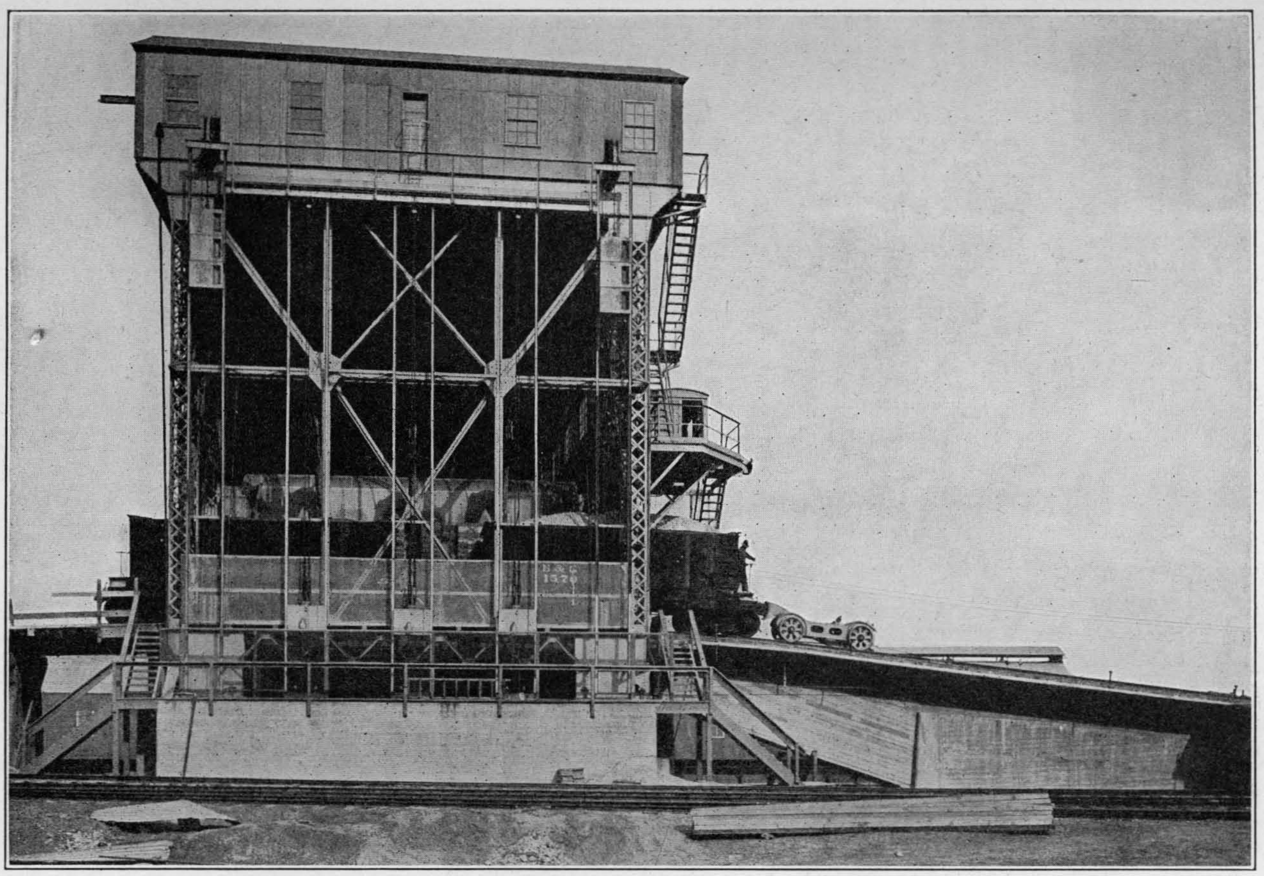


FIG. 3. LOAD KICKING EMPTY CAR OFF THE CAR-DUMPER PLATEN

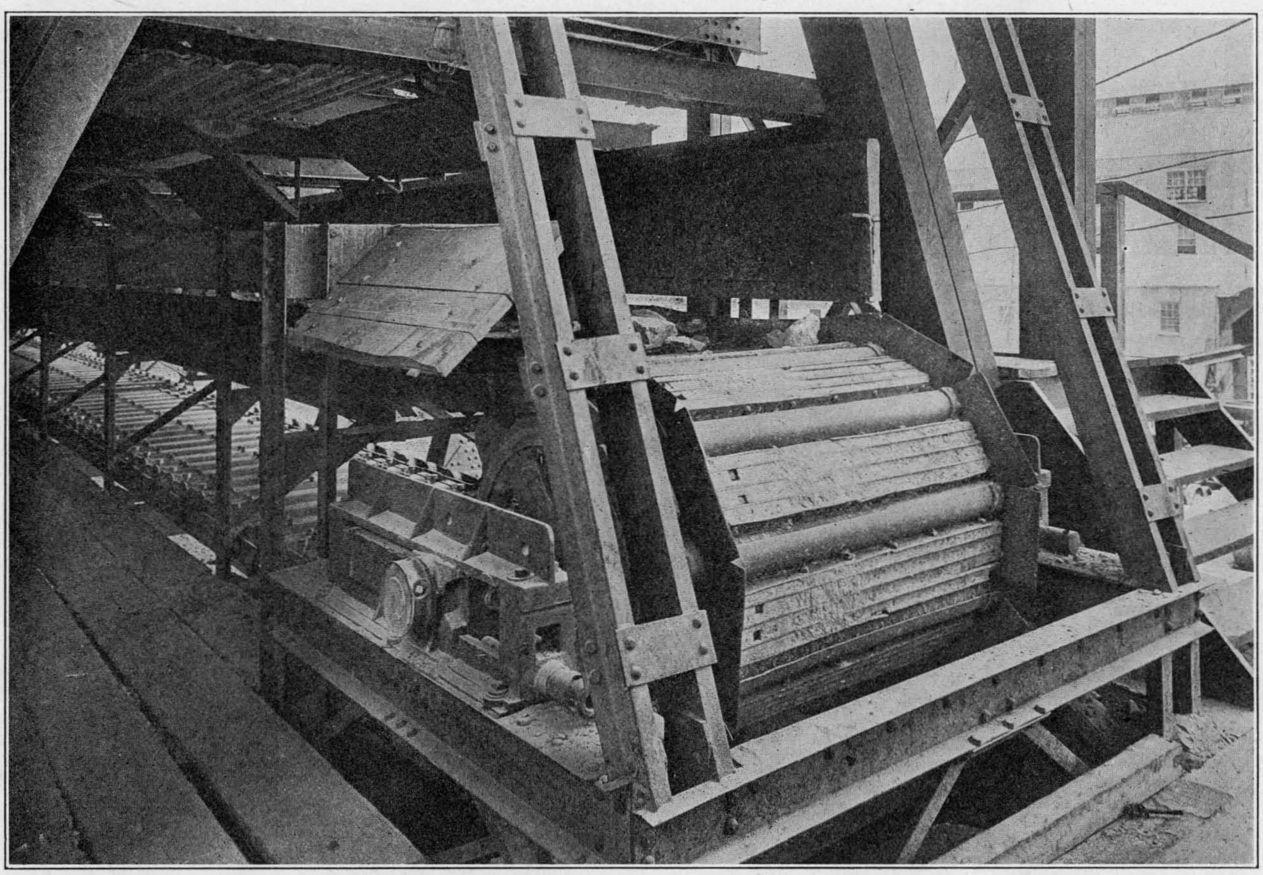


FIG. 4. STEEL PAN-CONVEYOR 48 INCHES WIDE

Fig. 5. GYRATORY CRUSHER, AND LOWER END OF GRIZZLEY

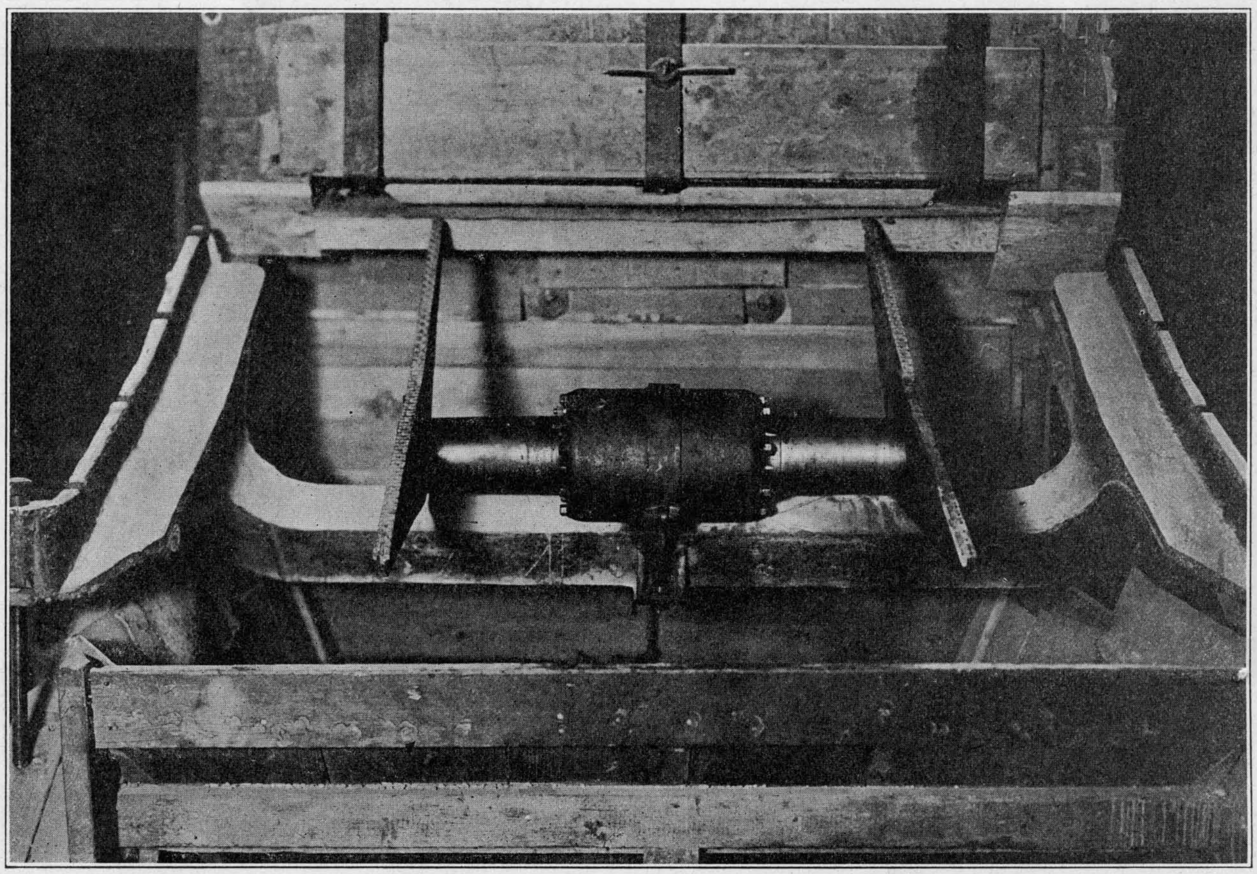


Fig. 6. MECHANISM OF DIRECT MOTOR-DRIVEN VIBRATING SCREEN

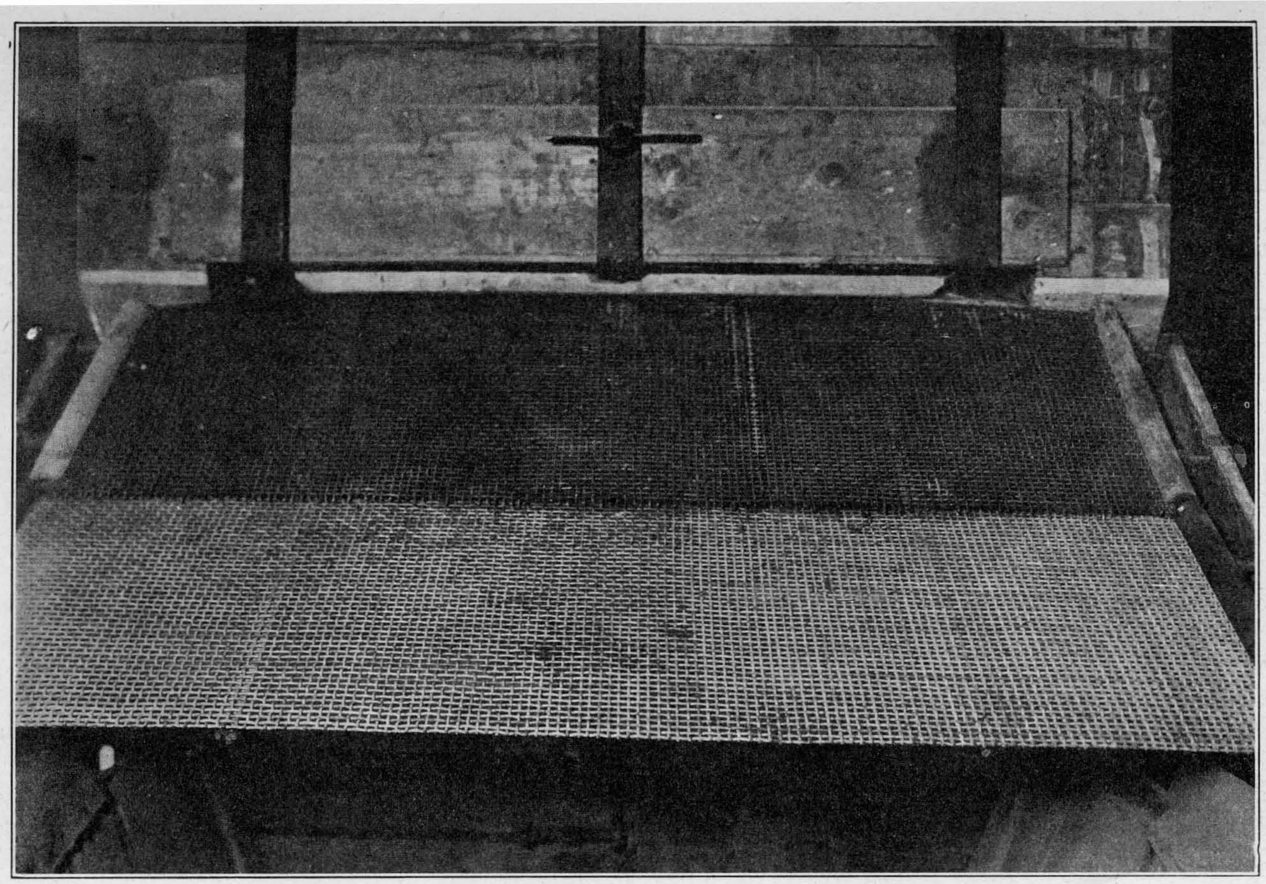


FIG. 7. TENSIONED SCREEN-CLOTH FOR RECEIVING VIBRATION

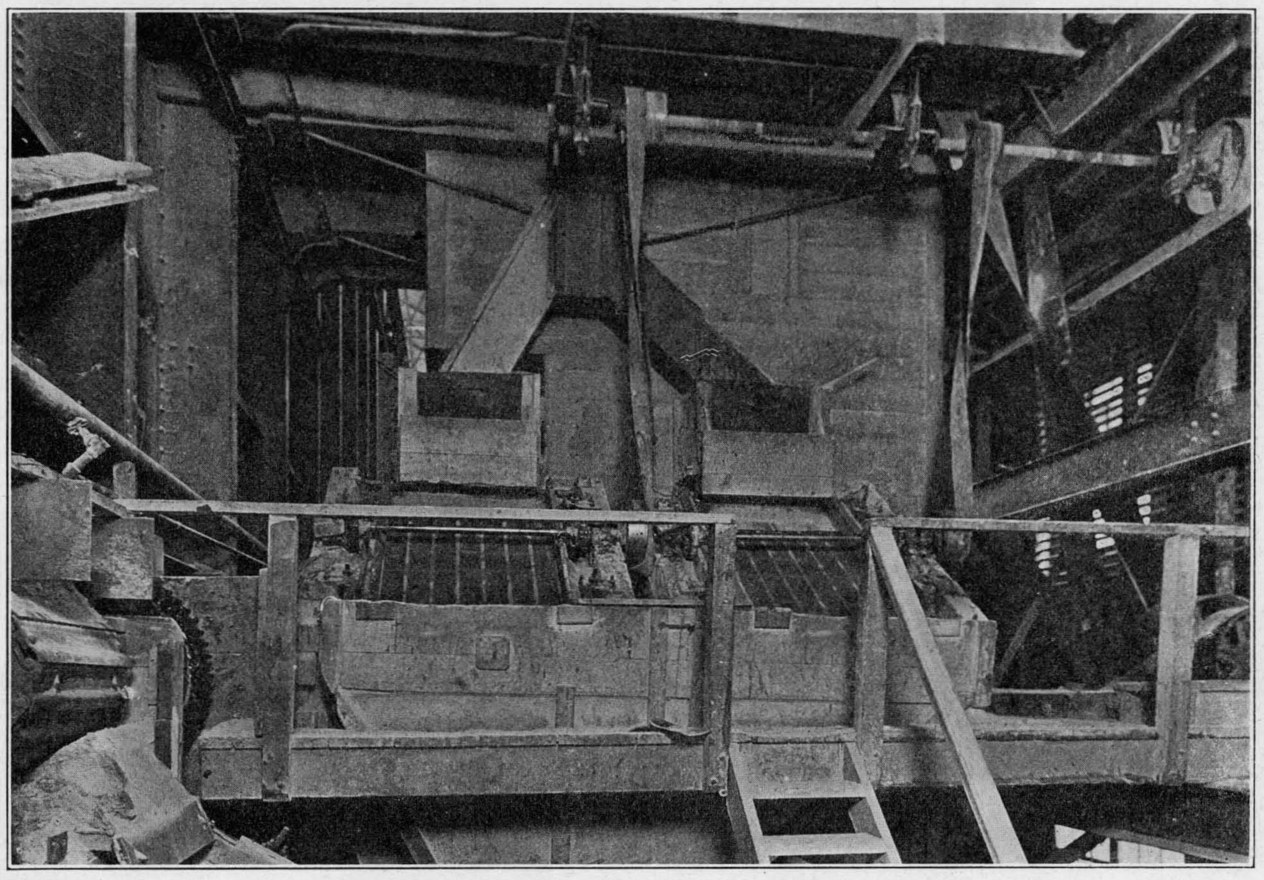


FIG. 8. ARRANGEMENT OF IMPACT-SCREENS WITH FEED-CHUTE

rolled steel. The oversize passes into pockets underneath and is drawn by steel apron-feeders, 48 in. wide, onto a steel pan-conveyor, also 48 in. wide. See Fig. 4. This carries it to two No. 8 McCully gyratory crushers, each of which serves a unit, and is supplemented by a spare crusher of the same kind available for either unit whenever required. The undersize of the lower grizzlies is drawn from pockets by steel apron-feeders 48 in. wide onto a rubber belt of the same width, which in turn delivers it to the first sizing-screen set at an angle of 42°, the apertures of which vary from 1 to 3 in. by changing the screen according to the character and moisture of the ore. From this point on the handling of the ore is the same as that coming from the car-dumper.

This system of unloading is ideal for seven months of the year—from March to November—except for those cars which are loaded with extra large rocks, the size of which prevents them from passing through the bottom of the cars, requiring hand-breaking from the cars and through the 12-in. grizzlies. During the period from December to February the ore freezes in the railroad cars to a depth of from 6 to 12 in. on the top, sides, and bottom of the cars. When the ore is frozen, the hopper-shaped bottom of the cars prevents its free discharge when the bottoms are dropped; an unloading gang of approximately 30 men per shift is required to pick and bar the entire contents of the car through the openings. The ore also freezes in the primary bins, and it appears to break more coarsely at the mine in the winter than in the summer, requiring excessive breaking of ore by hand in order to dump it through the first grizzlies. For these months the loss of capacity is equal to 25%, as well as being costly. These troubles are overcome by the car-tipple and the No. 27 gyratory crusher.

Returning to the car-dumper, the contents of a car, after being tilted, discharges onto grizzlies set at an angle of 35°, made up of 12-in. 28½-lb. Bethlehem I-beams, 7¾ in. by 32 ft. long, capped with a special manganese-steel casting. These built-up grizzley-bars are so spaced that the openings at the top are 5 in. and at the bottom 6 in., the object of this differential spacing being to prevent wedging of the oversize. The oversize passes to a No. 27 Allis-Chalmers gyratory crusher, with a double discharge, which, at the present time, is the largest ever built, designed to handle 54-in. material and reduce it to an average of 4½ inches. See Fig. 5. At the time this photograph was taken the bed of ore had not formed in the crusher-pit. This crusher is driven by a 300-hp. squirrel-cage motor at 600 r.p.m., direct-connected through a set of Wuest cut herringbone gears reducing the speed to 275 r.p.m. To take care of repairs on this crusher, the shaft and mantle of which weigh 55 tons, there is a 60-ton (with 10-ton auxiliary) 4-motor electric traveling crane. This crane also serves the drives on the 60-in. conveyors. The undersize from the grizzlies is collected in storage-pockets designed to hold 1300 tons of free-feeding ore, and is fed with the crusher-product by means of two 72-in. pan-conveyors discharging toward each other, onto a short grizzley with openings of 2½ in., to a 60-in. rubber belt-conveyor, the function of the short grizzley being to produce a bed of fine on the belt-conveyor before the larger material, which has passed the crusher and grizzlies, reaches the belt. The 60-in. conveyor is 224-ft. centres, set at an angle of 18°, and is operated by motors the speed of which may be varied at the option of the operator in the secondary crushing plant by means of a push-button control from a minimum speed of 115 ft. to a maximum speed of 350 ft. per minute. The flexibility of this control makes it possible for the secondary plant to be run at its greatest efficiency, owing to its finer grinding and smaller capacity as compared with the primary crushing plant. The 60-in. belt-conveyor discharges onto a second grizzley, the bars of which are specially bulb-shaped, 25 ft. long and inclined at an angle of 45°. The openings of this grizzley are changed to suit the character and moisture of the ore. The oversize from the grizzlies discharges into three No. 8 McCully gyratory crushers, and, as previously mentioned, two of the crushers, each of which serves a unit, are supplemented by a spare crusher of the same kind, available for either unit whenever required. It is at this point that the ore from the reserve crushing plant joins the circuit. The undersize from these grizzlies discharges onto two short rubber belt-conveyors, 36 in. wide, 20 ft. long, operating on an angle of 16°, at a speed of 400 ft. per minute. Serving each of these conveyors is a screen, 40 in. wide, 10 ft. long, and set at an angle of 45°. These screens take out the minus 1-inch material, which passes by means of a chute to rubber belt-conveyors delivering to the fine-crushing department. The oversize from this set of screens by-passes the McCully crushers but unites with their product.

This description is confined to one of the two units into which the car-dumper is divided, as each side of the crusher-discharge is an independent unit.

THE CRUSHED PRODUCT and screen-oversize are now divided into two parts, one going to the new roll installation and the other to the old. That portion going to the new is collected on a 60-in. rubber belt-conveyor, 25 ft. long, with a slope of 19°38′, and operating at a speed of 350 ft. per minute. The ore from this belt discharges to another conveyor running at right angles, also 60 in. wide, 150 ft. long, with a slope of 21°30′, traveling at the rate of 350 ft. per minute. This belt elevates and discharges into two 72 by 20-in. Garfield rolls of the moving pedestal type, operating at a speed of 125 r.p.m. The crushed product from each roll discharges on a separate rubber belt-conveyor, 42 in. wide, 118 ft. long, having a slope of 21°30′, and a belt-speed of 350 ft. per minute. At the terminus of each belt is a screen-tower, where the ore is delivered to four rotary feeders, each tower and set of feeders serving a roll. The feeders are 36 in. diameter, 64 in. long, and revolve at the rate of 2 r.p.m. Receiving the ore from each feeder is a 48 by 72-in. Mitchell vibratory screen. The oversize discharges onto a conveyor which runs horizontally at a speed of 250 ft. per minute.

It is 42 in. wide, 25 ft. long, and discharges into a set of 72 by 20-in. Garfield rolls, operating at 125 r.p.m. The roll-product returns to the 60-in. conveyor feeding the first set of rolls. This puts the product in a closed circuit with one 60-in., two 42-in. conveyors, four sets of 72 by 20-in. rolls, and eight Mitchell vibratory screens, where it remains until it is all crushed to pass the screen-aperture. The undersize from the vibrating screens is conveyed by a 36-in. conveyor, 60 ft. long, operating at a speed of 340 ft. per minute, to the two conveyors, each of which are 36 in. wide, 150 ft. long, and traveling at a rate of 250 ft. per minute, which, in turn, discharge to similar conveyors 120 ft. long, which carry the ore to the secondary bins. The last two sets of conveyors serve both the old and the new coarse-crushing plants. That part of the feed passing to the old crushing plant is discharged to the boot of a 36-in. elevator equipped with staggered rows of buckets. This elevator travels at the rate of 350 ft. per minute and has a lift of 72 ft. From the elevator the ore is discharged into a chute having a screen-bottom 24 in. wide, 52 in. long, set at an angle of 45°, the aperture of the screen being one inch. The undersize passes by means of a chute to a 36-in. conveyor, which delivers to the 36-in. conveyors delivering to the secondary bins. The oversize is fed to a set of 72 by 20-in. rolls operated at a speed of 100 r.p.m. From the rolls this material flows down a chute, a portion of which has a screen-bottom 30 in. wide and 120 ft. long, placed at an angle of 40°. The screen has an aperture of one inch, the undersize to the conveyors delivering to the secondary bins. The oversize passes to a 30-in. elevator equipped with staggered buckets, the lift being 60 ft. The elevator discharges into a chute, a portion of the bottom being equipped with a screen 24 in. wide, 52 in. long, having an aperture of one inch and inclined at 40°. The undersize joins the belt-conveyors delivering to the secondary bins. The oversize returns to the last-mentioned elevator, which puts it in a closed circuit with the elevator screens and rolls, where it remains until crushed to pass the aperture of the screen.

THE VIBRATING SCREEN mentioned in the description of the secondary-crushing plant, has been developed by B. A. Mitchell, mechanical engineer to the Utah Copper company, during the last eighteen months and is now being placed in service. The principle is new and unique. The vibration is developed by a fractional horse-power motor which is totally enclosed with the vibration-producing mechanism, and, therefore, is absolutely dust-proof, fool-proof, and water-proof. This mechanism is underneath the screen-cloth (See Fig. 6) and in close contact with it at all times, transmitting its peculiar vibration to the tensioned (See Fig. 7) screen-cloth. This construction gives a free open screening surface for the material to be screened and for observation by the operator; it also permits rapid changing of worn-out screens. This type of screen requires fewer operators, less repairs and power (the power required being ¼ horse-power); it gives greater effective screening surface; therefore, greater screening efficiency than any other screening mechanism.

An idea of the character of ore mined and concentrated may be gained from the analysis which represents the ore milled for the month of November 1917:

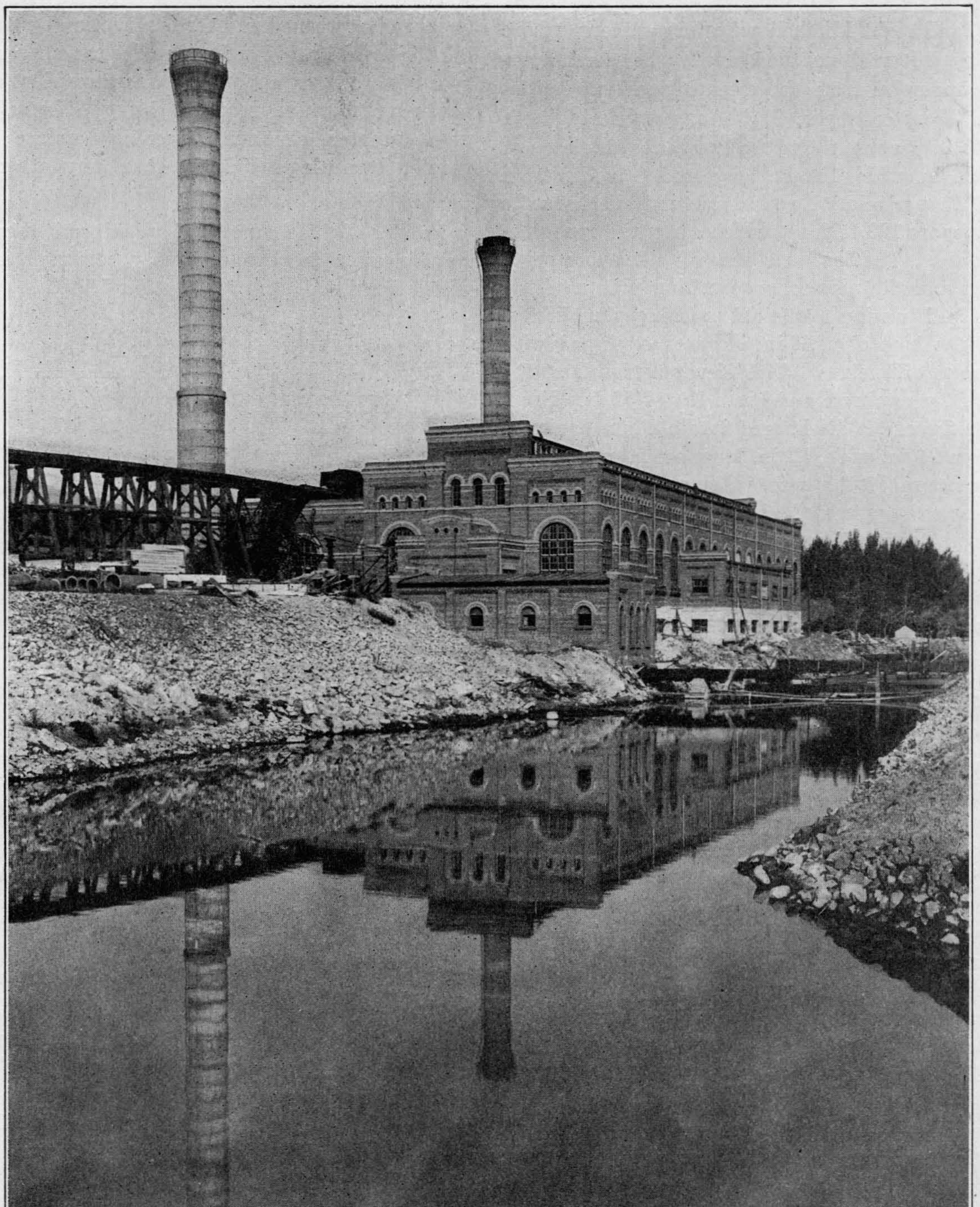
	%		
Copper	1.27	Gold	0.0075 oz.
Iron	2.55	Silver	0.0675 oz.
Silica	68.80	Loss on ignition	1.85%
Aluminum oxide	14.56		
Sulphur	2.64		
Calcium oxide	0.22		
Potassium oxide	5.94		
Magnesium oxide	1.32		

The following is screen-analysis of the product as it left the crushing plant prior to the installation of the additional crushing machinery, and is therefore coarser than that which will eventually be obtained with the additional crushing arrangements.

Coarse-Crushing Product

	Mesh and opening	Per cent material	Accum. % material
Opening	1.050 in.	1.17	1.17
"	0.742 "	8.44	9.61
"	0.525 "	22.27	31.88
"	0.371 "	16.30	48.18
Mesh	3	12.11	60.29
"	4	7.24	67.53
"	6	4.07	71.60
"	8	4.27	75.87
"	10	3.61	79.48
"	14	2.82	82.30
"	20	2.31	84.61
"	28	1.92	86.53
"	35	1.74	88.27
"	48	1.55	89.82
"	65	1.53	91.35
"	100	1.27	92.62
"	150	0.82	93.44
"	200	1.07	94.51
Pass	200	5.49	100.00

From the bins the ore is drawn by steel apron-feeders adjustable to three rates of speed by step-pulleys, to which is attached a mechanical counter or tachometer, registering the number of revolutions of the head-pulley of the feeder, so that it is easy to adjust the proportion of tonnage passing into the thirteen sections into which the mill is divided.



THE POWER-PLANT

THE POWER-PLANT, MACHINE-SHOP AND FOUNDRY

By FRANK G. JANNEY

POWER is being furnished to the Utah Copper Co.'s plants at Garfield by the Utah Power & Light Co. and is delivered from their terminal station, about five miles west of Salt Lake City, over two steel-tower transmission lines at 42,000 volts, three-phase, sixty-cycle, to the Utah Copper company's central station at Magna. These lines are connected into the station through 70,000-volt oil-circuit breakers to a ring bus in the station; each line is protected with reverse-energy relays, which, in the event of trouble on either of the lines, automatically disconnects it from the system, thus avoiding an interruption to the plants.

The ring bus, which is composed of 1-in. copper tubing, is so arranged that any part of the equipment can be cut out for inspection or repairs, through sectionalizing switches, without affecting service. Distribution is made from this station to the various sub-stations at the Magna and Arthur mills. Power is delivered to each sub-station with two circuits of three wires each of 250,000 C. M. stranded conductors at a potential of 42,000 volts; both circuits are supported on steel towers with suspension insulators, each circuit is controlled with overload relays, and is of ample capacity to furnish all power required at point of delivery.

At the sub-station the circuits are controlled by 44,000-volt oil circuit-breakers from which power is delivered to a vertical bus, supported with suspension insulators, through disconnecting switches and an automatic 44,000-volt circuit-breaker, to the transformers, through which the pressure is reduced to 460 volts. The installation at the Arthur sub-station consists of three banks of transformers, having a normal capacity of 21,500 kva. supplying current to 1283 motors, ranging in size from ¼ to 300 hp. at 440 volts. An extra transformer is provided and arranged with primary and secondary switches so that it may be put in service to relieve a defective transformer without causing an interruption of service.

The secondary distribution circuit to the motors is of three-conductor varnished cambric insulated cables with a double-braid weather-proof covering, tested for a working voltage of 1000 volts. Each secondary circuit is protected with an automatic oil-switch, through inverse time-limit overload-relays. Motors are protected with overload-relay and low-voltage release, by which in event of an interruption of power-service, all motors are automatically disconnected from their supply-circuit.

A complete signal system of colored lights is maintained between the sub-station and all parts of the mill, enabling the operators to control the power-supply at all points, and thus avoid any unnecessary high-peak demand. This signal system is also used for reaching any of the foremen or heads of departments that may be needed. A telephone system is also maintained with instruments placed in all offices and convenient places in the plant, thus enabling the superintendent and others in charge to be in close touch with operations in the various departments. The efficacy of the telephone and signal system is best demonstrated during an interruption of power service. When all the machinery is at rest the time required to get it moving and operating rarely requires more than 12 or 15 minutes.

FOUNDRY. In 1910 this was in a small, poorly lighted and ventilated, building 61 by 62 ft. Outgrowing this small building, a new shop was completed in the early part of 1911, the building being 80 by 120 ft. Remodeling and additions to the mills and the building of the Bingham & Garfield railroad increased the work to such an extent that the foundry was lengthened 75 ft. in 1912, 30 ft. in 1915, and 90 ft. in 1917. Today the main building of the foundry covers a space of 80 by 315 feet.

After allowing for chipping-room, core-room, and cupola-space, the molding-floors have an approximate area of 16,000 sq. ft. The foundry equipment is modern in every respect, making work-conditions ideal. In the core-room are modern oil-fired furnaces. The crane equipment includes two 15-ton traveling cranes in the central bay and one 3-ton electric traveling crane in the side bay. Both of these cranes run the entire length of the shop. Two gib-column cranes of 2000-ton capacity are in the core-room.

The iron foundry is equipped with No. 3, No. 5, and No. 7 Whiting cupolas, having a capacity in excess of 150,000 lb. per heat. The air-blast is furnished by three Westinghouse centrifugal air-compressors with a capacity of 19,200 cu. ft. per minute. The amount of air is registered by two Clark blast-meters. Cinders and foundry waste are handled by a Link-Belt elevator-hoist to the disposal yard.

Castings leave the chipping-room on cars, cross a set of scales, and are delivered to the machine-shop casting-yard or loading-dock. The charging-floor is built of steel and concrete, covered with square wooden blocks set on end and wedged with asphaltum. Circulation of fresh cool air, combined with good light and plenty of room, make a perfect charging-floor.

In the iron and coke yards, which are on a level with the charging-floor, are concrete bins for storing molding-sand, pig-iron, and scrap. All material in the material yard is handled with a 20-ton electric locomotive crane, equipped with magnet. This crane does all the unloading of material, handles the drop at the breaking-pit; it also handles the large boats into which the coke is loaded for delivery to the charging-floor.

The brass foundry is placed in a building separate from the iron foundry and is equipped with three oil-fired blast-furnaces, with a total capacity of 9000 lb. per day. Brass castings vary in size from a few ounces to 800 lb., and in quality from yellow brass to the best grade of bronze.

In both the iron and brass foundry castings are made by hand-molding, but most of the work is done by molding-machines of the latest type. Castings are not only made for the mills and mines and railroads of the Utah Copper company, but special machinery is made for the other associated companies. The production of castings has increased from a few thousand pounds per month in 1910 to 1,800,000 lb. per month in October, 1917. Up to January 1, 1919, there has been produced 70,000,000 lb. of castings. The working force in this department reached its maximum in October, 1917, when 240 men were employed.

The pattern-storage building is 126 ft. long, 64 ft. wide, and 41 ft. high; it consists of three floors equipped with pattern-racks. It is modern in every respect and approximately 10,000 patterns are kept in storage.

THE MACHINE-SHOP lies between the foundry on the east and the mill-building on the west. It is of timber and frame construction, 320 ft. long, 44 ft. wide, and 35 ft. high. The noteworthy feature of this building is its lighting arrangement, the entire structure being enclosed by continuous windows, which can be opened, thus giving perfect ventilation. The machine-shop floor is on the same level as the foundry. Castings from the foundry are either delivered direct into the machine-shop by service cars or are stored in the stock-yards and delivered to the machine-shop as required. At the west end of the shop are service-tracks connecting with the erecting or rigging sheds of the mill. Over these tracks is a 25-ton crane for handling castings direct into the plant or loading them on cars for shipment to the various plants. The heavy machine-tools of the Utah Copper company are in this shop, it being the intention to concentrate the heavy machining of all castings at this plant and then distribute them to the other plants as required. Approximately 75% of all machinery used in the construction and remodeling of the plants has been designed and fabricated in this shop. Machine-tool equipment is of the highest class and latest design, driven by individual motors. A clear aisle or runway space is provided through the shop. There are two electric overhead cranes, one of 10-ton and the other of 5-ton capacity. These cranes span the width of the shop and travel its length.

The blacksmith and boiler shops are under one roof, being an extension of the warehouse building. The floor of these shops is 10 ft. below that of the machine-shop. For the handling of heavy bar-iron and steel plate, there is a 5-ton hand-operated crane running the full length of the shops. Both shops are equipped with modern tools capable of doing the most complicated class of work.

SOME ENGINEERING FEATURES IN CONNECTION WITH OPERATIONS

By R. C. GEMMELL

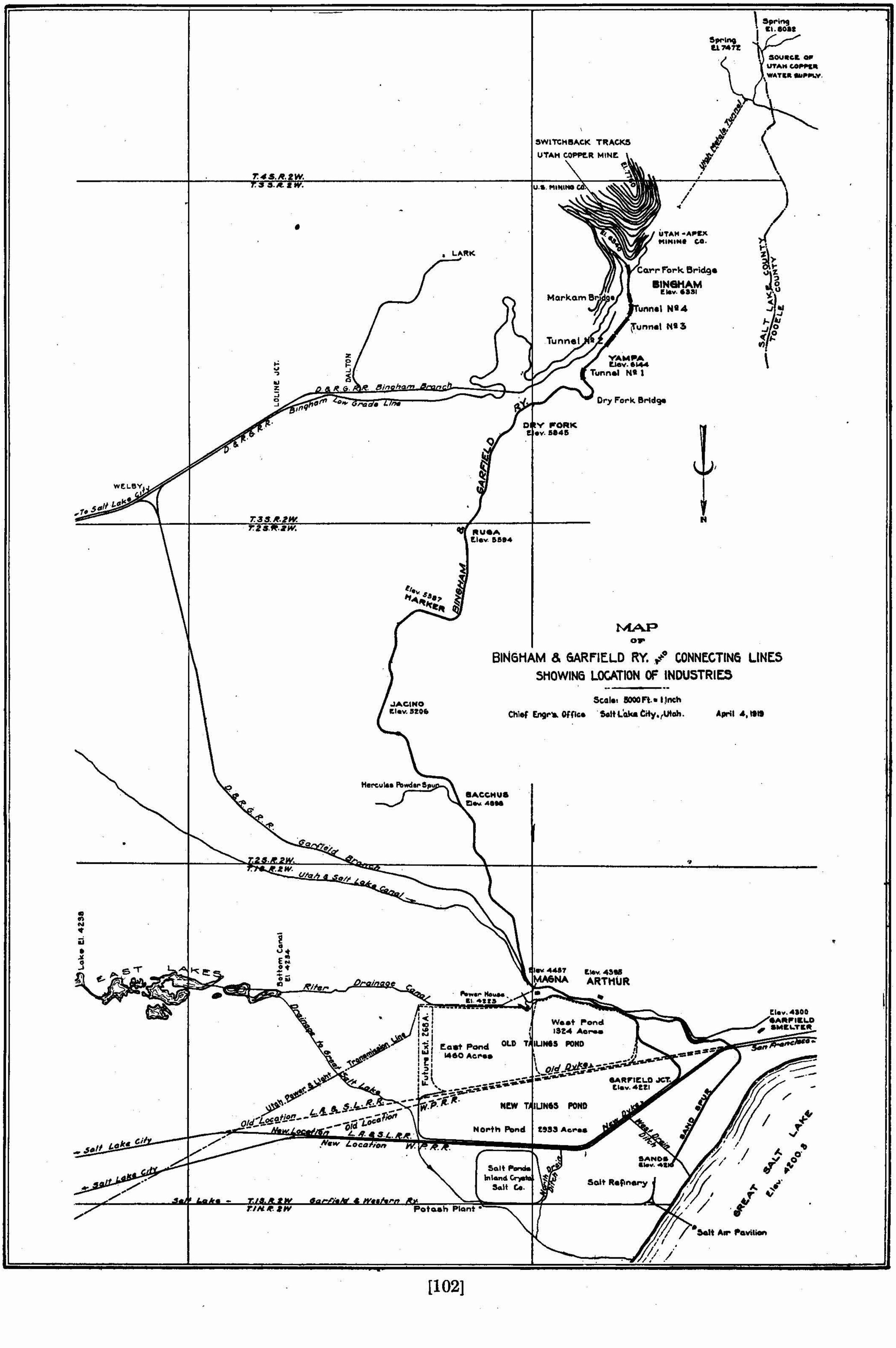
THE BINGHAM & GARFIELD RAILWAY. In order to ensure the proper and economical handling of its ore, the Bingham & Garfield railway was built during 1910 and 1911 by the Utah Copper company. The road traverses the eastern slope of the Oquirrh mountains from the mines, at Bingham, to the mills and smelter, at Garfield; the distance on the main line from station to station being about 20 miles. Branch lines to other industrial points increase the total length of the line to 37 miles. Additional tracks for serving the Utah Copper company and other mining companies at Bingham total 43.5 miles. This, together with about 53 miles of yard and siding tracks, makes the total length of all tracks 133.5 miles, as shown upon the map, Fig. 1.

The Denver & Rio Grande railroad already had in operation a road between Bingham and Garfield; this followed a winding and circuitous route between the mines at Bingham and the Magna plant, the distance between these two points being 27.5 miles. During the spring of 1908, the Utah Copper company engineers made surveys which proved that a more direct line could be built without encountering insurmountable difficulties, and in the spring of 1910 it was decided to begin construction. Surveys were made and lines were projected on grades varying from 2 to 2.5%, and, at first, a maximum curvature of 12° was assumed. A maximum grade of 2.5% was finally adopted upon the assurance of locomotive builders that they could furnish an articulated Mallet compound engine capable of hauling 40 empty cars of 60 tons capacity up such a grade at an average speed of 12 miles per hour. It was found practicable, except in two cases, to use nothing greater than 8° curves. These two exceptions, namely, a 10° 10-ft. curve for the Dry Fork crossing and a 10° curve immediately below, were employed in order to effect large savings in the cost of construction.

The greater portion of the road consists of heavy cuts and fills along the mountain slope. Just above the town of Bingham, however, because of the danger to the buildings in the canyon directly below and because of the excessive curvature that would be required to follow the canyon closely, it was decided to use the more costly method of tunnels and high viaducts. Starting at the mine, the road first crosses Carr Fork canyon on a steel viaduct of the tower-and-girder type, 690 ft. long and with a maximum height of 190 ft. See Fig. 2. From this viaduct the line traverses the Bingham assembly-yards and crosses Markham gulch on a second steel tower-and-girder viaduct, having a length of 640 ft. and a maximum height of 225 ft. After crossing Markham gulch, the track passes through four tunnels; the first is on a 3° curve, with a length of 1282 ft.; the second and third are on tangents, with lengths of 2085 and 773 ft., respectively, and the fourth is on a 6° curve, with a length of 682 ft. The last viaduct on the line—the one across Dry Fork—is also of the tower-and-girder type, with a length of 670 ft. and a maximum height of 188 ft. This viaduct is on a 10° 10′ curve, the curve forming a horseshoe having a total angle of 195° 42′. The line from Dry Fork to Magna runs through rough country, along and across the foothills, necessitating heavy cuts and fills, which render construction expensive. See Fig. 3. The entire line from the assembly-yard at Bingham to the Magna plant has an average grade of 2.05%; the distance being 17.2 miles, and the elevation at Bingham 6331 ft., whereas the altitude at Magna is 4457 ft. above sea-level.

The tunnels are all single-track, 18 ft. wide and 22 ft. high above the top of the rails. About half of the length of the tunnels is unlined. Where timbering was necessary, the three-segmental arch, with timber-plates along the tops of the posts, was used. The timbered sections are rendered partly fire-proof by means of redwood strips, inch-boards being used on the side segments and two-inch boards on the top section. In order to economize labor and permit of more rapid progress in tunneling, a 'jumbo,' or loading machine, was so devised as to permit the excavation being done on three benches simultaneously. This 'jumbo' was so constructed as to allow the broken rock from the two upper benches, or headings, to run into the cars by gravity, but it was necessary to shovel the rock from the lower heading. This arrangement made possible an average progress of about five feet per day. In order to rush the work as much as possible, air-drills only were used. Two crews, advancing from each end of the openings, were employed at all times. The work was so well planned and skilfully executed that it was completed on time, and with no fatal accidents to mar the record of the achievement. After the tunnels were completed, in order to have an accurate record of the yardage removed and a sectional outline of the openings, a tunnel-sectioner was devised and used with good results.

The care of the drainage along the right-of-way in the more mountainous portions of the road proved to be difficult. The standard openings adopted were concrete arches, varying in span from 3 to 10 ft. The greatest difficulty in draining was encountered in deep steep


MAP
OF
BINGHAM & GARFIELD RY. AND CONNECTING LINES
SHOWING LOCATION OF INDUSTRIES
Scale: 5000 Ft. = 1 Inch
Chief Engr's. Office Salt Lake City, Utah. April 4, 1919
SWITCHBACK TRACKS
UTAH COPPER MINE
U.S. MINING CO.
UTAH-APEX MINING CO.
Carr Fork Bridge
BINGHAM
Elev. 6331
Markam Bridge
Tunnel No 4
Tunnel No 3
Tunnel No 2
YAMPA
Elev. 6144
Tunnel No 1
Dry Fork Bridge
DRY FORK
Elev. 5845
LARK
DALTON
LOLINE JCT.
D. & R. G. R.R. Bingham Branch
Bingham Low Grade Line
D. & R. G. R.R.
WELBY
To Salt Lake City
BINGHAM & GARFIELD RY.
RUSA
Elev. 5594
HARKER
Elev. 5387
JACINO
Elev. 5206
Hercules Powder Spur
BACCHUS
Elev. 4896
D. & R. G. R.R. Garfield Branch
Utah & Salt Lake Canal
Spring
El. 7472
Spring
El. 8082
SOURCE OF UTAH COPPER WATER SUPPLY.
Utah Metals Tunnel
SALT LAKE COUNTY
TOOELE COUNTY
N
T.4S.R.2W.
T.3S.R.2W.
T.2S.R.2W.
T.1S.R.2W.
T.1N.R.2W.
EAST LAKES
Lake El. 4238
Bottom Canal El. 4234
Riter Drainage Canal
Drainage to Great Salt Lake
Utah Power & Light Transmission Line
MAGNA
Elev. 4487
ARTHUR
Elev. 4395
Power House El. 4323
West Pond
1324 Acres
East Pond
1460 Acres
OLD TAILINGS POND
Old Dyke
Future Ext. 265 A.
GARFIELD JCT.
Elev. 4221
NEW TAILINGS POND
North Pond 2933 Acres
New Dyke
West Drain Ditch
Elev. 4300
GARFIELD SMELTER
San Francisco
SAND SPUR
SANDS
Elev. 4210
GREAT SALT LAKE
Elev. 4200.5
Old Location L.A. & S.L. R.R.
New Location
Old Location L.A. & S.L. R.R.
New Location W.P. R.R.
W.P. R.R.
Salt Lake City
Salt Lake City
Salt Ponds Inland Crystal Salt Co.
North Drain Ditch
Salt Refinery
Salt Lake - Garfield & Western Ry.
Potash Plant
Salt Air Pavilion

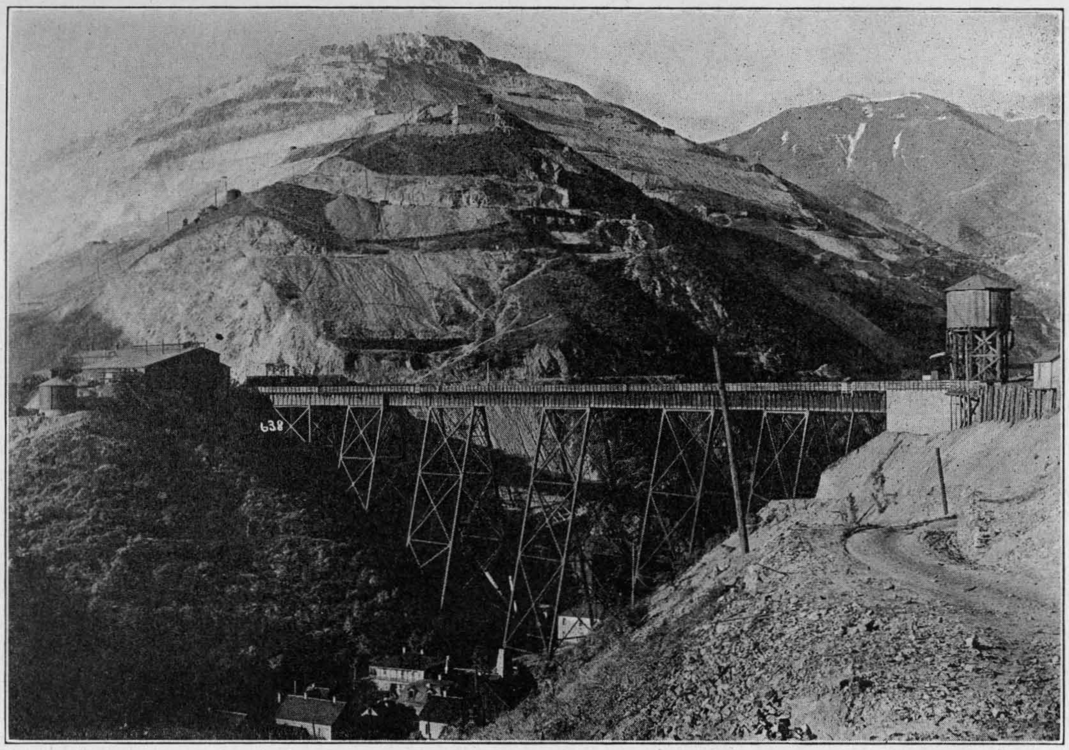


FIG. 2. CARR FORK VIADUCT, WITH THE UTAH COPPER MINE IN THE BACKGROUND

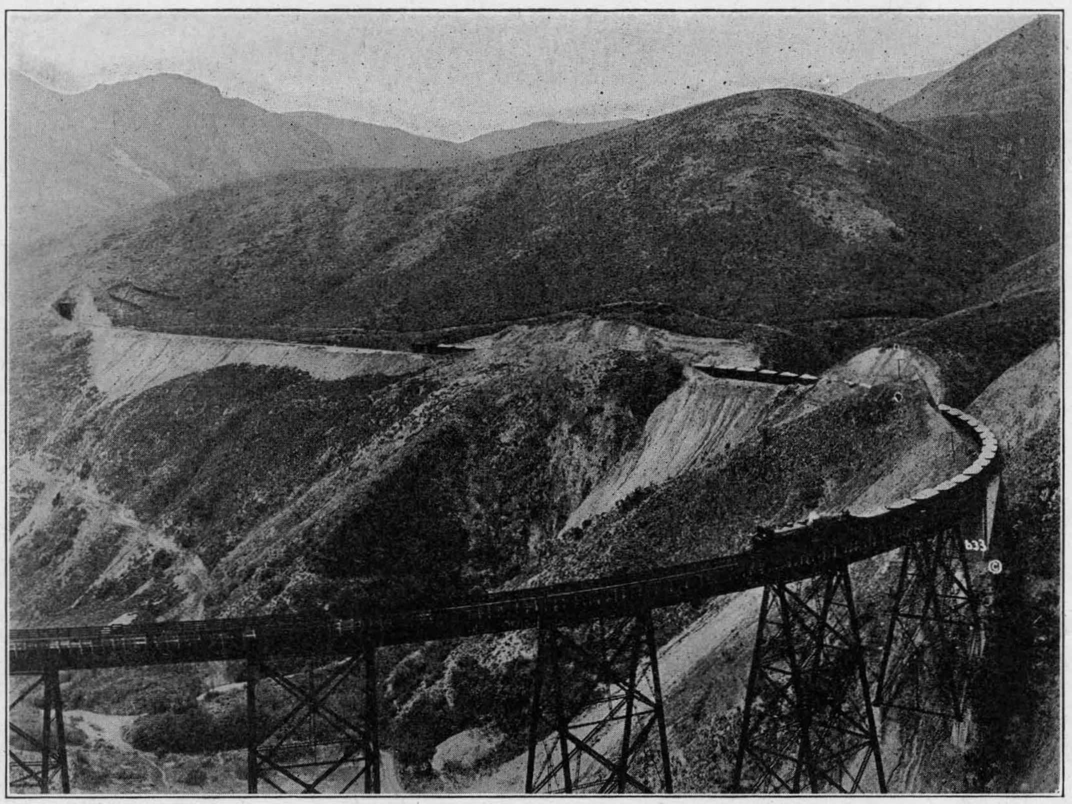


FIG. 3. DRY FORK VIADUCT

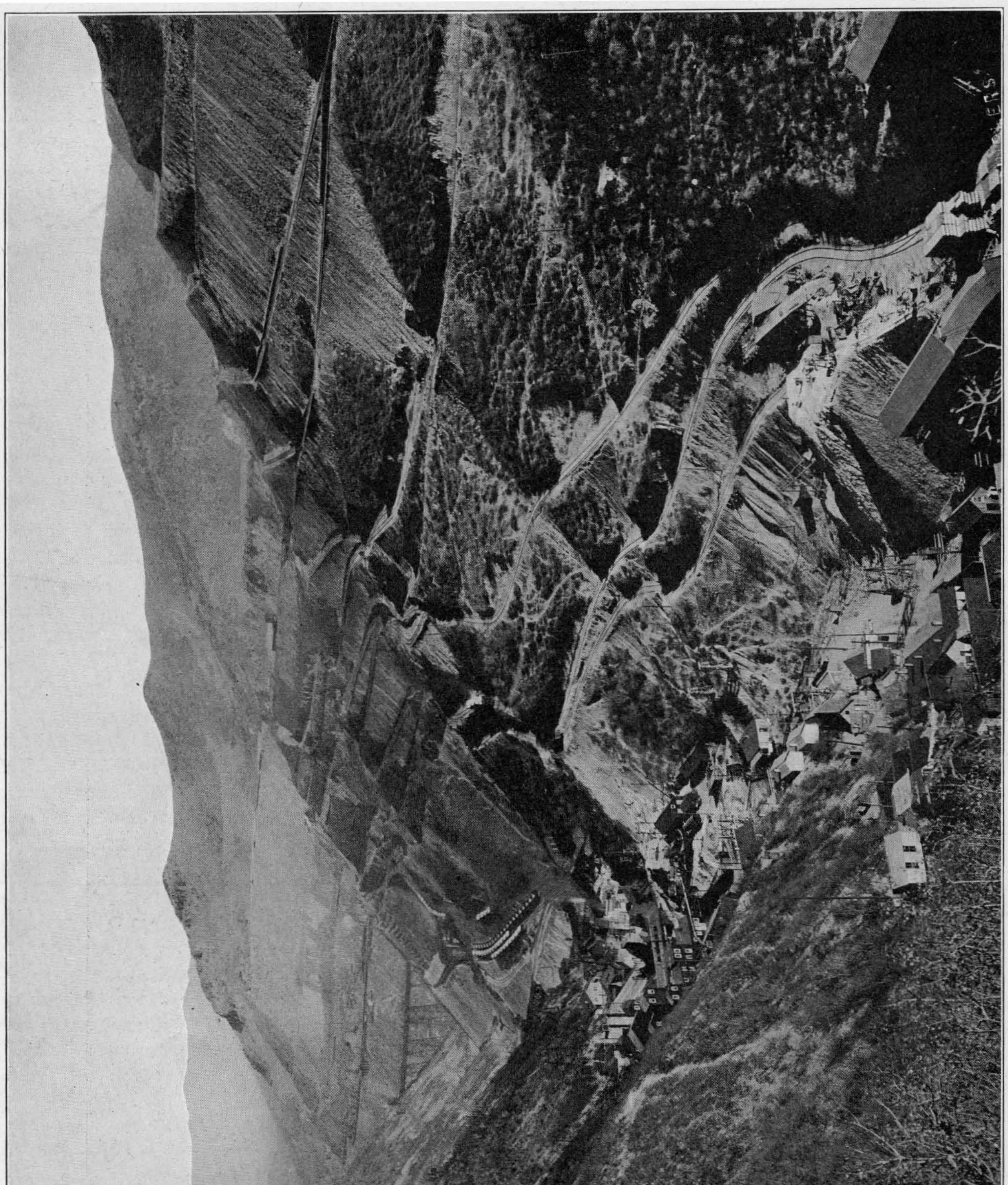


FIG. 4. SWITCH-BACK SYSTEM IN CARR FORK CANYON

gulches, where the greatest amount of embankment was required. In order to complete the work on time, it was necessary that the grading be done uninterruptedly; therefore it was decided to provide drainage by driving tunnels through the original ground, thus permitting the filling of the gulches without delay. Miners were employed in driving these tunnels, and this method of disposing of the drainage has proved satisfactory in every way.

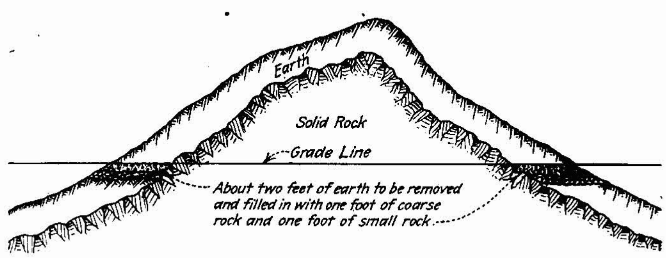
The contract for the grading and concrete work was given to the Utah Construction Co. on March 30, 1910. Men, teams, and equipment were in place and in operation by the early part of May. Grading was started on April 17, 1910, and the line was completed by April 1911. During this period, the Construction company moved 746,970 cubic yards of solid rock excavation, 618,222 cubic yards of loose rock excavation, and 315,070 cubic yards of earth excavation; it drove 4795 linear feet of railway-tunnel, 818 linear feet of drain-tunnel, used 1,494,416 ft. B.M. of timber for lining tunnels, and placed 11,698 cubic yards of concrete. This is considered a remarkable record in railroad construction on such a comparatively short piece of line.

An ingenious plan, known as the 'Swede trap,' was used in excavating the deep cuts on the road. This scheme involves the driving of a small tunnel along the centre of the cut and just below the grade of the roadbed large enough to accommodate a small dump-car. This tunnel is then protected by timbers loosely placed over the centre and on the grade-line. The material to be removed, after being loosened by blasting, runs onto the boards above the cars and is allowed to fall into the cars in the tunnel below by removing the timbering as required. In some of the larger cuts, as much as 95% of the excavation was handled in this manner.

The contract for the steel viaducts, aggregating 3000 tons of structural steel, was given to the Kansas City Structural Steel Co. This company had the plates and girders fabricated by the Pennsylvania Steel Co. and the towers fabricated by the Minneapolis Steel & Machinery Co. The actual construction was done by the Garrick & Garrick Construction Co. The work had been so well planned and placed in the field, and the steel so accurately fabricated in the shops, that no difficulty was encountered when the viaducts were erected. The concrete piers upon which the towers are constructed are 5 ft. square on the top and 9 ft. square at the largest section. They are built with a batter of one-half inch per foot. The height of the piers varies from 12 to 96 ft., depending on the distance to bedrock, upon which all of them are founded.

The heavy-service tracks of the road are laid with 90-lb., open-hearth, A. R. A. section, series 'A' rail. All of the other tracks are laid with 65-lb., open-hearth, A. S. C. A. section rails. Ties of untreated Oregon fir, 7 in. by 8 in. by 8 ft., spaced 18 ties to a 33-ft. rail and fully equipped with tie-plates, are used. The tie-plates under the 90-lb. rail are 8 by $8\frac{1}{2}$ in., whereas under the lighter rails 8 by 8-in. plates are used. All of the heavy-service tracks are ballasted with one foot of crushed rock below the ties.

During the year 1918 the road handled 12,439,394 tons of freight, being an average of 34,081 tons per day. Since nearly all this freight is handled down-grade from the mines at Bingham to the mills and smelter at Garfield, it was found unsafe to operate with more than 40 ore-cars per train, using standard braking equipment, and from actual tests made, it was determined that it was not advisable to load these 40 cars with ore weighing to exceed 56 tons per car when using such standard braking equipment. Therefore, arrangements were made to equip the ore-cars with the empty and load brakes, manufactured by the Westinghouse Air Brake Co. The heavy Mallet type of locomotive had solved the problem of moving empties up the grade at a satisfactory speed, and this special braking equipment was required to permit the safe operation of the heavy trains down the grade.



The empty and load brake so operates as to increase the total braking power, giving practically uniform braking power on car-units, whether empty or loaded, and regardless of the service in which they are being operated. The special braking equipment is entirely interchangeable with the standard freight-brakes, including the well known quick-service type 'K' retarded release and uniform re-charge freight-triple. In the actual road-tests when using the empty and load brakes, it was found that the cars could be loaded with 69 tons and that a 50-car train could be handled; at the same time maintaining a high factor of safety with a consumption of less than half the air by the standard equipment. Taken as a whole, this equipment has proved very satisfactory, holding the heavy trains to a uniform speed on the 2.5% grades.

The problems of construction and operation of the railroad at the mines were even more varied and complicated than on the main line. The mine being of the steam-shovel, or open-cut, type, it was necessary to construct railroads from the main assembly-yard at the base of the Utah Copper mountain to the various steam-shovel levels. There were 23 of these levels; the one at the top being at an elevation of about 1500 ft. above the bottom level. In addition to the tracks necessary to bring the ore down to the assembly-yards, long level lines were built to permit the removal of the large volume of cap, or waste, from the orebody. Owing to the necessity of continuous operations, the best way of providing facilities for transporting the ore from the various levels was by means of switchbacks, as illustrated in Fig. 4. In order to hasten the mine operations and make them more flexible, the various steam-shovel levels are served at both ends by these systems of switchbacks, and the trackage at the mine is so inter-connected that it is difficult to conceive of an accident or set of unusual circumstances which could seriously interrupt the output of ore.

During the average day's operations, 65,000 tons of ore and waste are transported over the tracks at the mine. The trains are handled by two types of locomotives; the smaller is a 50-ton 4-wheel saddle-tank type; the larger is a specially designed 80-ton side-tank switching locomotive. The smaller engines haul trains of five cars, while the heavier ones are used to serve the upper steam-shovel levels and handle from 10 to 12 cars at a time. Owing to the excessive steepness of the grade on the switchbacks and the sharp curves over which the ore-trains must be moved, unusual care is taken in the operation of the trains up and down the mountain. The present system necessitates the use of flagmen placed at advantageous intervals in towers, from which their view is entirely unobstructed. It is required of the locomotive engineers that they get clear right-of-way signals before they are allowed to move their trains to the next level.

In addition to transporting the business of the Utah Copper company, the tracks of the Bingham & Garfield railroad are available to the various mines at Bingham for freight service; in other words, the entire district has unusually good freight-handling facilities. The road is connected with the Denver & Rio Grande railroad at Bingham and at Magna, and with the Los Angeles & Salt Lake railroad and the Western Pacific railroad at Garfield. Branch lines have been constructed to the Garfield smelter of the American Smelting & Refining Co. and to the Bacchus plant of the Hercules Powder Co., which plant manufactures practically all of the powder consumed in Utah and the adjacent States.

Disposal of Tailing. The Utah Copper company commenced active operations of its mine and of its experimental mill in Bingham canyon on the first day of July 1904. At that time the ore-reserves were estimated at 37,500,000 tons. The Magna plant at Garfield was started during the month of June 1907, by which time the ore-reserves had been doubled. By the end of the year 1910, the reserves had been increased to 203,500,000 tons by development and through the acquisition of the Boston Consolidated Mining Co.'s property; and at the close of the year 1911, they had been further increased to 301,500,000 tons. When operations were started in 1907, the tailing from the Magna plant, owned by the Utah Copper Co., and those from the Arthur plant, then owned by the Boston Consolidated Mining Co., were discharged to the north of the mills and impounded behind small dikes, which had been built to protect the property lying to the east and the Los Angeles & Salt Lake railroad, lying to the north.

The total area of the tailing-ponds was 3000 acres, and by the end of the year 1915 the dikes around this area had been raised to an average height of 20 ft. By this time the estimate of ore-reserves had been increased to 390,000,000 tons, with a probability that much additional ore would be developed in the future. An investigation showed that in order to impound the ultimate tonnage of tailing, it would be necessary either to raise the dike around the 3000 acres to an average height of 80 ft., or move the Los Angeles & Salt Lake and the Western Pacific main tracks far enough north practically to double the area for tailing disposal. It was estimated that the cost of raising the existing dikes would be about $1,680,000, and that the cost of purchasing additional ground, moving the two railways, and building the new north dike would be about $860,000. Therefore the latter plan was adopted. One of the reasons for making this decision was the fact that if the Utah Copper company should store an enormous amount of tailing adjacent to the railway-lines for a distance of about four miles, with settling-ponds retained by dikes 50 to 100 ft. high, the impounded water and tailing would prove a continuous menace to the operation of the railways. After considerable time spent in negotiations with the officials of the railroad companies, their consent was obtained to the moving of their main-line tracks.

By this plan, the lands to be used solely for impounding tailing from the concentrating plants aggregate about 6000 acres. This area was bisected in an approximately east-west direction by the main-line tracks of the Los Angeles & Salt Lake and Western Pacific railroad companies, and it was necessary to move their tracks to the north a distance varying between 1 mile and 1½ miles for a length along the main lines of about 9 miles. The scheme also necessitated moving and extending the main line of the Bingham & Garfield railway near Garfield to a new connection with the Salt Lake route. The ultimate capacity of the pond will be sufficient to handle all the tailings rejected from the Magna and Arthur plants during the life of the property insofar as it can be estimated now.

By the appended map, Fig. 1, it will be noted that the new sites of the Los Angeles & Salt Lake and Western Pacific roads are parallel to each other and 100 ft. apart. In addition to railway construction, it was necessary to build about seven miles of dike around the new tailing-pond. The contract was given to the Utah Construction Co. at a unit price of 25 cents per cubic yard for all material on the railroad construction finished to a grade-line, and at a unit price of 20 cents per cubic yard for embankment in dikes, which it was not necessary to finish to any specified grade. Because of the wet condition of the ground, drag-line excavators were used. Ditches were first constructed to the Great Salt lake in order to drain the ground as much as possible before building the embankments for the two railroads. After the railroad embankments were completed, a tailing-dike was built, borrowing the necessary material from the inside. In all this work, the drag-line machines handled an average of about 22,000 cubic yards of material per month per machine. Concrete dewatering-boxes were constructed through the tailing-dike, and the dike was rip-rapped on the inside with slag obtained from the slag-dump of the Garfield smelter.

Water-Supply for the Plants. On account of the greatly increased tonnage, it became apparent that more water ought to be secured for milling operations; therefore, some years ago, the Utah Copper company filed on 100 cubic feet per second of the surplus water of Utah lake. In order to deliver this water to the plants, it became necessary to purchase a right-of-way through the Utah & Salt Lake canal, and to enlarge that canal from its intake at the Jordan narrows to the reservoir at the Magna plant, a distance of 27.5 miles. The work was finished on April 29, 1918. The water is delivered through the canal at an elevation of 18 ft. below the bottom of the Magna reservoir, thus effecting a large saving in the cost of power over the former method of lifting all the water 220 ft. from the lagoon at the old Magna steam-plant.

There are times during the winter season when the flow of water in the Utah & Salt Lake canal is blocked by snow and ice. In order to ensure a sufficient supply of mill-water at all times, and as a protection against unforeseen accidents, practically a duplicate water-supply was secured by the purchase of 1325 acres of land lying about 9 miles east of the plants. The topography of this land is such that it forms a natural reservoir for the drainage from higher lands; but, nevertheless, its elevation is sufficient so that the storage-water can flow by gravity to the lower pumping-plant at Magna through a canal constructed for that purpose. The position of these eastern lakes is shown on the map, Fig. 1, as well as the position of the Riter drainage canal through which the water is delivered to the pumping-plant. It is believed that the present water-supply of the Utah Copper company for milling purposes is now such as to make practically impossible any combination of circumstances that would necessitate the closing down of the mills, or even any reduction in their capacity, because of water shortage.