

Plate X.—A, Outcrop of Magma vein, from Flindt tunnel; B, Outcrop of Magma vein, glory hole, and No. 1 shaft.

which became the shipping point for the mines of the Pioneer district. The townsite of Hastings, immediately west of the Gem or Lake Superior and Arizona mine, was projected before 1882. Apparently the townsite was unsurveyed and allowed to lapse; no record other than the original application for it appears in the U.S. Land Office.

The Silver Queen mine was shut down about 1893 because of the low price of silver. George Lobb, from the Silver King, served as repairman and caretaker for several years. In 1901 he sorted out some ore from the Silver Queen dump. Henry Krumb, mining engineer, states in his first report to the Magma Copper Company, March 6, 1910: "Seven cars of silver ore were shipped by Mr. George Lobb in the early days of the property."

1906-7: A report dated November 9, 1906, by George Andrus, a mining engineer of Globe, shows that little exploration was carried on in the Silver Queen between 1882, the date of the map above mentioned, and 1906.

The Andrus report is summarized as follows: On the 100 level a crosscut driven 84 feet north from the shaft passed through porphyry and into quartzite. At or near the contact was an 8-inch streak of malachite. A crosscut driven south through porphyry and 80 feet into limestone showed no ore.

On the 200 level a crosscut north from the shaft passed from limestone into porphyry at 6 feet. At 34 feet a stringer of malachite was cut. A crosscut driven south 25 feet showed no ore.

On the 300 level a crosscut north from the shaft passed from limestone into porphyry at 36 feet. At 50 feet the vein (probably a fracture zone in the porphyry), 4 feet wide and completely oxidized, was encountered. A sample across the full width of the vein ran Au 0.04 oz., Ag 0.70 oz., Cu 2.4%. At 70 feet the crosscut passed into quartzite, and a drift eastward followed decomposed porphyry for 35 feet.

On the 400 level a crosscut north from the shaft passed from limestone into porphyry at 100 feet and near this point encountered a strong 5-foot vein in the porphyry. This vein was followed for 36 feet and was found to contain three streaks of oxidized copper minerals. The north streak, 18 inches wide, averaged 2.6% Cu, the south streak 3.0% Cu, and the center streak, 8 inches wide, assayed 10.2% Cu.

Andrus stated: "Technically there is no ore in sight, and none in quantities can be expected until the leached zone is passed through, unless the 'Superior vein' (the Martin-Troy contact. See p. 152.) develops ore, which is very probable as the Lake Superior and Arizona mine develops ore at a very shallow depth." He recommended that the drift be driven west along the lime-porphyry contact on the 400 level to the intersection of the "Superior vein" on the south side of the (Magma) fault. The shaft should be sunk 200 feet deeper before further crosscutting. "An adit should be driven from Hub creek (now Magma wash). It will follow the (Silver Queen) vein about 500 feet and enough ore will be taken out to pay expenses." (This adit, called the Flindt tunnel, was driven in 1910.) At the time of his visit, hoisting was done by a horse whim and buckets.

In 1906, when the Lake Superior and Arizona mine was being actively developed, attention was called to other properties in the vicinity. Messrs. Fisk and Crawford, of Globe, leased the Silver Queen mine from Mr. Swain and organized the Queen Copper Mining Company. Officers of this company were A. M. Crawford, President; Richard Fleming, Vice-President; E. F. Kellner, Secretary and Treasurer; William D. Fisk and George Gausler, Directors. A steam hoist was installed at the Silver Queen shaft, on the crest of a ridge about 500 feet above the flat. Wood fuel for the boiler was hauled by pack animals. Operations were carried on for about a year following late 1906 but were suspended when the First National Bank of Globe failed during the depression of 1907.

Mr. Kellner states³⁴ that no work was done by the Queen Copper Mining Company below the 300 level. No operations below

³⁴Oral communication, November 28, 1942.

water level, which fluctuated about 50 feet annually, were attempted. In the winter of 1906-7 some chalcocite was found on the 300 level in small stopes left by the early silver miners. Oxidized copper ores were not mined during the period of the lease. In 1907, 212 tons of ore, mostly chalcocite, were shipped to the Humboldt (?) Smelter. Returns were: Cu 23.03%, Ag 65.66 oz., Au 0.21 oz., Fe 7.85%; total value, \$27,777.

After 1907, as the bond and option were about to expire, Mr. Fisk purchased control of the stock of the Silver Queen Mining Company.

1910-11: Mr. Fred Flindt, a field mining engineer for George Gunn and William Boyce Thompson, came to Superior to examine the Daggs group of claims south of Queen Creek. While there his attention was attracted to the strong faulting along the Silver Queen vein, and he took an option on the property for his principals.

Messrs. Gunn and Thompson were experienced mine operators. Mr. Gunn, whose headquarters were in Salt Lake City, was interested in coal mines in southern Wyoming. Mr. Thompson was largely instrumental in the development of the Inspiration Copper mine at Miami, Arizona. At their direction a more detailed examination of the Silver Queen was made by Henry Krumb. Mr. Krumb has been consulting mining engineer for the mine since that date and is now (1942) a director of the Magma Copper Company.

His report dated March 6, 1910, is summarized as follows:

I was very favorably impressed with this property, as it has the strongest cross fissure in the district. The lime-quartzite contact on the north side of the dike has been thrown about 400 feet to the east relative to the contact on the south side of the dike.

On the 300-foot level the work is all north of the shaft, as the footwall of the dike is 40 feet north of the shaft. In a drift to the east a bunch of high-grade chalcocite was exposed, and above the level some stoping had been done. The plan of development should be to locate the intersection of the lime-quartzite with the dike and the fissure at the level of enrichment. As there is some chalcocite exposed on the 300-foot level, this may be the proper horizon, but I would suggest that the 400-foot level be unwatered for inspection, as I believe the 300-foot level is not deep enough. The lime-quartzite contact should be explored on both sides of the dike. If this is done on the 300-foot level, a drift should be driven east on the north side of the dike and another drift should be driven west on the south side of the dike.

About June, 1910, soon after this report was submitted, the Magma Copper Company was organized. The company was incorporated under the laws of Maine, with capitalization of \$1,000,000, shares \$5 par. First officers were E. M. Leavitt, President and Treasurer, and F. S. Schmidt, Engineer. James Neary, from the Lake Superior and Arizona mine, became Superintendent. The company took a bond and lease dated August 1, 1910, on the Silver Queen property. The Silver Queen Company was

to be paid \$500,000, of which, at the option of the Magma, \$300,000 might be paid in shares of the Magma Copper Company. Work was started at once. The Flindt adit tunnel was driven from the surface and connected with the Silver Queen shaft on the 215 level. The shaft was deepened to 650 feet. This work was completed by the middle of 1911.

1912: The shaft was deepened to the 800 level, and the 500, 600, and 800 levels were actively developed. An operating staff was transferred from the Inspiration mine to the Magma mine. Among these men were W. C. Browning, who became General Manager of the Magma Copper Company; E. H. Lundquist, Mine Superintendent; Henry Robinson, Master Mechanic; and I. A. Ettliger, Chief Engineer. Operations were carried on thenceforth with vigor and efficiency. Officials of the Magma Copper Company were H. F. J. Knobloch, President; H. E. Dodge, Secretary-Treasurer; W. H. Aldridge, Managing Director.

During 1910-12 the Silver Queen vein was not highly regarded as a prospective ore zone. Work in the adjoining Lake Superior and Arizona mine had focused attention on the Cambrian quartzite-Devonian limestone contact, hereinafter referred to as the L. S. and A. contact (the Superior vein of Andrus' report). Accordingly, when the 650 level was opened a crosscut was driven west to intersect this contact. It was found to be barren. A drift followed the contact, and where it intersected the Silver Queen porphyry dike a rich bonanza of supergene chalcocite was found. The Silver Queen vein, which followed the dike, was immediately recognized as the productive structure, and exploration was centered upon it.

In the summer of 1912, Dr. F. L. Ransome, of the U.S. Geological Survey, spent two days in the district. In a short description of the Queen mine, he noted that large ore bodies recently had been opened on the 800 level which exceeded in size those above that level. Bornite and chalcopyrite were the predominant minerals on the 800 level.³⁵

The townsite of Superior was surveyed in 1912 by Ed Stewart, of Globe. Previous to that time the village consisted of a few wooden dwellings and tent houses scattered haphazardly on the plain north of Queen Creek. The post office had been established in 1902, at the beginning of operations in the L. S. and A. mine. It was suggested that the new post office be named Sieboth, after A. C. Sieboth, Superintendent of that mine, but this was not acceptable to the postal authorities, and the name Superior, after the mining company, was selected. George Lobb became the first postmaster and was succeeded in 1907 by E. F. Kellner, Jr. The town, never incorporated, is governed by representatives of the County Supervisors.

³⁵Ransome, F. L., Copper deposits near Superior, Arizona: U.S. Geol. Survey Bull. 540, p. 153, 1914.

In 1912, ore from the mine was hauled by wagons to Florence at a cost of \$5.00 per ton, and the return rate for supplies was \$7.00 per ton. From Florence the ore was shipped by rail to the Hayden smelter. Only high-grade ore was shipped, but lower-grade material was developed. Shipments during 1912 were as follows:

Class	Tons	Cu	Ag	Au	Fe	Value, less treatment and freight
1st	104	48.19	68.0	0.11	11.2	\$ 15,495
2nd	3,168	15.92	18.02	0.06	12.4	\$141,542

1913: In a report dated April 21, 1913, Mr. Krumb stated:

The ore is not in continuous shoots, but is irregular in lenses or pockets. To a depth of 400 feet the ore occurred in small bunches, but on the 650- and 800-foot levels, it is more continuous. I consider this a favorable indication and expect that with greater depth the ore occurrence will be still more regular and in larger bodies.

The mine cannot be operated at a profit with expensive steam power as at present.

As a result of favorable showings on the 800 level, the company prepared to expand operations. After experiments on Magma ores by the General Engineering Company, of Salt Lake City, a concentration process, using gravity separation and Callow pneumatic flotation machines, was evolved.

1914: Work on the concentrator, begun early in the year, was completed in August. The initial capacity was about 200 tons per day. An aerial tramway 2,600 feet long was constructed from the portal of the Flindt tunnel to the new mill.

The increased scale of operations necessitated more adequate transportation facilities. Some years previously the L. S. and A. Company had surveyed a broad-gauge branch railroad from Florence. Its estimated cost, about \$500,000, was considered by the Magma Copper Company as too high for that time. It was decided to build a narrow-gauge railroad 30.4 miles long, connecting with the Arizona Eastern Railroad near Florence. The minimum estimated cost was \$160,000. E. G. Dentzer,³⁶ who had been a construction engineer for the Inspiration Copper Company, directed the survey and construction of the new railroad. Work upon it commenced November, 1914, and was completed May, 1915. As first built, it amounted to little more than stringers laid on the ground, with maximum grade of about 4 per cent and maximum curvature radius of 120 feet. The estimated cost was closely met, and improvements came later. At this stage a greater investment seemed unjustified, as the future of the mine was uncertain. It was not then known whether the bornite of the bottom level was supergene (secondary) or hypogene (primary); if supergene, the ore in depth might change to a lower grade.

³⁶Now General Manager of the Magma Copper Company.

A power line 15 miles long was constructed to Inspiration, where it connected with the government-owned power line from Roosevelt dam, on Salt River. Power was contracted from the U.S. Reclamation Service for about \$49 per-horsepower year.

Altogether, improvements completed during the year cost nearly a million dollars.

1915: A new three-compartment shaft, No. 2, located 400 feet north of No. 1 shaft, was sunk from the Flindt tunnel from the 215 to the 1,200 level. This became the main working shaft of the mine, and No. 1 continued to be used as an auxiliary for the 800 and higher levels.

The mine earned a net profit of \$611,729.02.

1916: No. 2 shaft was sunk to the 1,500 level, on which the Magma vein was found to be mineralized from wall to wall. Where intersected by the main crosscut south from No. 2 shaft, the vein was 34 feet thick, with an average of 10.52% Cu, 5.37 oz. Ag, and 1.26 oz. Au per ton. The copper was mainly in chalcopyrite and bornite. Drifting on the vein showed an average for 235 feet of 5.00% Cu, 4.4 oz. Ag, and 0.52 oz. Au.

In the annual report for 1916, W. H. Aldridge, President of the Magma Copper Company, wrote: "The favorable showing on the 1,500-foot level, which is not yet fully developed, insures the continuation of the main ore shoot below the 1,500-foot level. As to what depth is problematical, but the fact that the bornite and chalcopyrite found in the lower levels is believed to be primary ore, the chances are favorable for the main ore shoot to continue for a considerable depth." A noteworthy feature is his statement regarding the primary (hypogene) origin of the chalcopyrite and bornite. It was based on the results of polished-section study of the ores, made by McLaughlin.³⁷

The capacity of the concentrator was increased to 300 tons daily. The additional section was designed for zinc-lead ores but was converted to treat copper ores.

1917: No. 2 shaft was deepened from the 1,500 to the 1,800 level, and intervening levels 100 feet apart were opened. All showed satisfactory copper values. As only No. 2 shaft extended below the 800 level, a new vertical shaft, No. 3, was started late in 1917, with its collar on the surface 400 feet south of No. 1 shaft. The southward dip of the vein was well established by that time, and the new shaft was expected to intersect it at some point below the 3,000 level. A contract was made with the U.S. Reclamation Service to construct a power line of 600 kw. capacity from Goldfield, on the Salt River, to Superior, supplementing the power line from Inspiration. That year the mine made a net profit of \$1,067,986.

³⁷McLaughlin, D. H., The origin and occurrence of bornite: Doctor's dissertation, Harvard University, 1917.

1918: No. 3 shaft was completed to the 1,600 level, and the Goldfield power line was constructed. The cost per pound of copper mined was 16.425 cents and the selling price about 25 cents. The high unit cost was due partly to increased prices during the first World War and partly to rapid development of the mine.

1919: Both No. 2 and No. 3 shafts were deepened to the 2,000 level. No. 3 shaft was equipped as the main hoisting shaft for depths below the 2,000 level. The main tunnel, connecting the surface with Nos. 2 and 3 shafts at the 500 level, was driven. Up to that time all hoisting had been to the Flindt tunnel level, nearly 300 feet higher.

In 1919 the net cost of producing copper was 14.92 cents per pound and the average net selling price 18.53 cents per pound. Net earnings, disregarding depletion, were \$178,077.

1920: Operations were somewhat curtailed, owing to the low price of copper and the high cost of supplies. Development on the 1,600 and 1,800 levels was continued actively. On the 1,600 level the main ore body was opened for a length of 1,300 feet and an average width of 12.5 feet. On the 1,800 level the corresponding stoping length was 1,400 feet and the average width 21.5 feet.

Electric haulage was installed on the Main tunnel (500) and 2,000 levels. During the year all classes of development work totaled 6,515 feet and showed 55.4 tons of ore per foot. Reserves of ore above the 2,000 level were estimated by Mr. Browning as follows:

	<u>Tons</u>	<u>Cu (%)</u>	<u>Ag (oz.)</u>	<u>Au (oz.)</u>
Copper sulfide ore.....	827,000	5.00	3.0	0.02
		<u>Zn (%)</u>	<u>Pb (%)</u>	<u>Ag (oz.)</u>
Zinc-lead ore.....	20,000	13.5	2.0	12.0

Total production for the year was 100,872 tons averaging 4.93% Cu, 3.412 oz. Ag, and 0.438 oz. Au.

1921: The company mined ore only during the first three months of 1921, and the concentrator was shut down at the end of March. The average cost of producing copper was 14.9 cents and the average net selling price 14.7 cents. The depression of 1921 was marked by an accumulation of nearly 8,000,000 pounds of unsold copper.

Development was continued with excellent results. In the annual report for that year, Mr. Browning made the following comments:

Some of the last cross-cuts driven on the 2,000 level have given very interesting mineral results. In addition to the bornite and chalcopyrite which have been the main copper mineralization for some time, these cross-cuts are showing quite an appreciable amount of chalcocite and gray copper (tennantite) which are very likely of primary origin, the same as the

chalcopyrite and bornite. It is exceedingly interesting and gratifying to secure these high-grade copper minerals at the depth now attained in the Magma mine, and in my opinion gives very encouraging indications that we may expect high-grade values for quite some greater depth.

These last observations are based in part on the results of a study by M. N. Short of polished sections of specimens collected during the summer of 1921.

Mr. Browning writes further:

It is interesting to compare the size of the ore body on the upper levels of the mine with the size of the main ore body on the lower levels of the mine, as represented by the 1,800 level. It will be recalled that above the 1,000 level the ore shoot was only 300 to 500 feet long, and from 5 to 8 feet in width. On the new lower levels there will be mined for every vertical foot of ore shoot about ten times the amount of ore that was produced per foot on the upper levels.

No. 2 shaft was sunk to a depth of 2,450 feet below the collar of No. 1 shaft. It was concreted between the 600 and 1,200 levels, where swelling ground near the vein endangered its alignment.

1922: As the mine had enough ore reserve to last for several years at an output of 600 tons per day, no new development work was undertaken. No. 2 shaft was deepened 100 feet. Sinking of the new air shaft, No. 4, 1,000 feet northeast from No. 1 shaft, continued.

A standard-gauge railroad was constructed from Magma Junction to Superior under the direction of E. G. Dentzer, Assistant Superintendent. The narrow-gauge line was then dismantled. The standard-gauge railroad parallels the narrow-gauge location across the desert but in the hills departs from it as much as $\frac{1}{2}$ mile, eliminating the heavy grades and sharp curves.

Up to this time concentrates and ore had been hauled to Hayden for smelting. During 1922 plans for a smelter, with an estimated maximum capacity of 3,000,000 pounds of copper per month, were prepared by Messrs. Bradley, Bruff, and Labarthe. To provide bricks for the new construction, a plant capable of making 20,000 bricks per day from local clay was erected at Superior. The capacity of the concentrator was increased from 400 to 600 tons per day without enlarging the building. No ore was stoped or shipped during the year.

To finance these improvements, the company issued and sold \$3,600,000 in ten-year 7 per cent convertible gold bonds.

1923: Construction of the smelter, started in December, 1922, continued. No. 4 shaft was completed to the 1,500 level. Development was largely limited to putting the mine in shape for the proposed increased production. Stoping was resumed. The railroad construction, started in 1922, was completed, and the broad-gauge locomotive proved able to haul eight times as much freight as was formerly handled by the narrow-gauge locomotive.

1924: Little new development was carried on. The ore body had been blocked out for several hundred feet below the stopes and sufficient tonnage developed to maintain the estimated production for several years. The ground in places was heavy and difficult to keep open. Mine oxidation was rapid and lowered the percentage of extraction by the concentrator. For these reasons the policy of the company was not to develop ore reserves unnecessarily far in advance of stoping.

Stoping was carried on with an average production of 617 tons per day, as compared with the original estimate of 600 tons. The copper market recovered to an average selling price of 12.865 cents per pound.

On March 29 production from the smelter began, and it proved more efficient than originally estimated.

1925: Mr. Browning, General Manager since 1913, was succeeded by William Koerner. The 2,250 level was developed. In previous years exploratory work in the east part of the mine (east of shafts No. 2 and 3) below the 1,000 level had partly developed some ore bodies of zinc, lead, and copper. During the year this development was continued, especially on the 1,600 level.

The copper market continued to improve and the price averaged 14.007 cents per pound. As the expansion program had markedly increased efficiency of operation, the net cost exclusive of interest and depletion was 7.51 cents per pound.

1926: A new shaft, No. 5, 2,400 feet southwest from No. 3 shaft, was sunk 650 feet. The 2,550 level was actively explored and a crosscut driven south from its western end to connect with the new shaft.

A surface electric tram was constructed from the Main (500 level) tunnel portal to the mill. The aerial tram, which formerly transported ore from the Flindt tunnel portal to the mill, was abandoned.

1927: No. 5 shaft was sunk to the 2,550 level, and the crosscut west of No. 3 shaft was extended to it. At a depth of 2,150 feet the West (No. 5) ore body was encountered.

On November 24 a disastrous fire in No. 2 shaft caused the loss of seven lives. The timbering of the shaft was destroyed from the 2,550 to the 1,200 level, above which the shaft had been concreted.

The average selling price of copper was 12.956 cents, and the cost after deducting gold and silver values was 9.972 cents per pound. The average extraction by the concentrator was 98.06 per cent.

1928: No. 5 shaft was sunk to the 2,960 level. No. 2 shaft was concreted from the 1,800 to the 2,550 level. Below the 2,000 level development was limited to the west part of the mine pending the repair of No. 2 shaft. Some development was done on the East 2,000 level.

The average selling price of copper was 14.79 cents, the average cost of production 9.227 cents per pound.

1929: Because of poor ventilation in the eastern part of the mine below the 1,500 level a new shaft, No. 6, was started early in the year. It is 4,500 feet easterly from No. 3 shaft and about 1,700 feet northeast from the tunnel on the Superior-Miami highway.

In order to provide ventilation for workings west of No. 5 shaft, another new shaft, No. 7, was begun. It is about 500 feet west of the Magma concentrator and 8,350 feet west of No. 6. Sinking on No. 7 totaled 149 feet for the year.

The new West (No. 5) ore body was developed on the 2,550 level. The Main ore body was actively developed on the 2,800 level. Concreting was completed in No. 2 shaft, which was deepened to the 2,915 level.

During 329 days of operation the mine produced an average of 823 tons a day. Selling price for copper averaged 18.2 cents, and cost after deduction for gold and silver values was 9.99 cents a pound.

1930: Following the panic of October, 1929, the selling price of copper steadily dropped and averaged 12.24 cents per pound for 1930. Accordingly, production was reduced. Development was carried on in both the west and east sections of the mine. On the 2,800 level the vein in the new West (No. 5) ore body was found to be as wide as on the 2,550 level but below commercial grade.

No. 7 shaft was sunk to the 2,444 level. No. 6 shaft reached a point corresponding to the 1,576 level. No. 2 shaft was deepened to the 2,178 and No. 3 to the 2,800 level. The mine operated 303 days with 30 per cent fewer shifts than in the previous year. The concentrator worked 330 days and the smelter 319.

1931: The price of copper continued to drop and, as sold by the company, averaged 8.93 cents per pound. The cost per pound after deducting gold and silver values was 8.39 cents; this included state and county taxes but no allowance for depletion or for federal taxes. As a result, the company did little more than break even.

Stoping was discontinued from June 13 to September 16; the concentrator shut down from June 16 to September 16 and the smelter from June 22 to October 1. Development was resumed July 27.

During the year No. 6 shaft was sunk to the 2,316 level and the East 2,000 drift was extended to the shaft; this greatly improved ventilation in the east workings. Some development was carried on in the east workings below the 2,000 level. No. 7 shaft was sunk to the 2,550 level and connected with the workings there. No. 3 shaft was lowered to the 3,000 level. From No. 5 shaft on the 3,200 level a crosscut was driven north toward the Magma vein. As rock temperature was 126° F. at this point, good ventilation

became vitally necessary and more than justified the large expenditure for shafts. Development was kept to a minimum. The ores on the 2,800 and 3,000 levels were mostly pyrite and chalcopyrite, with very little bornite and lower in grade than those above.

1932: The company operated at a loss. The average price received for copper was 6.15 cents, and the cost of production was 8.39 cents per pound.

Stoping was discontinued during June 12 to December 5 and development from July 1 to September 16. The concentrator was idle from June 10 to December 15, as was the smelter for the rest of the year after June 30.

No. 3 shaft was sunk to the 3,200 and No. 6 to the 2,550 level. As developed on the 3,200 level, the main ore body averaged higher in copper than on the 3,000 level.

1933: The copper market remained depressed. In general, the company followed its previous year's schedule. Breaking of ore was discontinued from June 19 to December 3. The smelter was idle from June 18 to December 21. The concentrator closed on June 26 and worked part time during October 1 to January 1. On the east side the 2,550 level was connected with No. 6 shaft, greatly improving ventilation. A ten-day layoff was again taken at the end of the year. Development was resumed on September 1.

Copper sold by the company averaged 6.767 cents and the net cost 7.80 cents per pound.

1934: The copper market improved somewhat, and the summer shutdown was materially shortened. Stoping ceased from July 11 to August 13. The concentrator and smelter were shut down for approximately a month.

The average selling price for copper was 7.85 cents, and the average cost after deducting gold and silver values was 5.73 cents per pound. This low cost was made possible by keeping development to a minimum.

The company's annual report gave estimates of reserves in the Main ore bodies on the 3,000 and 3,200 levels.

1935: The copper market remained about the same as in the preceding year, and the company again closed down during July. Development was kept to a minimum, and most of the new workings were preparatory for stoping. Some exploratory work was done on the east side on and below the 1,800 level. A new shaft, No. 8, was begun from the surface between No. 3 and No. 5 shafts. It was designed for ventilation, to provide an upcast draft for the entire western side of the mine.

The average net selling price of copper was 7.68 cents, the cost 5.62 cents per pound.

1936: The copper market improved slightly. The mine and plant were again closed for approximately one month during the summer. The 3,600 level was actively explored by workings from No. 5 shaft. No. 3 shaft was sunk to the 3,400 level and No. 8 shaft to the 2,447 level.

The average net selling price was 9.24 cents and average cost 5.69 cents per pound.

1937: The copper market improved decidedly. Production in the mine was curtailed for about a month during the summer; the mill worked on a one-shift basis, and the smelter shut down a month during that period. The Main ore bodies were actively explored on the 3,600 and 4,000 levels. The 3,600 level showed considerable enargite and more bornite than any higher level below the 2,800. No. 8 shaft was completed to the 4,000 level.

An air-conditioning plant was installed on the 3,600 level at No. 5 shaft. It consisted of two Carrier refrigerating machines, designed to cool the stopes of the Main ore body on the 3,600 and 3,400 levels. The results exceeded estimates.

Active exploration was carried on in the east section of the mine, on and below the 2,250 level. Small amounts of copper ore were mined on the East 3,000, 2,800, and 2,550 levels. Some copper-zinc ore came from the East 2,550, 2,250, and 2,000 levels.

A new unit of 250 tons daily capacity for treatment of complex zinc-copper ore was added to the concentrator.

The average selling price for copper was 12.04 cents and the average cost 7.67 cents per pound.

1938: The average net cost of producing copper, after deducting gold and silver values, was 7.75 cents, and the average price was 9.52 cents per pound. At the end of the year 7,003,179 pounds remained unsold. The 1938 copper market was erratic, with demand concentrated in four months during which 70 per cent of the year's sales occurred. Dividends for the year amounted to \$1.50 per share.

The mine produced continuously from January 1 to July. Through July, maintenance and some development were carried on. Normal production was resumed August 1 and carried on until October 24, the effective date of the "Fair Labor Standards Act of 1938." Through the remainder of the year mine production was curtailed to approximately five days per week.

Two new Prescott Horizontal Duplex pumps were installed on the 3,600 level near No. 8 shaft. Each pump had a capacity of 600 gallons per minute pumping to the surface, a lift of over 3,200 feet.

The mill treated 246,690 tons of copper ore assaying 5.12% Cu and 32,974 tons of zinc-copper ore assaying 2.08% Cu and 8.41% Zn. The tailings averaged 0.31% Cu and the copper recovery was 95.87%.

Operations in the zinc section of the mill were carried on intermittently and much experimental work was done. The yield

amounted to 1,159 tons of zinc concentrates which assayed 49.1% Zn and 1.83% Cu.

1939: Copper sales averaged 10.655 cents a pound. Dividends were \$2.75 per share.

A Carrier centrifugal refrigerating machine was installed on the 4,000 level.

The mill treated 236,991 tons of copper ore assaying 5.22% Cu and 67,074 tons of zinc-copper ore assaying 1.99% Cu and 8.09% Zn. The tailings averaged 0.32% Cu and the copper recovery was 95.22%. Zinc concentrates, 5,155 tons assaying 48.63% Zn, were shipped.

At No. 3 shaft a new motor generator set was installed and the hoist rebuilt. This new equipment was capable of hoisting its load from 5,000 feet at a speed of 1,600 feet per minute. The old hoist had reached the limit of its capacity at 3,600 feet.

1940: Dividends totaled \$2.50 per share.

The average net cost of producing copper was 7.80 cents a pound, and the average price was 11.321 cents a pound.

On June 30, 1940, William Koerner, General Manager for sixteen years, died. He was succeeded by Edward G. Dentzer.

During July, mine operations were limited to maintenance, development, and some stoping.

In January, the Koerner vein was discovered by diamond drilling. Although idle during July, the mill during the year treated 237,003 tons of copper ore assaying 5.16% Cu and 79,044 tons of zinc-copper ore assaying 1.74% Cu and 7.31% Zn. The tailings averaged 0.29% Cu and the copper recovery was 95.61%. The 7,098 tons of zinc concentrates produced and shipped assayed 49.64% Zn.

The Mowry and Bilk claims in the Silver King district were acquired for reserve water supply. A two-compartment shaft on the Bilk claim was retimbered to a depth of 125 feet and equipped with a deep-well pump with a capacity of 200 gallons a minute.

1941: Dividends totaled \$2.50 per share.

The average net cost of producing copper, after deducting gold, silver, and zinc values, was 7.90 cents a pound. The average price received by the company on all copper produced in 1941 was 11.96 cents a pound.

Because of the urgent need for copper and at the request of a federal government agency production was increased, beginning in September, to the practical maximum of the property.

The mine was shut down during July except for maintenance, repairs, development work, and approximately 20 per cent of normal stoping. On August 4 normal production was resumed and carried on at an increasing rate until October 1, when a maximum hoisting capacity was reached. From October 1 to the

PRODUCTION OF MAGMA MINE

Year	Ore (tons)	Copper (lbs.)	Silver (oz.)	Gold (oz.)	Zinc (lbs.)	Total average grade		
						Cu (%)	Ag (oz.)	Au (oz.)
1911	129*	67,540
1912	11,214	1,104,010	64,160	201
1913	14,135	1,400,312	69,281	119
1914	18,917	2,203,604	118,310	522
1915	59,219	6,046,459	366,027	1,976
1916	93,808	8,473,580	506,523	3,173
1917	107,529	10,148,632	537,995	5,979	5.375	5.752	0.056
1918	90,321	10,968,556	463,503	5,699	6.637	5.452	0.059
1919	89,156	9,698,002	416,773	4,577	5.943	4.974	0.053
1920	100,872	8,854,917	303,258	4,437
1921	21,445	2,028,889	59,794	819	5.249	3.149	0.038
1922
1923	78,889	6,956,576	204,081	2,567	5.267	2.867	0.034
1924	222,307	23,301,511	533,204	7,589	6.115	2.797	0.033
1925	229,377	27,020,516	719,881	9,085	6.50	3.43	0.031
1926	248,787	29,135,132	860,184	9,100	6.27	2.78	0.028
1927	221,855	28,502,521	989,652	8,528	6.61	3.44	0.031
1928	263,094	35,228,810	917,048	8,665	7.19	3.42	0.034
1929	269,579	36,516,511	1,031,535	10,196	7.62	3.92	0.036
1930	251,872	31,558,508	830,009	8,519	6.90	3.31	0.033
1931	231,862	28,760,628	701,576	7,513	6.65	2.98	0.033
1932	149,010	21,675,521	486,036	5,152	7.53	3.22	0.035
1933	145,425	19,628,135	473,384	4,597	7.92	3.62	0.035
1934	264,094	31,646,576	713,712	9,100	6.54	2.93	0.034
1935	259,553	29,730,633	625,839	8,661	6.49	2.46	0.032
1936	274,065	30,280,458	552,114	7,944	6.30	2.45	0.029
1937	302,360	31,043,725	507,593	9,142	5.92	1.83	0.030
1938	334,628	32,151,163	466,660	8,650	966,548
1939	364,281	34,065,869	599,588	11,610	4,261,616
1940	378,024	34,281,249	629,417	11,307	5,992,796
1941	405,540	37,152,224	631,189	11,741	7,715,313	5.23	1.79	0.024
1942	40,390,671	547,396	12,818	7,315,034
1943	414,718	37,223,963	408,397	11,177	7,780,306	4.91	1.20	0.024

*High-grade ore shipped to smelter.

THE SUPERIOR MINING AREA

end of the year the mine produced approximately 15 per cent, on an annual basis, in excess of the production of the past two years.

Development work in the Magma mine was done on the East 3,400, 3,800, and 4,000 levels, the West 4,200 and 4,400 levels, and in the Koerner vein on the 3,600 and 4,000 levels.

The mill treated 245,885 tons of copper ore, assaying 5.26% Cu and 80,810 tons of copper-zinc ore, assaying 1.77% Cu and 8.16% Zn. The tailings averaged 0.23% Cu and the recovery was 91.12%. There were produced during the year 9,137 tons of zinc concentrates assaying 49.67% Zn.

In order to take care of the additional output of the Magma mine, the amount of gold and silver custom ore treated by the smelter during the last half of the year was reduced to approximately 25 per cent of the amount treated during the first half of the year.

The labor situation took a decided turn for the worse. High wages in defense plants attracted many people. Many of the younger men entered military service. It became necessary to train inexperienced help. Wages were the highest since the first World War, but production per man-shift dropped steadily during the year.

MINERALIZATION

Summary: The ore bodies of the Magma mine occur as replacements of crushed wall rock within two fault zones of the east-west system. Those of the Magma vein, in the Magma fault, constitute by far the greater proportion of the tonnage extracted or developed. The ore is not continuous throughout length and depth but consists of several distinct shoots separated by barren vein material.

The Main ore body is the largest. It is contained laterally between a vertical north-south plane 500 feet east of No. 3 shaft and the Main fault. It is developed vertically from the 400 to the 5,000 level as measured below the collar of No. 1 shaft.

The West or No. 5 ore body is in a faulted segment, possibly of the Magma vein, west of the Main fault.

The East ore bodies, in the Magma vein east of shafts No. 2 and 3, are of limited dimensions. As developed, they extend from the 1,400 to the 3,000 levels.

The only ore body of the mine not in the Magma fault is within the Koerner fault, nearly parallel to and about 1,200 feet south of the Magma fault. The Koerner fault is of small displacement and without surface expression. The ore body has its apex apparently below the 3,200 level and extends vertically downward to the 4,200 level or lower. At present it is only partially explored.

The ores of the Main, West, and Koerner ore bodies are similar mineralogically. Copper is the principal ore metal, but silver and gold are recovered as by-products in smelting. The principal ore minerals are pyrite, bornite, chalcopyrite, and enargite, with

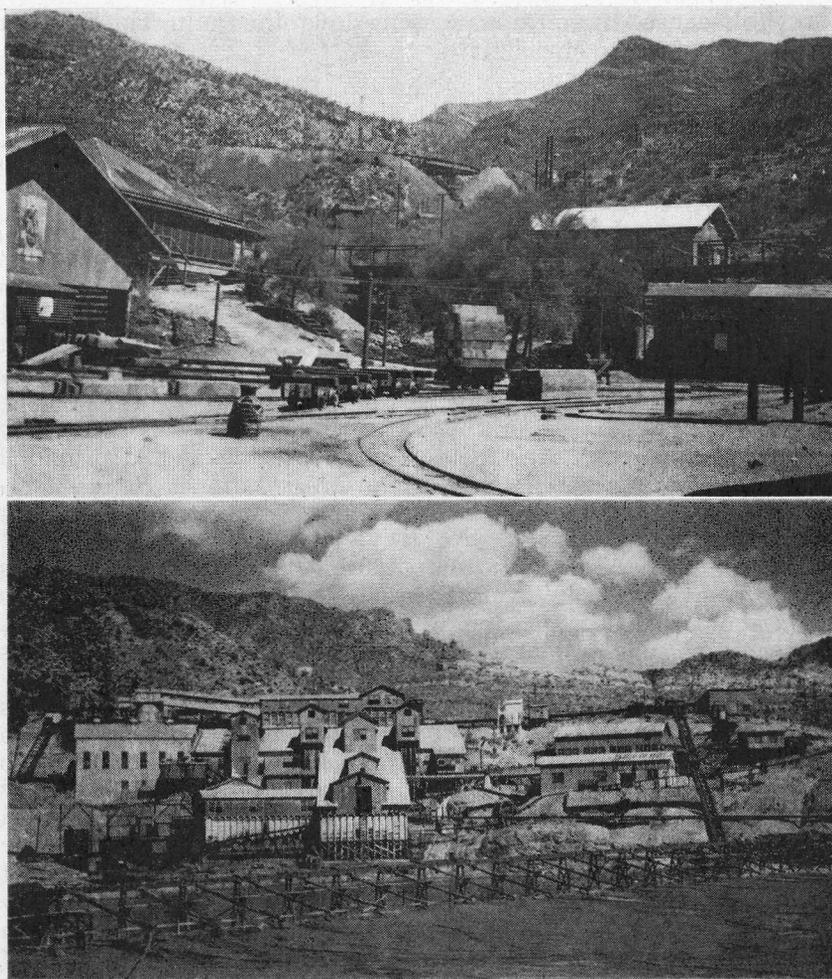


Plate XI.—A, Magma mine from main portal; B, Magma concentrator.

subordinate tennantite and hypogene chalcocite. Some sphalerite occurs in the upper eastern portion of the Main ore body, especially from the 900 to the 1,200 level. Below this, sphalerite occurs sparsely in small bunches. A little galena accompanies sphalerite but insufficient to be recovered as a by-product in concentration.

Sphalerite is the predominant mineral in the upper and more easterly, and chalcopyrite in the lower and more westerly, of the East ore bodies.

Supergene enrichment is important in the Main ore body above the 800 level. In the West ore body it formed a small tonnage, although locally intense near the Main fault. There the enrichment is not related to the upper surface of the ore body but was

probably caused by solutions seeping down the fault. The Koerner ore body shows no enrichment.

Main ore body: Considering the size of the Magma ore bodies, the outcrop of the Magma vein is inconspicuous. Above the Main ore body the bleached, faulted porphyry dike is stained by oxidized copper and iron minerals with locally small masses of residual chalcocite. (Pl. X B).

The Main ore body has its apex a short distance above the 400 level and extends to the lowest workings. Most of the production of the mine has come from this Main ore body or shoot, which has been mined almost continuously below the 900 level. Above that level the ore was bunched. The Main ore shoot pitches west at about right angles to the sedimentary beds. The reasons for its localization are not clear. This portion of the vein was replaceable by the copper-bearing solutions either because of its permeability or because some condition of premineral faulting had provided passage for a greater portion of the ore solutions. Relative replaceability of the wall rocks affected somewhat the lateral extent of this ore body.

The Main ore body above the 1,500 level consists of two shoots which join near that level (Pl. II). The western shoot has its apex about 50 feet above the 1,200 level at 4,600 departure. Here the ore consists of sphalerite and a little galena, with only traces of copper. Between the 1,300 and 1,400 levels it changes abruptly into a bornite-rich ore with little or no zinc and lead. In levels above the schist the width of the Main ore body ranges from 5 to 40 feet. Where the vein is wide, the ore generally occurs in two or more rich stringers separated by poorer vein material, partly below commercial grade. For example, on the 2,000 level a rich stringer occurred near the hanging wall and another one near the footwall. Some selective mining was practicable, but in most places the vein was mined for its full width.

Complete data of stoping length, width, and grade for the Main ore body on various levels are not available, but the following are given in annual reports of the Magma Copper Company.

Level	Length	Av. width	Cu (per cent)	Ag (oz.)	Au (oz.)
1,000	300-500	5-8
1,600	1,300	12.5	(6.00	3.2	0.04)*
1,800	1,550	20.8	5.72	2.64
2,250	1,050	27.5	8.3	3.94	0.025
3,000 (west part)	300	10.2	4.25	0.9	0.01
3,000 (east part)	735	13	5.9	1.1	0.01
3,200 (west part)	750	8	5.1	0.67	0.01
3,200 (east part)	500	13.8	4.2	0.71	0.01
3,400	1,498
3,600	1,540
4,000	2,260
4,400	1,800

*Figures for 1,000 feet of stoping length.

Below the 4,400 level the ore body is not fully developed, but exploration to date indicates the ore to be less continuous than on the 4,000 and higher levels.

West ore body: The West ore body is in a faulted segment, possibly of the Magma vein, west of the Main fault. It was discovered accidentally. In sinking No. 5 shaft, a small stringer of ore was found at the 2,150 level. Followed westward, this stringer developed into a commercial body. The West ore body is in a vein that continues above, below, and west as far as explored. The vein strikes nearly east and dips steeply north. The apex of the ore body is at the 2,100 and its bottom at the 2,675 level. The maximum stoping length, 750 feet, is on the 2,550 level. Its average stoping length is about 250 feet, its average width about 15 feet, and its average copper content about 7 per cent.

Bornite is the predominant copper mineral, with subordinate chalcopyrite, tennantite, hypogene chalcocite, and sphalerite. Enargite is absent. Oxidation and supergene enrichment have been intense in places, especially on the 2,550 level near the Main fault. Here some specimens consist of massive steely supergene chalcocite with some malachite, cuprite, and native copper.

Prior to faulting, the West ore body was several hundred feet higher than it is at present.

There is no well-defined enrichment zone paralleling the upper surface of the ore body, but nearly all specimens show at least some supergene chalcocite. In comparison with the weight of hypogene copper minerals, the supergene minerals are relatively unimportant. The proximity of oxidation to the Main fault suggests that oxidation and enrichment are due to solutions seeping down the fault.

The West ore body is not a faulted segment of the Main ore body, as the latter is nowhere in contact with the Main fault. On the 2,800 level the Main ore body is within 50 feet of the fault, but with each succeeding deeper level the fault flattens and diverges farther from the Main ore body. The West ore body is now nearly mined out.

East ore bodies: The East ore bodies lie east of zero crosscut and include zinc-copper in the upper levels and copper in the lower. Sphalerite predominates above the 2,550 level and chalcopyrite below. The ore in the eastern portion of the mine is not continuous, and none of the known ore shoots persists for over several hundred feet. No structural reason can be given for the scarcity of ore east of zero crosscut, and no apparent controlling structural features join the various isolated bodies. It can only be assumed that in contrast to the Main ore body the mineral-carriers and the forces impelling these carriers were not sufficiently strong to form continuous ore. Sulfides were deposited only where permeability and conditions for replacement were favorable.

The possibility of ore in the lower levels east of zero crosscut is indicated. On the east 3,400, 3,600, 3,800, and 4,000 levels ore has been found that has no continuation to levels directly above or below.

Koerner vein: The role that chance may play in the discovery of new ore bodies was demonstrated in the case of the Koerner vein. In January, 1940, a diamond drill hole was driven horizontally from the 4,000 level. It was not drilled primarily to seek ore but to develop water for the lower levels. The drillers, directed to drill southerly, lined up the hole in such direction as to give maximum space for removing drilling rods. This direction happened to be S. 10° W. At a distance of approximately 1,225 feet the hole passed into an ore body in the Koerner vein. This vein has proved to be spotty. Had the hole been driven due south it would have encountered the vein in a barren part, and further exploration of it might have been delayed indefinitely.

The Koerner vein is identical with the Koerner fault. The ore body is similar to the Main ore body, but smaller. The company's annual report for 1941 states that on the 4,000 level a total of 1,800 feet of drifting, of which approximately 900 feet were in ore, had been done. The average width was about 9 feet and the average grade somewhat better than 5 per cent copper. On the 3,600 level, drifting on the vein has totaled 1,797 feet, of which approximately 950 feet are in ore 4 to 15 feet wide and averaging about 5 per cent copper. On the 3,200 level, the vein has been cut by diamond drills at three places above favorable areas on the 3,600 level. As none showed ore bodies, the 3,200 level may be above the productive part of the vein. On the 4,400 level at No. 14 crosscut, the vein is below commercial grade. One raise on the vein from the 4,000 to the 3,600 level and another on the vein from the 4,000 to the 3,800 level showed that the ore is not vertically continuous (Fig. 11).

Mineralogically, ore of the Koerner vein is indistinguishable from that of the Main ore body on the same levels. Bornite and enargite are the predominant copper minerals, with subordinate chalcopyrite and tennantite and minor hypogene chalcocite. The difficulty of ventilation at present when refrigeration machines are not available retards exploration and mining of the Koerner vein. This vein is, however, an important reserve for future operation.

A diamond drill hole driven south from zero crosscut on the Koerner vein traversed 2,300 feet of diabase and ended almost directly below Queen Creek. No ore was encountered.

FAULTING

The ore bodies of the Magma mine are associated with east-west fault fissures, as are other ore bodies of the Magma and

Belmont subareas. Postore faults of northwest to northeast strike have displaced the east-west fissures.

PREORE FAULTS

Magma fault: The Magma fault, in which occurs the Magma vein, has the largest displacement of the east-west faults. It can be traced on the surface for about 3,300 feet, limited on the west by the Main fault and on the east by overlying dacite. Underground it has been followed for almost 7,000 feet, with a width of zone ranging from less than 1 foot to more than 50 feet.

Above the 2,250 level of the mine the Magma fault is approximately east-west, but below that level it strikes nearly N. 80° E. Locally, its strike varies from N. 65° E. to N. 80° W. Its average dip from the surface to the 900 level is 65° N., and below that level it is 78° S. (Fig. 6); local changes in dip are common.

Movement on the Magma fault did not take place along a single fracture but along a zone of closely spaced fractures. Shattering near the vein is slight, and the walls commonly are definite, especially in diabase. On the lower levels in Pinal schist the fault forms a wide shear zone whose walls are not easily determined. The apparent horizontal movement of the Magma fault near the Main ore shoot averages about 475 feet, with the north side moved east in relation to the south side. There are indications that movement decreased with depth and increased eastward (Fig. 7). Offsetting by the fault in the upper levels was described by Ransome as follows:³⁸

An observer walking east on a level driven along the vein has on his right hand rocks higher in the stratigraphic column than those on his left. Moreover, in consequence of the general easterly dip of the beds, after he first sees limestone on the right hand he must continue for 400 to 450 feet before he reaches the same stratigraphic horizon on the left.

Formations south of the fault are about 500 feet below those north. This movement was not vertical but oblique (Fig. 9).

A line joining the bends in the vein at the 650 and 900 levels is assumed to be the direction of the fault movement. This axis inclines westward, at an angle of about 55° to the dip of the beds. It is believed that if movement had been vertically downward, the rocks adjacent to the fault would have been more extensively shattered and fractured. In places the fault is clean-cut, sharp, and narrow and in the absence of shattering, difficult to follow.

The indicated direction of movement is only approximate, as the Magma fault is a warped surface and the axis of its bend is not likely a straight line.

The major movement was along the footwalls and hanging walls, and in places they show considerable gouge. Some of the

³⁸Ransome, F. L., Copper deposits near Superior, Arizona: U.S. Geol. Survey Bull. 540, p. 145. 1912.

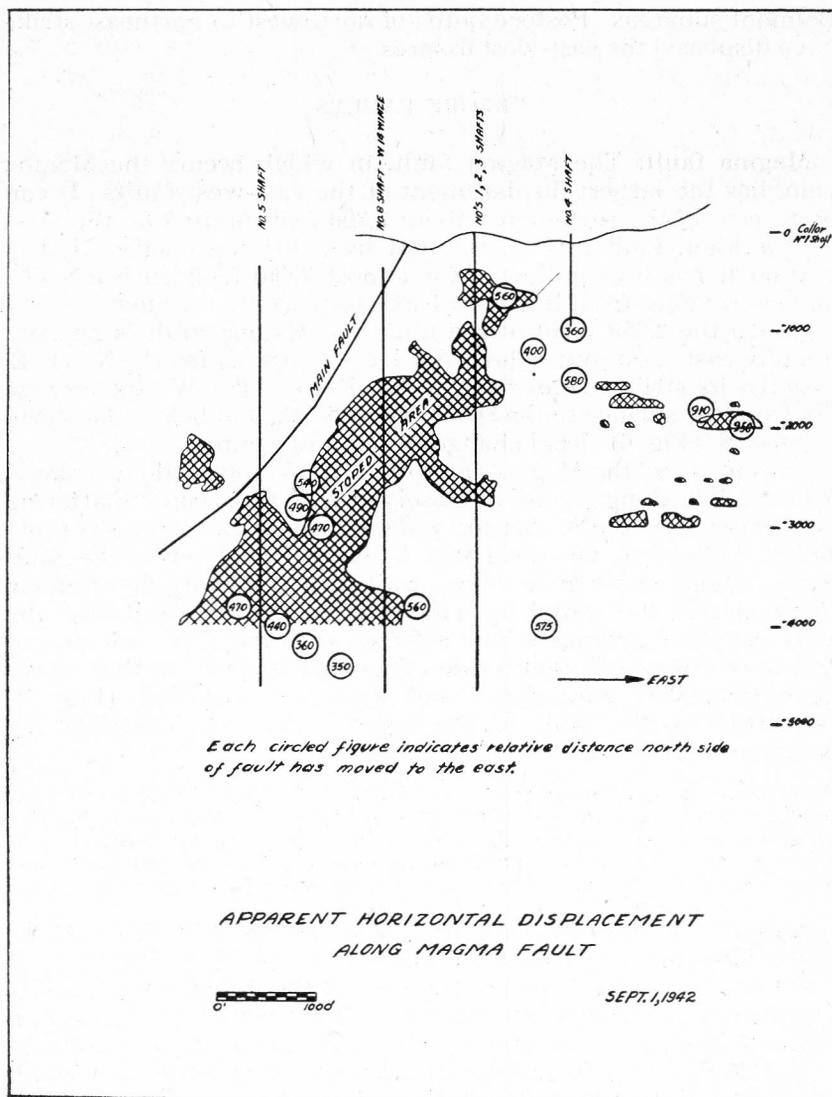


Figure 7.—Apparent horizontal displacement along Magma fault.

movement within the fault zone was postore, but probably most of it was contemporaneous with movement along the walls.

With depth, particularly in Pinal schist, the walls of the fault zone tend to flare. The zone is wider, the walls less defined, and horses of relatively unbroken, unmineralized wall rock are common.

The age of initial movement along the Magma fault is post-Upper Pennsylvanian (post-Naco limestone) and predacite. Prob-

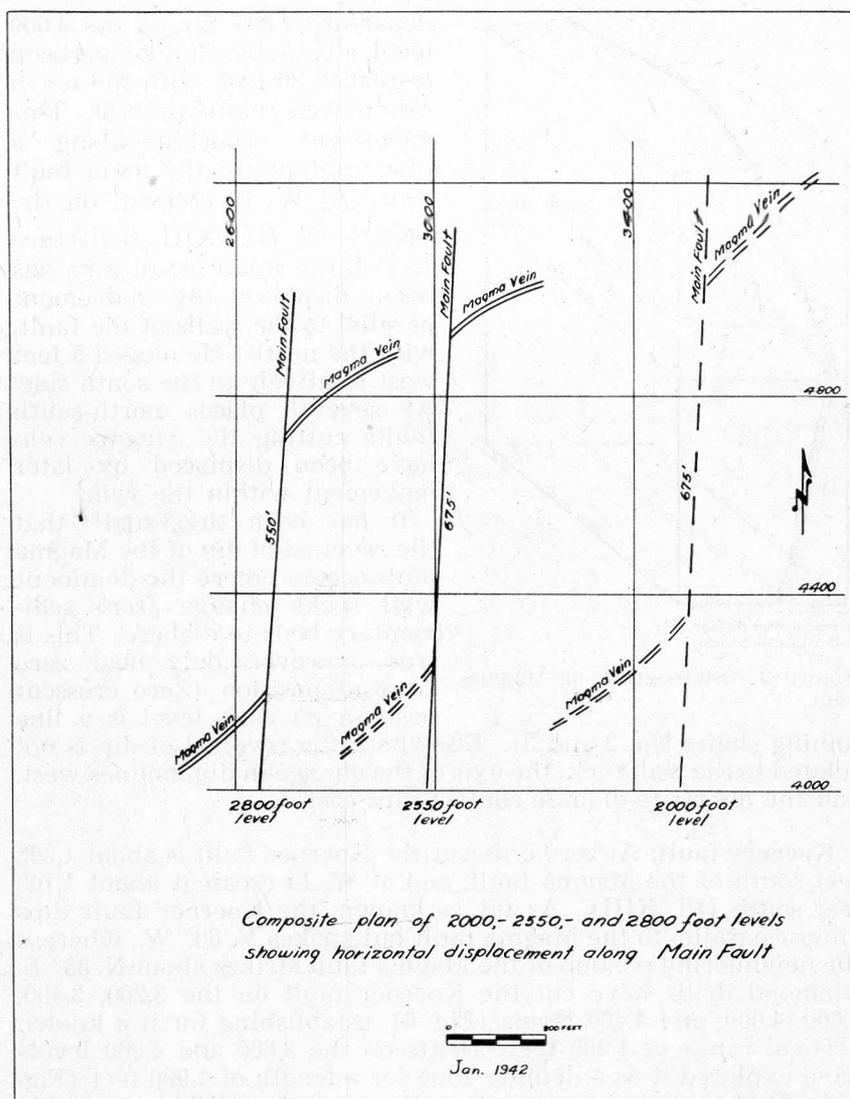


Figure 8.—Composite plan showing horizontal displacement along Main fault.

ably the forces causing it³⁹ and the other east-west faults of the region were connected with the emplacement and cooling of the Central Arizona batholith in early Tertiary (Laramide) time. Additional local movement within the Magma fault zone occurred at a much later time. Basalt dikes later than the dacite are displaced within the vein on planes of movement generally subparallel to the walls of the fault. Near W. 32 crosscut (Pl. XIII,

³⁹Short and Ettlinger, op. cit., p. 185.

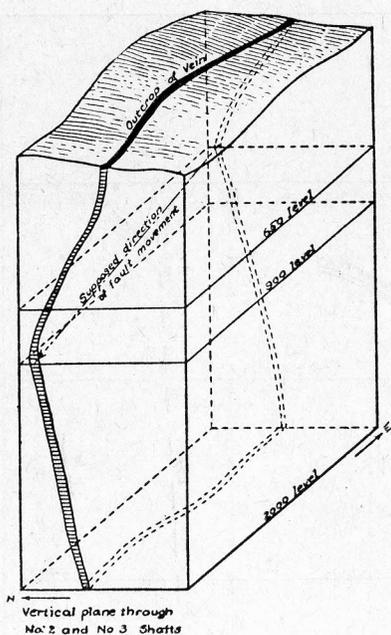


Figure 9.—Stereogram of Magma vein.

joining shafts No. 2 and 3). Elsewhere the reversal of dip is not related to the wall rock; the axis of the change in dip inclines west, and the quartzite-diabase contact dips east.

Koerner fault: At zero crosscut the Koerner fault is about 1,300 feet south of the Magma fault, and at W. 14 crosscut about 1,100 feet south (Pl. XIII). As far as known, the Koerner fault dips almost parallel to the Magma fault but strikes N. 80° W., whereas the neighboring portion of the Magma fault strikes about N. 85° E. Diamond drills have cut the Koerner fault on the 3,200, 3,400, 3,600, 4,000, and 4,400 levels (Fig. 6), establishing for it a known vertical range of 1,200 feet. Drifts on the 3,600 and 4,000 levels have explored it as a definite zone for a length of 1,900 feet (Fig. 11). At the eastern and western limits of the drift on the 4,000 level its zone narrowed to 1 to 4 feet.

The outcrop of the Koerner fault has not been definitely determined. Several small faults crop out on the ridge south of No. 3 shaft, and one of them may be the Koerner fault. The displacement along these faults does not exceed 25 feet, and they can be traced for a length of only about 1,000 feet. Below these outcrops a group of faults striking approximately east cut the 500 haulage

departure 2,825 E.) on the 4,000 level, a large basalt dike has been displaced 20 feet, with the north side moved relatively east. This movement occurred along a plane oblique to the main fault zone. At W. 10 crosscut on the 4,000 level (Pl. XIII, departure 4,515 E.), a small basalt dike has been displaced by movement parallel to the walls of the fault, with the north side moved 5 feet west relatively to the south side. At several places north-south faults cutting the Magma vein have been displaced by later movement within the vein.

It has been suggested⁴⁰ that the reversal of dip of the Magma fault occurs where the dominant wall rocks change from sedimentary beds to diabase. This is true, however, only near zero crosscut position (Zero crosscut position on each level is a line

⁴⁰Short, M. N. and Wilson, E. D.: Magma mine area, Superior, Arizona (in *Some Arizona Ore Deposits*): Arizona Bureau of Mines Bull. 145, p. 93, 1938.

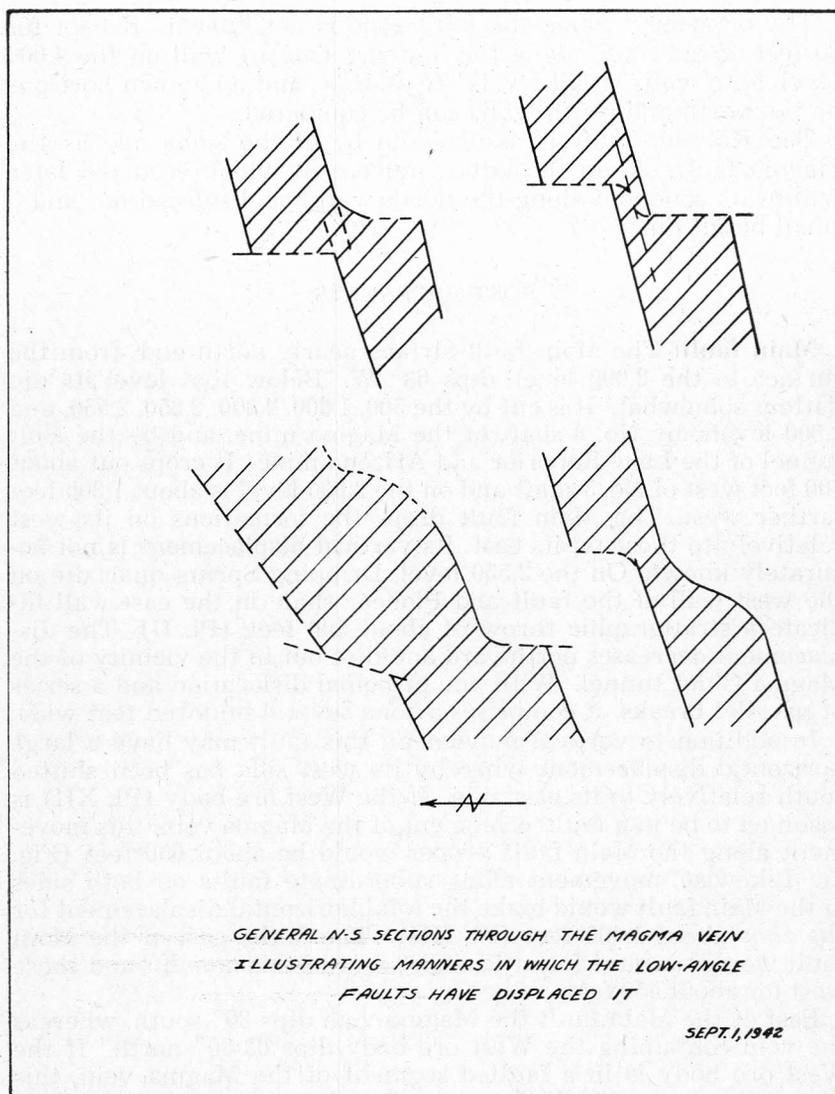


Figure 10.—General north-south sections through Magma vein, showing low-angle faults.

level a short distance south of No. 3 shaft. Two strong faults in E. 19 crosscut on the 1,600 level projected to zero crosscut strike close to the upward projection of the Koerner fault. The known dip of the Koerner fault in the lower levels, the faults projected from E. 19 crosscut on the 1,600 level, and the faults cutting the 500 level all indicate the outcrop to be about 1,000 feet south of the Magma outcrop.

The movement along the fault zone is not known. Except for 80 feet of quartzite along the hanging (south) wall on the 4,000 level, both walls on all levels are diabase, and no known horizons on the north and south walls can be compared.

The Koerner fault is assumed to be of the same age as the Magma fault. As in the latter, movement which occurred later within its zone and along the north wall has displaced ore and a small basalt dike.

POSTORE FAULTS

Main fault: The Main fault strikes nearly north and, from the surface to the 2,000 level, dips 63° W. Below that level its dip flattens somewhat. It is cut by the 500, 1,800, 2,000, 2,250, 2,550, and 2,800 levels, by No. 8 shaft of the Magma mine, and by the Holt tunnel of the Lake Superior and Arizona mine. It crops out about 700 feet west of No. 3 shaft and on the 2,800 level is about 1,800 feet farther west. The Main fault drops the formations on its west relatively to those on its east. Its vertical displacement is not accurately known. On the 2,550 level, Dripping Spring quartzite on the west wall of the fault and Pioneer shale in the east wall indicate a stratigraphic throw of about 500 feet (Pl. II). The displacement decreases northward and dies out in the vicinity of the Magma Chief tunnel. With one principal dislocation and a series of parallel breaks, it comprises a zone several hundred feet wide.

In addition to vertical movement, this fault may have a large horizontal displacement whereby its west side has been shifted south relatively to its east side. If the West ore body (Pl. XII) is assumed to be in a faulted segment of the Magma vein, this movement along the Main fault proper would be about 650 feet (Fig. 8). Likewise, movement along subordinate faults on both sides of the Main fault would make the total horizontal displacement for the zone about 1,400 feet (Pl. XII). The faults east of the Main fault would account for at least 300 feet of this amount and those west for about 450 feet.

East of the Main fault the Magma vein dips 80° south, whereas the vein containing the West ore body dips $68-80^{\circ}$ north. If the West ore body is in a faulted segment of the Magma vein, this change in dip might indicate rotation from a point about 2,500 feet north of the Magma vein or approximately where the Main fault dies out. Another possibility is that the West ore body, if part of the Magma vein, may have been above the bend in dip of the vein (p. 79) prior to displacement by the Main fault.

East of the Main fault, the Paleozoic and Apache beds dip approximately 30° east and are relatively undisturbed except by the Magma fault. In contrast, the area between the Main and Concentrator faults, where not covered by dacite, reveals a mosaic of fault blocks. Locally, remnants of downfaulted dacite indicate postdacite faulting, presumably of the same age as the

Main fault. The northward trend of the sedimentary beds is still apparent but very irregular.

Prior to exploration in lower levels, the Concentrator fault was regarded as subordinate to the Main fault, but it is now known to be the principal dislocation of the district. The latter is a branch of the former, and their surface junction, concealed by alluvium, is probably near the Superior High School.

Transverse faults east of Main fault: The Magma vein has been displaced by several faults. Their strikes range from N. 85° W. to N. 70° E. but mostly are between N. 60° W. and N. 45° E. Their dips range from 45° W. to vertical. Their displacements are various, with a maximum of 120 feet. For the majority of these faults, particularly near the Main fault, the west side moved south.

Several of the faults are designated by letter on Plate II. Faults A, B, C, and D, at or near the western limits of the Main ore body, are described as follows:

Fault	Av. strike	Av. dip	Vertical range (levels)	Relative movement
A	N. 30° E.	80° W.	3,800-4,400	West side 35 ft. S.
B	N. 15° E.	75° W.	2,800-4,400	West side 20± ft. S.
C	N. 30° E.	50° W.	1,200-3,200	West side 15± ft. S.
D	N. 20-35° E.	45° W.	Surf.-2,550	West side 40± ft. S.

These four faults in echelon form a continuous zone from the surface to the 4,400 level.

Except for faults C and D, the persistent cross faults west of zero crosscut are limited to the lower levels of the mine. There are numerous faults above the 3,000 level, but none has been traced for more than 100 to 200 feet.

The faults east of zero crosscut appear to be more persistent than those on the west side. Faults H and J can be traced with reasonable certainty from the 4,000 to the 3,200 level and with less certainty to the 1,800 level. Other faults in upper levels of the east side may continue over long ranges, but, as in the case of faults H and J, definite evidence of them is missing on a few levels.

In some places, a fault can be followed through stopes from one level to the next. Generally, however, the faults were projected from level to level entirely on their characteristics of movement, dip, and strike. The dip of a fault plane, as seen in a drift, however, may be entirely different from its general dip over a 1,000-foot range. Its strike also may vary from place to place. The relative direction and amount of movement is the most important criterion used in connecting faults exposed only on the main levels.

A summary of data for faults E, F, G, H, and J is given in the following table:

Fault	Av. strike	Av. dip	Vertical range (levels)	Relative movement
E	N. 15° E.	65° W.	3,200-4,400	West side 15 ft. S.
F	N. 55° W.	75° E.	3,600-4,400	West side 20 ft. N.
G	N. 30-55° W.	85° E.	3,000-4,400	West side 25-100 ft. N., increases with depth
H	N. 45° W.	75° W.	3,000-4,000	West side 120 ft. N.
J	N. 5° W.	70° W.	3,200-4,000	West side 5 ft. N.

Faults E, F, and G are fairly close together, with F apparently cut off by G just above the 4,400 level and cut off by E between the 3,400 and 3,600 levels. Faults E and G may intersect near the 3,000 level.

Between the 1,500 and 1,800 levels east of zero crosscut is a fault striking N. 75° W., nearly parallel to the Magma vein, in which the east side has been displaced for 120 feet.

Very few other faults mapped show movement of over 5 to 10 feet. Many of them are found only on one level, although five or six show vertical ranges of 300 to 500 feet. The faults seen only on one or two levels have an average dip of about 75° W., but their strikes range from N. 60° W. to N. 60° E.; they are markedly similar in dip, and less noticeably similar in strike, to the major transverse faults. These smaller faults are more numerous in the west side of the mine, in or near the Main ore body. Of the faults mapped on the various levels, only about one in five is strong enough to continue to the next level. In many places faults offsetting the vein as much as 15 or 20 feet are not found in drifts only 50 to 100 feet away.

As the Main fault is approached from the east, the smaller as well as the larger transverse faults show definite relationship to the Main fault. It is most striking as to direction of movement. For all of them, as well as for the Main fault, the west side has moved south with reference to the east side. Most of the subordinate faults strike northeast, making an angle of about 45 degrees with the Main fault. The movements along these subordinate faults vary up to about 100 feet, and the total movement along them is about 300 feet. The direction of movement along the fault planes could not be definitely determined, but an oblique movement is indicated. In several places where the vein has been offset, ore is found on one side and low-grade vein on the other of the fault. At these places it is common to find drag ore on only the side of the fault next to the ore. A simple horizontal movement, except under very unusual conditions, could not cause this difference. An oblique movement, on the other hand, would have a horizontal component, and a drift following the vein and fault would encounter drag ore only next to the ore-bearing vein segment. Such conditions have been encountered in the Magma

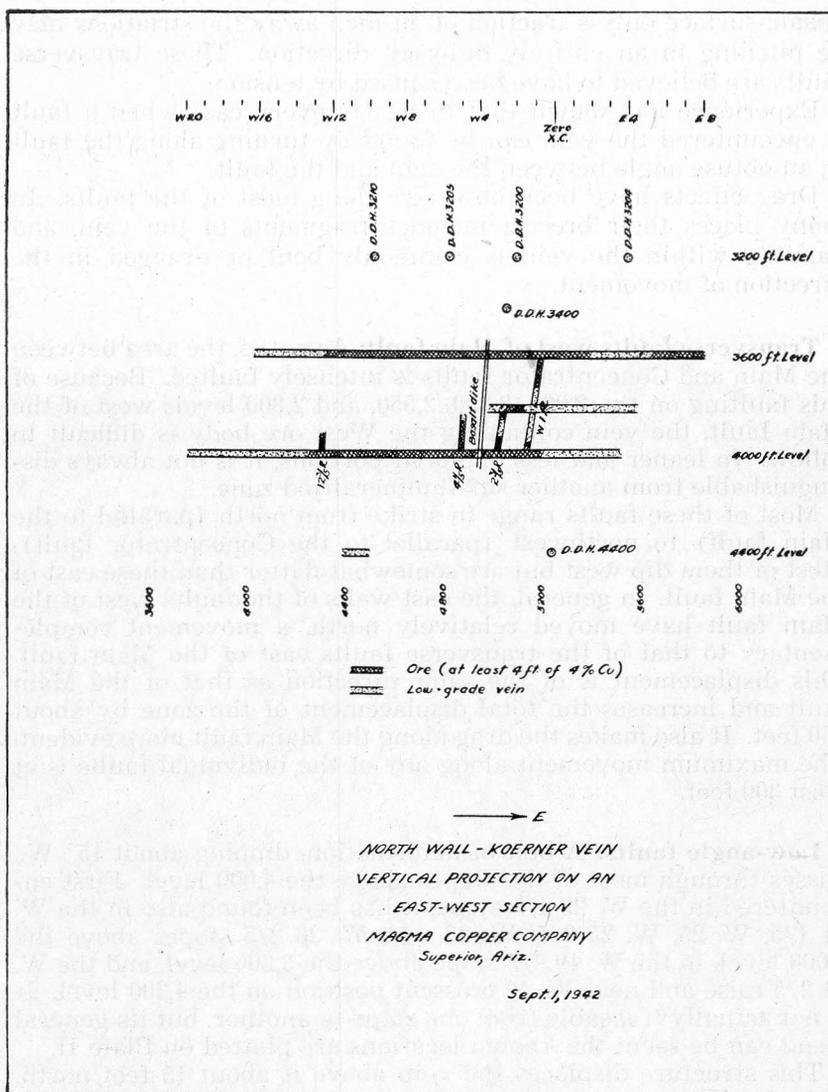


Figure 11.—Longitudinal projection of north wall, Koerner vein.

mine. A vertical movement could give the same effect, but to offset the vein 100 feet would require a vertical movement of about 400 feet, which seems excessive. Many striations on slickensides pitch down the fault plane at a small angle from the horizontal, but they cannot be relied upon as definite evidence of directions of faulting. Movement has taken place on a series of parallel planes rather than on a single plane and, although on one plane the striations may be in one direction, on the next slick-

inside surface only a fraction of an inch away the striations may be pitching in an entirely different direction. These transverse faults are believed to have been caused by tension.

Experience has shown that in nearly every case when a fault is encountered the vein can be found by turning along the fault in an obtuse angle between the vein and the fault.

Drag effects have been observed along most of the faults. In many places their breccia includes fragments of the vein, and banding within the vein is commonly bent or dragged in the direction of movement.

Transverse faults west of Main fault: As noted, the area between the Main and Concentrator faults is intensely faulted. Because of this faulting on the 2,000, 2,250, 2,550, and 2,800 levels west of the Main fault, the vein containing the West ore body is difficult to follow. In leaner and less explored portions, it is not always distinguishable from another small mineralized zone.

Most of these faults range in strike from north (parallel to the Main fault) to northwest (parallel to the Concentrator fault). Most of them dip west but are somewhat flatter than those east of the Main fault. In general, the east walls of the faults west of the Main fault have moved relatively north, a movement complementary to that of the transverse faults east of the Main fault. This displacement is of the same direction as that of the Main fault and increases the total displacement of the zone by about 500 feet. It also makes the drag along the Main fault more evident. The maximum movement along any of the individual faults is at least 300 feet.

Low-angle faults: A belt of deformation, dipping about 15° W., passes through most of the stopes above the 4,000 level. First encountered in the W. 23 $\frac{4}{5}$ stope, it has been found also in the W. 24 $\frac{1}{5}$, W. 25, W. 25 $\frac{2}{5}$, W. 28, and W. 28 $\frac{2}{5}$ stopes above the 4,000 level, in the W. 19 $\frac{4}{5}$ stope above the 3,800 level, and the W. 34 $\frac{2}{5}$ raise and near W. 40 crosscut position on the 4,200 level. It is not actually traceable from one stope to another, but its general trend can be seen; the known locations are plotted on Plate II.

This structure displaces the vein above it about 15 feet north. This displacement is accomplished in places by a definite fault through either the hanging or footwall and in other places by a sharp fold. In no place have both walls been affected in the same manner. Figure 10 shows several ways in which the offsetting has taken place. In places the flat faults have been offset and dragged by movement parallel to the Magma fault. The later transverse steep angle faults have displaced this belt and in a plane parallel to the Magma fault have the effect of vertical movement.

This flat-dipping structure, for want of a better name, is referred to as "low-angle faults." Consisting partly of faults and partly of a fold or roll, it may represent a sharp fold in the vein which,

under stress of deformation, produced faulting in places; or it may be a fault in which the vein took up some of the movement by folding in one wall instead of by faulting in both. Whatever the actual mechanics of the movement, there is a zone in which the vein has been moved horizontally. It has not been possible to mine these portions in a normal manner.

This zone of deformation has not been found east of W. 19 2/5 raise above the 3,800 level.

RELATION OF FAULTING TO ORE

Only two faults in the Magma mine, the Magma and the Koerner, are known to be preore. Movement continued after the initial displacement and offset some of the preore movement, but the period in which it began is not clear. There appears to be relationship between the ore and some of the transverse faults, which fact suggests that these faults influenced ore deposition.

As shown on Plate II, faults A, B, C, and D bound the western extremity of the Main ore body and indicate a continuation of its westerly pitch. In only a few isolated stopes has any mining been done between these and the Main fault. In addition there appear to be more transverse faults cutting the ore bodies than the barren vein. If the faults are postmineral, these relations are purely coincidental and had no bearing on ore deposition. If the faults are premineral and all evidence of premineral movement is destroyed, perhaps by postmineral movement, these relations are then important and can be used as guides in prospecting. At present the age of some of these faults must remain in doubt. The broad relations certainly indicate premineral origin, but actual evidence favors postmineral inception. The possibility of premineral faulting should be kept in mind when prospecting is undertaken near these faults.

The proximity of the Main fault to the Main ore shoot suggests for it a possible premineral origin. This assumption would be logical if faults A, B, C, and D are premineral, as they appear to be related to the Main fault. To suggest that the Main fault might be premineral raises several questions, especially in its relation to the Concentrator fault which has such a large postmineral displacement. With the information now available these questions cannot be answered, and no attempt can be made to prove a premineral origin for the Main fault.

FOLDING AND TILTING

Small-scale folding is represented only by flexures between faults. The rocks tended to deform by breaking rather than by bending.

East of the Main fault the predacite beds have been tilted approximately 30° E. Between the Main and Concentrator faults

they also have been tilted, but the tilting has been greatly modified by smaller faults.

The mechanics of tilting cannot be adequately explained until the detailed structure of a larger region surrounding the Superior district has been worked out. As the tilting preceded the dacite eruption, and the Main and Concentrator faults are postdacite, these faults cannot be regarded as the cause unless a predacite initial movement for them is assumed. The eastward-tilted beds may represent a limb of a large broken fold, and the preore east-west faults may be accompanying tear breaks.

ORE DEPOSITION

Processes: Replacement was by far the most important process in deposition of the Magma ore. This process may be defined as the dissolving of a mineral or group of minerals and the simultaneous deposition of another mineral or group in its place. Solution and deposition ordinarily proceed concurrently without intervening development of appreciable open spaces, and the substitution commonly involves no change in volume. A second, but distinctly minor, process of the deposition of Magma ores is deposition in open spaces. Filling of open fractures did not occur, and this second process is indicated only in filling of small vugs. In places the process of ore deposition has occurred in two stages, solution and deposition, leaving small irregular cavities. These were lined with quartz crystals, and the spaces between were later filled with sulfides.

RELATION OF ORE TO WALL ROCKS

Quartz monzonite porphyry: From the surface to the 1,200 level a dike of quartz monzonite porphyry occurs within the Magma fault zone. Below that level the dike in many places forms either the north or south wall of the vein, and on the 4,400 level the vein passes without displacement through a porphyry dike. The dike is later than the Magma fault but earlier than the mineralization. The ore is probably genetically related to the quartz monzonite porphyry, in that the dike represents offshoots of the igneous mass which furnished the ore solutions. From the levels now open for inspection it is evident that the porphyry dike was a poor host to the ore solutions. In several places the dike is rather strongly mineralized but not sufficiently to constitute ore.

Pyrite is the most common sulfide in the dike. It lacks copper mineralization probably because it was intruded after the major movement and accompanying brecciation of the country rock. There has been but slight fracturing of the dike, and the ore solutions had little chance of penetrating and replacing the tight porphyry.

Diabase: As shown on Plate II, there is a definite relation between the diabase and the shape of the Main ore shoot. All pronounced increases in stope length of the Main ore body are where both walls are diabase, and most of the maximum increases are nearly parallel to the contact between diabase and sedimentary beds. This increase in stoping length is most noticeable in the lower diabase sill. Probably because of certain physical characteristics, the diabase is more favorable for ore deposition than the sedimentary rocks, particularly the lower quartzite and shale. Undoubtedly the brecciated diabase was more susceptible to replacement by the ore-bearing solutions than was the highly siliceous quartzite and shale.

In 1941, Kuhn made a detailed study of the relation between ore bodies within the Magma vein and various types of diabase that form a large part of the vein walls.⁴¹

All specimens were taken from the north or footwall side of the Magma vein. In general, the footwall drifts were best for systematic collecting. Since thin sections of highly altered rocks contain few primary minerals and show little of the original texture, specimens with minimum alteration were selected wherever possible. The specimens were collected below the 1,200 level at about 400-foot intervals, both horizontally and vertically, and any irregularity in distribution or type was taken into consideration. A total of 68 specimens, mostly from the footwall drift about 75 feet north of the Magma vein, were collected.

This study of the diabase proved that the locations of ore shoots within the Magma vein have in no way been influenced by variations in type of diabase.

Although no difference in the replaceability of the various types of diabase could be noted, it is quite evident that the stope length is greater in the lower olivine-diabase sill than in the upper quartz-diabase sill. In view of Kuhn's investigation, however, this difference may more logically be ascribed to some other feature, probably the presence of Apache beds above the lower sill rather than to a difference in the types of diabase.

Basalt: The basalt was intruded much later than deposition of the vein and hence did not influence ore deposition. The effect of the basalt in the mineable portions of the vein is to dilute and spread apart the ore. The wider dikes, which in most places cut the vein at about right angles, have pushed the ore apart and replaced it with barren basalt. The smaller dikes must be mined with the ore, which causes dilution.

Sedimentary rocks: The Pioneer shale, Barnes conglomerate, and Dripping Spring quartzite are highly siliceous, and for this discussion can be grouped as one unit. Plate II shows that on the

⁴¹Kuhn, Truman H., Private report to the Magma Copper Co., 1941.

lower levels the stoping length of the Main ore body is shortest where the Apache group forms one or both walls of the vein. There also appears to be a difference mineralogically between the sulfides in the vein where the walls are formed by the Apache group and where they are diabase. On the 2,675, 2,800, 3,000, and 3,200 levels where one or both walls are sedimentary, the sulfides are predominantly pyrite and chalcopyrite, much of which ore is low grade. In contrast, much richer bornite-chalcopyrite ores are in the vein above, and richer bornite-chalcopyrite-tennantite-energite-chalcocite ores are below. No satisfactory explanation can be given for this distribution of the sulfides.

The Main ore shoot does not reach the Troy quartzite, but its upward extension, as represented by smaller discontinuous ore bodies, has quartzite, limestone, or diabase as wall rocks. Practically all the zinc ore bodies have quartzite or limestone for one wall. Except for one or two small stopes in the upper levels, no ore has been mined where both walls are limestone. These relations between copper ore and the Paleozoic beds in the Magma mine are probably more a function of the strength and zoning of the ore solutions and of oxidation than any particular property of the quartzite and limestone. The location of the zinc ore bodies, although mainly a matter of zoning, appears to be somewhat influenced by the Troy quartzite (Pl. II).

Replacement bodies in limestone account for only a small amount of the ore. They are limited to the zinc-copper stopes in the upper levels east of zero crosscut. Particularly in stopes above the 1,800 level, a 20-foot bed at the bottom of the Martin limestone, just above Troy quartzite on the south side of the vein, has been replaced for about 20 feet. This horizon just above the Troy quartzite-Martin limestone contact is particularly important in the Lake Superior and Arizona mine.

In E. 33 south crosscut on the 1,800 level, there has been some replacement of limestone. This area is near the projection of the Martin-Escabrosa contact, and the beds at this contact may be slightly more replaceable than others.

In the Belmont mine area, the top 20 to 25 feet of the Escabrosa limestone is the host rock for most of the ore bodies. This horizon has not been prospected in the Magma mine.

Pinal schist: Mining has not progressed sufficiently to establish any definite relations between ore and schist, but certain important tendencies are strongly suggested.

The Main ore body has a schist wall only in the western portion of the mine below the 3,600 level. There the ore shoot maintains the same westerly pitch as in the lower diabase sill.

Of great importance is the character of the Magma fault and vein in the schist. On the upper levels, particularly in diabase, the vein walls are sharp, with little mineralization and alteration beyond them. Such is not the case in the schist. There the zone

of faulting and mineralization is considerably wider than in the upper levels and commonly contains masses of fairly unbroken and unmineralized wall rock. Its walls are less distinct and in many places indefinite. Adjustment of forces within the fault zone did not take place in one definite zone but rather in a series of them perhaps over 100 feet wide. The ore solutions worked their way up through zones of weakness within the broad zone, and ore deposition was not confined between definite walls. As a result of this spreading out, the ore bodies are not continuous for any great distances but tend to be lenticular both horizontally and vertically. These lenses of ore may be close together and constitute a single ore body, they may be separated by low-grade mineralized schist, or they may be separated by unmineralized, fresh schist. In position they may be opposite one another, end to end, in echelon, close together, or far apart. Each lens has definite walls striking and dipping about parallel to the general trend of the vein, and an adjacent lens may have equally definite walls. The boundaries of the vein are arbitrary lines enclosing all these lenses and all the mineralized schist. The full width of the vein in the schist cannot be mined, and much of it will be of very low grade.

In conjunction with the widening of the mineralized zone in the schist, the mineable vein forms two branches, the "Main vein" and the "North branch." This split has been found only where both walls of the vein are in schist. On the 4,000, 4,200, and 4,400 levels, the junction of the North branch and the Main vein is fairly well known, and this junction plunges east approximately parallel to the Pinal schist-Pioneer shale contact (Pl. II). The distance between the two branches west of the contact is fairly constant, ranging between 75 and 125 feet. In a general way these two branches appear to form a hanging-wall and footwall belt for the wide mineralized zone. Both branches are strongly mineralized in places but at others are almost barren. The ore in the branches occurs as lenses such as those already described. Between the Main and North veins the schist in many places is altered, broken, and mineralized. Not enough prospecting has been done to determine whether or not any ore lies between the two branches.

WALL-ROCK ALTERATION

No detailed study was made of the wall-rock alteration. Extensive serpentization and uralitization (formation of fibrous hornblende), probably during or closely following the emplacement of the diabase, rendered the diabase essentially impervious to ore solutions a short distance from the permeable fault zone. Later alteration by the ore solutions sericitized the feldspar and produced some kaolin. Chlorite was formed from the earlier augite and hornblende. Some carbonates were introduced. Alter-

ation of the siliceous and aluminous sedimentary rocks and schist likewise consists of silicification and sericitization, but it is of less intensity than in the diabase.

AGE OF MINERALIZATION

Ore deposition closely followed the premineral faulting and may have begun before it entirely ceased.

WATER TABLE

In 1910, before extensive mining operations, water stood at about the 400 level (elev. 3,150 feet above sea level) in the No. 1 shaft, but since that time it has been continually lowered by new workings. At present, water is flowing into the 1,100 level (elev. 2,450) of the No. 5 shaft, at the bottom of the dacite (elev. 2,500) in No. 6 shaft, and above the 1,600 level (elev. 2,000) near E. 22 crosscut position. Water was encountered also in the east drift of the 4,000 level, in the 4,400 and 4,600 west drifts, in drill holes on the 4,800 level, and in the Koerner vein on the 3,600 and 4,000 levels. In the Lake Superior and Arizona shaft, the water table is near the tenth level (elev. 2,400). About 400 gallons of water per minute are now being pumped from the Magma mine.

OXIDATION AND SUPERGENE ENRICHMENT

Considering the large proportion of iron in the Magma ores and their concentrated rather than disseminated nature, it is surprising that the outcrop does not show a strong gossan. Oxidizing processes have resulted in a leaching of iron at the surface (Pl. X B).

Plate X A shows the outcrop of the Magma vein, indicated by the line of prospect pits.

At moderate depth, about 100 feet below the surface, carbonates of copper with only a relatively small proportion of iron oxides appear. Even at the top of this zone oxidation was incomplete, and pockets of chalcocite occur high above the general zone of supergene enrichment. The relatively impermeable character of the ores has been a factor in protecting them from complete oxidation, but the chief influence has been the arid climate of the region.

The absence of gossan is attributable to the extremely low proportion of pyrite in the hypogene ore. Blanchard and Boswell⁴² state that hydrolysis of ferric iron to limonite is prevented by free sulfuric acid and promoted by copper. In other words, a high concentration of copper in a ferric sulfate solution counteracts the effect of sulfuric acid in keeping iron in solution and precipitates the iron as limonite. Pseudomorphs of limonite after pyrite are

⁴²Blanchard, R. and Boswell, P. F., Notes on the oxidation products derived from chalcopyrite: *Econ. Geol.*, vol. 20, p. 616, 1925.

most commonly formed by neutralization of sulfuric acid by carbonate solutions. It follows that if small or moderate amounts of pyrite had been present in the outcrop before oxidation, a considerable part of the iron would have remained as limonite.

Furthermore, Blanchard and Boswell show that sericitized porphyry acts similarly to carbonates in its tendency to neutralize sulfuric acid and thus permits ferric iron solutions to hydrolyze.⁴³ Oxidation products from pyrite in general travel only a limited distance is sericitized porphyry before being precipitated. Therefore, had there been a small or moderate amount of pyrite in the present outcrop, some of this iron would have precipitated in the porphyry as limonite.

Following Blanchard and Boswell, limonite is used as a field name for fine-grained yellowish or brownish deposits derived by decomposition of iron-bearing minerals. Presumably it is generally ferric oxide monohydrate (goethite), but it may be ferric oxide (hematite), lepidocrocite, jarosite, or certain basic ferric sulfates.⁴⁴

Deeper exploration after 1910 showed that the zones of oxidation and enrichment have no apparent relation to present water level. Sulfides prevail in the western portion of the mine from the 500 to the 800 levels. In the extreme eastern parts between these levels, carbonates and oxides of copper appear with the sulfides, and the porphyry is iron-stained where it is not mineralized. Above the 500 level the bottom of the oxidation zone is practically horizontal.

Between the 800 and 1,500 levels the zone of oxidation does not reach the productive ores, but successively farther east and downward the porphyry changes in color from grayish white, with little or no iron oxide, to rusty brown or gray with intermittent reddish brown streaks along fracture planes. Similar changes are noted where the vein lies entirely within diabase. This change at the lower limit of intense oxidation is indicated by a line on Plate II. The bottom of the oxidation zone dips in a direction opposite to the present surface.

The line on Plate II indicating the lower limit of oxidation is not sharply defined, and a zone of incomplete oxidation underlies it. In places, such as E. 16 to E. 22 stopes on the east 1,500 level, the ore is preponderantly sulfide, but scattered small pockets or seams of limonite and other oxides occur in the midst of sulfides. Such interfingering of oxides and sulfides is common in arid regions.

Above the oxidation line as drawn on Plate II, in stopes above the East 1,800 and 2,000 levels, sphalerite is rare, and much of the zinc has been removed by oxidizing solutions. Some zinc remains as smithsonite and zinciferous tallow clay, associated with abun-

⁴³Blanchard and Boswell, *op. cit.*, p. 631.

⁴⁴*Op. cit.*, p. 615.

dant limonite and hematite. The lower limit of oxidation below the 2,000 level is unknown.

The local base level of the district is the bed of Queen Creek, and drainage is westward. Moreover, before pumping the water level stood 150 feet above the local base level and was higher eastward. Experience has shown that important oxidation below water level is exceptional without long periods of geologic time or unusual conditions of water circulation. It is likely to be local, as cusps extending down from a more or less regular surface.

If oxidation took place after tilting, the water level must have been at least 1,600 feet deeper than at present, as oxidation appears in the 2,000 east drift and the present water table is the 400 level. However, deep oxidation, entirely unrelated to present water level, is common in the ore deposits of the Southwest.

The parallelism shown by the bottom of the zone of oxidation and the base of the dacite (Pl. II) suggests that the oxidation took place after tilting, during the long erosion period marked by the Whitetail conglomerate. On the other hand, the lower limit of oxidation roughly parallels the sedimentary beds. This relation might mean that ore deposition and oxidation preceded the tilting, but such an interpretation seems difficult to reconcile with the structural history of the region and with the occurrence of sulfides down the dip in the L. S. and A. mine (p. 138).

Intense oxidation is limited to places where the vein is in Paleozoic sedimentary beds; little limonite or other oxidation products have been found where the vein is in diabase. This difference is believed to be a factor of permeability. The diabase away from the fault zone is very impermeable in comparison with the sedimentary rocks. The permeability of the sediments permits general circulation, but in the diabase water circulation it is limited to the fault zone. Within the vein, circulation was apparently too limited for oxidation but sufficient to produce small quantities of supergene copper sulfides.

As shown on Plate II, the lower limit of oxidation above the 500 level is essentially horizontal and only 50 feet below the water level which existed before mining. In relatively recent times the water level may have been 50 feet lower than it is at present, and oxidation could have followed it to this depth. As the dacite has been removed from above this portion of the deposit in relatively recent times, recent oxidation would be effective. Thus, for this portion of the mine, recent oxidation appears to have been superimposed on earlier oxidation.

PARAGENESIS

Introduction: Paragenesis refers to the relative ages or periods of deposition—the origin, association, and sequence—of minerals in an ore body. These data are obtained by microscopic study of thin and polished sections of ore specimens. Although the study of mineral paragenesis may not lead directly to discovery of new

ore bodies, it gives information regarding the complex processes of ore deposition and to the character of ore solutions. It usually can determine whether sulfide ores are hypogene or supergene.

It is not practical to give in this bulletin a comprehensive discussion of the microscopic criteria; they have been adequately outlined in the literature.⁴⁵

Specimens studied: The following description of the ore minerals of the Magma mine is based on a study made with the reflecting microscope on 314 polished sections. Ninety of them were from the 1921 collection of Short and Ettlenger, comprising specimens from the 400 to the 2,000 levels, inclusive. Of the remainder, collected more recently, 116 are from the Main ore body below the 2,000 level, 14 from the West (No. 5) ore body, and 94 from the East ore bodies.

HYPOGENE SULFIDE MINERALS

Pyrite (FeS₂): The most abundant sulfide in the Magma mine is pyrite. Especially on the lower levels, large tonnages of pyrite, too low in copper to be mined at a profit, occur in many places. Low-grade pyrite is found along the margins of most of the ore bodies. On the upper levels the percentage of pyrite in comparison with the copper sulfides is low, which fact accounts for the lack of conspicuous gossan along the outcrop of the Magma vein. Pyrite crystals are common in polished sections of ore, but most of the pyrite is massive, without crystal faces. The massive pyrite was probably formed by precipitation around closely spaced nuclei, with allotriomorphic development of the grains. The pyrite grains almost invariably show embayments or veinlets filled by other sulfides, but no evidence of its replacement of other sulfides was observed. Pyrite in the "exploded bomb structure" similar to that depicted by Graton and Murdoch⁴⁶ is common in Magma ores (Pls. XX D; XXIII B; XXV B, C). This structure indicates that pyrite is earlier than the minerals in the cracks of the "bomb."

It is clear that pyrite is the earliest mineral of the Magma ores and that deposition of pyrite ceased before that of the other sulfides began. This relation does not imply that other metals were lacking in solutions depositing pyrite. No doubt they were present, but the excess of iron and sulfur was precipitated in advance of copper, lead, and zinc compounds.

In almost every specimen quartz accompanies pyrite. In some specimens small grains of pyrite are attached to proportionately larger veins of quartz (Pl. XVIII C); here the quartz evidently is earlier. In some tiny vugs the space between acicular crystals

⁴⁵Bastin, E. S. and others: Criteria of age relations of minerals with especial reference to polished sections of ores: *Econ. Geol.*, vol. 26, pp. 562-610, 1931.

⁴⁶Graton, L. C. and Murdoch, J., The sulphide ores of copper: *Am. Inst. Min. Engrs. Trans.*, vol. 45, p. 37, 1913.

of quartz is filled with quartz and sulfides. This quartz possibly grew by replacement of the sulfides, but more probably it lined the vugs which were later filled with sulfides. More commonly quartz veinlets cut pyrite (Pl. XXVI A). Quartz overlapped both the beginning and end of pyrite deposition, but in no specimen is quartz later than other sulfides. The period of maximum quartz deposition immediately followed that of pyrite.

Sphalerite (ZnS): Sparse sphalerite occurs in the main ore bodies of the Magma mine as small grains surrounded by copper sulfides. It is clearly later than pyrite (Pl. XXIII B), earlier than the copper minerals, with the possible exception of enargite, and earlier than galena (Pls. XXIV B; XXV A; XXVI B). The relation between sphalerite and enargite is not clear, as contacts between the two were not observed. Enargite is a relatively high-temperature, and sphalerite a relatively low-temperature, mineral. Hence, they tend to be mutually exclusive.

Sphalerite is abundant on the eastern margin of the Main ore body in the upper levels. In the lower levels it is sparse in the Main ore body but abundant in scattered ore shoots east of the main crosscuts (between shafts No. 2 and 3). Here it is almost invariably accompanied by galena. This association suggests that the two minerals may be contemporaneous, but polished sections reveal galena embayments and veinlets, some of which are parallel to cleavage directions of sphalerite, indicating that sphalerite is earlier than galena. The universal association of sphalerite and galena in many ore deposits suggests similar conditions of solution and deposition. Probably the sphalerite was deposited first and was more or less unstable in the presence of the remaining solutions that deposited galena. Sphalerite tends to form crystal faces, and euhedral forms surrounded by other sulfides are common. Most of them show embayments occupied by galena or copper sulfides.

Enargite (Cu_3AsS_4): Enargite is not found in the upper levels but occurs with increasing abundance downward from the 3,200 level, near No. 5 main crosscut. On and below the 4,200 level it is the most important ore of copper. Here it occurs massive, with prominent, coarse cleavage. The cleavage surfaces generally show rounded areas of pyrite.

In polished sections enargite is almost invariably intergrown with tennantite, and the patterns indicate that tennantite replaces enargite. In some places veinlets of tennantite follow the cleavage of enargite, but in other places the tennantite veinlets are curved and branching. Where replacement is nearly complete, the enargite consists of small, irregularly shaped areas rather closely bunched together and completely surrounded by tennantite. In ordinary light the two minerals cannot be distinguished by the unaided eye, but in polarized light tennantite is isotropic and en-

argite strongly anisotropic with pink, blue, and green interference colors. They may also be distinguished by etching with 20 per cent KCN solution, which stains enargite black and does not affect tennantite.

These enargite-tennantite intergrowths are common in many parts of the world, as, for example, at the Caridad mine.⁴⁷

According to Schneiderhöhn and Ramdohr⁴⁸ the process cannot be considered a replacement in the ordinary sense, since little material is transferred, but the process is analogous to an inversion on cooling. Enargite forms at higher temperatures, but as cooling proceeds it is unstable and reacts with copper-bearing solutions to form tennantite.

On the 3,200, 3,600, and 4,000 levels enargite is the dominant mineral of arsenical copper ores in the western parts of the ore body, whereas tennantite prevails in the eastern margins of the ore body on the same level. Reversals of this general association are common in specimens a few feet apart. This association might be regarded as evidence that the ore body has been tilted eastward; if restored to horizontal, the western part of the ore body on a given mine level would be lower and the eastern part higher than at present. Statistically, tennantite is the dominant sulpharsenite on the 3,200 level, and enargite is the dominant sulpharsenite on the 4,000 level.

The same relations obtain at Butte, where enargite is characteristic of the central copper zone in which relatively high temperatures prevailed, and tennantite is the dominant arsenic mineral in the intermediate copper zone where relatively lower temperatures prevailed.

Enargite is later than pyrite and earlier than the other copper minerals. The age relation between enargite and sphalerite has not been determined, but they are probably nearly contemporaneous.

Tennantite ($5\text{Cu}_2\text{S}_2(\text{Cu,Fe})\text{S}_2\text{As}_2\text{S}_3$): Tennantite was not found in the mine above the 1,000 level. That it was once present but destroyed by enrichment process seems unlikely, as tennantite is replaced with difficulty by chalcocite and covellite. Probably it was never deposited above the 1,000 level.

From the 1,000 to 3,200 levels it is abundant but less so than bornite and chalcopyrite. Where present it is almost invariably intergrown with bornite without evidences of replacement; probably it is essentially contemporaneous with bornite. The absence of any remnants of enargite indicates that the tennantite in this zone is not a replacement of enargite. Isolated specimens of

⁴⁷Wandke, Alfred, The Caridad mine, Sonora, Mexico: Econ. Geol., vol. 20, p. 315, 1925.

⁴⁸Schneiderhöhn, H. and Ramdohr, P., Lehrbuch der Erzmikroskopie: p. 462, Berlin, 1931.

enargite-free tennantite are also found below the 3,200 level, especially in specimens of low arsenic content.

Chalcopyrite (CuFeS_2): The most widespread copper sulfide is chalcopyrite. Above the 3,200 level it accounted for almost, if not quite, as much copper as bornite. Below the 3,200 level it is less abundant than bornite. In the richer ore bodies bornite predominates over chalcopyrite, but, on the other hand, in the low-grade pyritic ores, in zinc-lead ores, and in the ore bodies east of zero crosscut, chalcopyrite is present in various amounts, whereas bornite is lacking or sparse. Some chalcopyrite occurs in most specimens of bornite, with which it appears to be essentially contemporaneous.

Bornite (Cu_5FeS_4): The Magma mine has long been noted for its bornite ores. In the early days much of the bornite was believed to have formed as a supergene enrichment of chalcopyrite, and consequently the rich bornite was expected to give way in depth to leaner pyrite-chalcopyrite ores.

The undiminished continuity of bornite to the bottom level of the mine demonstrates conclusively its hypogene origin. Microscopic evidence confirms this conclusion. The intergrowths between bornite and chalcopyrite indicate essential contemporaneity. They differ widely from those of supergene enrichment, in places overlap of bornite is indicated by veinlets of bornite in chalcopyrite, indentations of bornite in chalcopyrite (Lindgren's "caries" pattern), and other textures indicative of local replacement.

Thin sections of bornite ores show bornite containing unreplaced remnants of rock minerals, bornite embayments in calcite cleavage planes, and bornite veinlets cutting across sericite and penetrating between sericite plates. Either bornite has extensively replaced rock minerals or has replaced pre-existing sulfides which have replaced rock minerals. There are no patterns indicating intensive replacement of the earlier sulfides, pyrite and sphalerite. Limited replacement of these sulfides has taken place, and in most places the original outlines of the pyrite grains have been preserved; where bornite is present, it is in cracks in pyrite (exploded bomb texture). Bornite and sphalerite largely appear to be mutually exclusive. In rich sphalerite ores bornite is absent, and the copper is represented by chalcopyrite or, less commonly, tennantite. In rich bornite ores sphalerite, if present, is represented by tiny isolated grains. The conclusion is then justified that the amount of bornite replacing earlier hypogene sulfides was small in comparison with that replacing rock minerals.

Bornite crystals have not been observed in the Magma mine. In general, bornite has a relatively weak tendency to form crystal outlines, even in open spaces.

Bornite intergrowths with chalcopyrite, with tennantite, and with chalcocite are common; those with galena are less so. All these minerals appear to be essentially contemporaneous.

Galena (PbS): In the Magma mine galena is almost universally associated with sphalerite. This association might suggest that the two minerals are contemporaneous, but detailed investigation shows that sphalerite is invariably the earlier mineral, as evidenced by embayments and veinlets of galena (Pl. XXVI B). The association indicates similar conditions of solution and deposition. It appears probable that the sphalerite, deposited first, was more or less unstable in the solutions which later deposited galena. It is doubtful that there was a hiatus between the two minerals. Zinc and lead were carried in solution together, but zinc came down first. Sphalerite always predominates in these mixtures. Galena is nowhere found in commercial quantities.

Intergrowths of galena with bornite, and galena with chalcopyrite, have been observed but are nowhere important. The patterns indicate that galena is essentially contemporaneous with the copper minerals.

Stromeyerite (Cu₂S.Ag₂S): Stromeyerite is rare in the Magma mine but has been observed in four widely separated localities. In the East ore bodies, 1,600 east drift, 17 4/5 stope, and 1,800 level, E. 21 1/5 stope, it is intergrown with bornite in subgraphic patterns. A similar intergrowth is found in the Main ore body on the 1,400 level; 8 crosscut (Pl. XXII B). A specimen from the 1,200 level, No. 8 raise, shows stromeyerite intergrown with galena in the "mutual boundary" pattern (Pl. XXII A).

Stromeyerite is probably contemporaneous with bornite and galena. Its location far below the enrichment zone and the patterns already described indicate that it is hypogene.

Deep-level chalcocite (Cu₂S): The term deep-level chalcocite is used to denote hypogene chalcocite which is intimately intergrown with bornite in graphics, irregular boundaries, and some types of gratings.⁴⁹

The graphic texture has been described elsewhere.⁵⁰ It is common in chalcocite-bornite specimens from the 1,200 to the 3,600 levels, but below the 3,600 level the chalcocite-bornite specimens are of the "patchy" type (Pl. XIX C, D). The areas in part show smooth boundaries, but in places the impression of partial replacement of bornite by chalcocite is marked. Probably chalcocite partially overlaps bornite, or the areas now occupied

⁴⁹This structure is fully discussed by Schneiderhöhn and Ramdohr (op. cit., p. 284) who termed it "lamellar structure" and believed it to be due to unmixing of a bornite-chalcocite solid solution. Bornite-chalcocite gratings are common in the central copper zones at Butte and in other districts but do not occur in the Magma mine.

⁵⁰Bastin, E. S. et al., op. cit., pp. 571-6.

by the intergrowth were once pure bornite which was later partially replaced by deep-level chalcocite.

Graphic textures have been observed in galena-bornite intergrowths (Pl. XXIII A) and many other mineral pairs which are invariably hypogene. It is generally agreed that galena is always hypogene, and this graphic is therefore hypogene. The question is whether the graphic texture is ever produced by supergene processes. Graphic bornite-chalcocite is observed in the zone of enrichment, but the texture has been largely destroyed by oxidation of the chalcocite to covellite (Short and Ettlenger, work cited, Figs. 18-21). In most places in the zone of enrichment the bornite has been completely replaced by supergene chalcocite, and the former graphics have been obliterated to form large masses of pure, steely chalcocite.

Hypogene chalcocite never occurs in isolated masses more than 5 mm. in diameter. Microscopic examination shows that it is everywhere intergrown with bornite.

The field relations offer the best evidence for hypogene origin of the deep-level chalcocite. They can be summarized as follows: If the chalcocite is supergene, the amount should diminish with depth; if hypogene, it may either diminish or increase in depth. The deeper the chalcocite occurs, the less is the likelihood that it is supergene.

Deep-level chalcocite persists to the bottom levels of the mine. Its abundance is indicated by the following table:

Year collected	Level	No. of specimens collected	No. of specimens with deep-level chalcocite
1921	2,000	21	7
1938	3,600	20	6
1940	4,000	28	9

In general, deep-level chalcocite is found only with richer bornite ores, but not all rich bornite ores contain chalcocite. The table, as well as that given in the earlier publication,⁵¹ shows that the ratio of specimens containing deep-level chalcocite is approximately constant from the 1,000 level downward. However, the 4,000 level has a longer stoping length than any level above it, and the total amount of deep-level chalcocite is greater on this level than on any above it. On the lower levels chalcocite accounts for about 5 per cent of the copper mined. The field argument is much more weighty than any based on interpretation of microscopic textures or on laboratory experimentation. In the opinion of M. N. Short, the field evidence proves beyond all doubt that deep-level chalcocite is hypogene.

Digenite (Cu₉S₅): Much of the deep-level chalcocite is actually composed of two minerals. One of them is white, is anisotropic

⁵¹Short and Ettlenger, op. cit., p. 207.

with steel-blue and pink interference colors, and has a strong etch cleavage parallel to the base. It is orthorhombic chalcocite. The other component is bluish and isotropic and gives an irregular and indefinite etch cleavage. This mineral was formerly believed to be isometric chalcocite, and its color was ascribed to a small percentage of dissolved covellite. The X-ray research of Buerger⁵² proved this bluish component to be digenite, Cu_9S_5 , a mineral formerly discredited. Digenite does not occur above the 3,400 level, but on this level and below, it forms a part of all deep-level chalcocite-bornite intergrowths. The patterns formed by chalcocite and digenite are of the mutual boundary type, and the two minerals are believed to be contemporaneous (Pl. XIX B). When etched with nitric acid, the difference between chalcocite and digenite disappears; both minerals turn dark bluish gray and present good etch cleavages.

SUPERGENE MINERALS

Oxidized minerals: Oxidized minerals are not abundant in the Magma mine. Those recognized are:

Azurite.....	$2\text{CuCO}_3 \cdot \text{Cu}(\text{OH})_2$
Chrysocolla.....	$\text{CuSiO}_3 \cdot 2\text{H}_2\text{O}$
Copper.....	Cu
Coronadite.....	$\text{MnPbMn}_6\text{O}_{14}$
Cuprite.....	Cu_2O
Dioptase.....	$\text{H}_2\text{O} \cdot \text{CuO} \cdot \text{SiO}_2$
Halloysite.....	$\text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2$
Hemimorphite.....	$\text{Zn}_3\text{Si}_2\text{O}_7 \cdot (\text{OH})_2 \cdot \text{H}_2\text{O}$
Hydrozincite.....	$2\text{ZnCO}_3 \cdot 3\text{Zn}(\text{OH})_2$
Limonite.....	Mixture of oxidized iron minerals
Malachite.....	$\text{CuCO}_3 \cdot \text{Cu}(\text{OH})_2$
Manganite.....	$\text{Mn}_2\text{O}_3 \cdot \text{H}_2\text{O}$
Olivenite.....	$4\text{CuO} \cdot \text{As}_2\text{O}_5 \cdot \text{H}_2\text{O}$
Psilomelane.....	MnO_2

With but few exceptions the oxidized copper minerals are found above the 500 level. Nowhere do they constitute ore. Emerald-green crystals of diopside, partly coated with small olive-green olivenite crystals, were found near the surface associated with chrysocolla, malachite, azurite, cuprite, and supergene chalcocite. Chrysocolla and malachite are the principal oxidized copper minerals. Chrysocolla occurred as small bunches and seams down to the 500 level. Most of the malachite replaced chalcocite and covellite. Native copper has been found in small amounts at the outcrop, in a diamond drill hole east of No. 6 shaft on the 2,000 level, and in $7\frac{1}{2}$ stope above the 2,250 level west of the Main fault.

White films of hydrozincite coat some of the sphalerite on the 1,600 level.

Limonite occurs in distinct veinlets cutting the sulfides and gangue minerals, as brown stains in the gangue minerals and as

⁵²Buerger, N. W., The chalcocite problem: *Econ. Geol.*, vol. 37, pp. 19-44, 1941.

pseudomorphs after pyrite. The first two modes of occurrence are due to transportation and subsequent hydrolysis of iron sulfate solutions. In the case of pseudomorphs after pyrite, the oxidized iron remained within the boundaries of the parent mineral.

Veins and irregular masses of manganese oxides, probably psilomelane and manganite, are common and fairly abundant south of the Magma vein on the East 1,600 level.

Kuhn (private report to Magma Copper Company, Sept. 1, 1942) noted that little ore has been found in the zone of oxidation. Only isolated bodies of chalcocite and chrysocolla ore with minor amount of azurite, malachite, and diopside were found in the upper levels. Native silver, native copper, cuprite, chalcocite, coronadite, and tallow clay (probably hemimorphite and halloysite) are found near the lower limits of the zone in the stopes above the 1,800 and 2,000 levels. It cannot be certain whether the small amount of ore in the zone of oxidation is due to leaching or to lack of deposition above what is now the lower limit of oxidation. The lack of a strong gossan and the small amount of secondary enrichment indicate however, that the small amount of ore in the oxidized zone is due more to a lack of primary deposition than to thorough leaching. Oxidation influences the zinc. Some of the zinc stopes above the 2,000 level have been discontinued because of oxidation. Those above the 1,500 level have not yet reached the oxidized zone but are fairly close to it. Two recently identified minerals were found in the 35, 35 1/5, 37, and 37 1/5 stopes above the East 2,000 level. They are near the lower limit of the zone of oxidation in what appears to be a bed replacement in limestone. One, a fairly common white, waxy clay, contains from 14 to 30 per cent zinc. Mr. J. J. Fahey, of the United States Geological Survey, examined it and wrote:

The sample of zinc-bearing material . . . is of a type referred to as tallow clay. It is composed of clay (probably halloysite) throughout which are disseminated very finely divided particles of a zinc mineral that has much higher indices of refraction than the enclosing clay, and probably is hemimorphite (calamine).

Occurring with the tallow clay is a small amount of a hard, black, zinc-bearing mineral identified by Dr. Harry Berman of Harvard University:

We find . . . that this black mineral is coronadite, a mineral having the composition $MnPbMn_2O_{11}$. Apparently the particular specimen sent . . . from the Magma has in it some zinc in place of the manganese or the lead, but in other respects it is the same substance. This mineral is exceedingly rare. It has been found definitely only once, in the Clifton-Morenci district, and perhaps also in a locality in Morocco.

Chalcocite (Cu_2S): Massive chalcocite ore bodies were mined from the 500 to the 650 levels. The derivation of part of this chalcocite from bornite is proved microscopically by occasional remnants of bornite in areas of chalcocite and by the fact that these rich chalcocite ores change in depth to ores in which bornite pre-

dominates. The chalcocite of the upper levels is steely white in the hand specimen. Under the microscope some specimens of it exhibit a peculiar mottled appearance (Pl. XVI B). It is generally of fine granular texture, the different grains having different colors varying from white through grays to very light blue. Rounded hazy patches of slightly grayer color in the chalcocite are believed to represent "ghosts" of the replaced bornite grains, the difference in color being due to incomplete removal of iron in the replacement process. These differences in color instantly disappear when the chalcocite is etched with an oxidizing reagent such as nitric acid or ferric chloride; the whole area turns a dark blue.

The completeness with which the hypogene sulfides of the upper levels have been replaced by chalcocite has obliterated all but insignificant traces of these minerals. Only pyrite appears to have survived the process, and even it has been strongly attacked. Some microscopic areas of unreplaced bornite surrounded by gangue are present. They owe their preservation to protection by the gangue. A small proportion of covellite is found in steely chalcocite (Pl. XVII B). The covellite originated as a direct oxidation product of the chalcocite. The steely chalcocite zone is irregular in shape and thickness. It grades upward into an oxidized zone containing pockets of residual sulfides. The lower limit of the steely chalcocite zone is more regular and parallels the sedimentary beds and the base of the dacite.

Native silver is found in supergene chalcocite (Pl. XV C). It seems to be later than the chalcocite. It has not been observed in polished sections of the hypogene ores, although analyses show silver. The hypogene silver may be present as stromeyerite or tennantite.

The zone of massive chalcocite changes rather abruptly in depth to a zone of rich bornite containing more or less supergene chalcocite. This zone is very limited and does not extend more than 150 feet below the bottom of the massive chalcocite zone. The 900 level marks its lower limit at its eastern end, and it extends diagonally upward toward the west, paralleling in a general way the dip of the formation. The bottom of this zone is irregular. The veinlets and gangue boundary rims of supergene chalcocite decrease in size and number with increasing depth until they practically disappear. Below this zone supergene chalcocite is seen only in microscopic veinlets and accounts for only an insignificant proportion of the total copper. The chalcocite in this zone is not of the mottled type. Most of it is distinctly bluish. This bluish chalcocite does not etch with dilute nitric acid and in some respects behaves as covellite. In places this blue chalcocite is replaced by tiny veinlets of covellite (Pl. XVI C). This evidence indicates that the blue chalcocite is supergene. Similar blue chalcocite is found in the Kennecott mine, Alaska. The blue color of the chalcocite has

been shown to be due to a small proportion of covellite contained in solid solution.⁵³ The presence of minute veinlets of covellite in blue chalcocite in Magma ores suggests that the blue color is due to oxidation of the chalcocite.

Covellite (CuS): Covellite occurs in the Magma mine as (1) a product of direct oxidation of chalcocite and (2) a replacement, generally of bornite and more rarely of galena.

The first mode of occurrence appears only well up in the oxidation zone where the covellite is invariably associated with more or less malachite and mottled chalcocite. This type of covellite is the only one observed in hand specimens (Pls. XV D; XVII B).

In the second mode of occurrence the covellite occurs as veinlets or as rosettes of small plates along seams or gangue boundaries (Pl. XVII A, C, D). The total amount of covellite is insignificant in comparison with hypogene sulfides and does not exceed 2 per cent in any polished section examined. On the 900 level it is relatively abundant, and from this level to the bottom of the mine very small amounts of it are seen in many of the specimens examined. The supergene derivation of this covellite is beyond question. Its distribution is spotty but shows a progressive diminution in the proportion of covellite to bornite from the 900 level downward. In sections from the 4,000 level the covellite is discernible only with higher magnification.

DISTRIBUTION OF COVELLITE

Level	Number of specimens collected	Number of specimens containing covellite	Per cent of specimens containing covellite
900	9	7	78
1,000	5	4	80
1,100	8	1	12
1,200	19	6	32
1,300	4	1	25
1,400	6	5	83
1,500	4	1	25
1,600	8	2	25
1,700	8	4	50
1,800	20	10	50
2,000	21	4	19
4,000	28	4	12

An investigation was made to determine whether mine oxidation might be responsible for the covellite. Specimens of bornite were collected on the 1,000 level from immediately beneath a crust nearly a foot thick which was obviously the result of mine oxidation during the eight years the drift had been open. If mine oxidation is competent to produce covellite it should have been

⁵³Posnjak, E., Allen, E. T., and Merwin, H. E., The sulphides of copper: Econ. Geol., vol. 10, p. 526, 1915.

produced here, but none was found. A specimen containing microscopic amounts of covellite was collected from the face of one of the crosscuts on the 1,800 level. As this specimen had been exposed to mine oxidation only a few hours before collection, its covellite cannot be due to mine oxidation. This evidence is conclusive that no covellite is formed by mine oxidation and that all the Magma covellite is supergene.

Although sphalerite enriches easily to covellite, no such enrichment was observed in polished sections of Magma ores. Occasional areas of sphalerite are observed in polished sections of ores from the zone of enriched bornite. The solutions evidently contained sufficient copper to develop covellite as a replacement of bornite but not sufficient to attack sphalerite.

Chalcopyrite (CuFeS_2): Chalcopyrite occurs as veinlets and plates cutting bornite. Although not important quantitatively, it is widespread and persists to the deepest workings of the mine. Some of these veinlets and plates follow cracks and seams in the bornite. These veinlets and plates are interpreted by M. N. Short as supergene. The replacement commonly took place along parallel directions in the bornite, giving a grating (Pls. XVII D; XVIII A, B).

VENTILATION AND AIR CONDITIONING⁵⁴

The Magma mine is in a semiarid region, with an average yearly precipitation from 1921 to 1943, inclusive, of 18.47 inches. The elevation of the 500 or main adit level is 3,034 feet above sea level. The average surface dry-bulb temperature is 72.4 degrees, and the average yearly surface wet-bulb temperature is 57.4 degrees. This represents an average yearly relative humidity of 38 per cent.

Rock temperatures: Rock temperatures taken on the lower levels of the mine are as follows:

Level	Degrees F.	Level	Degrees F.
2,000	109	4,000	140
2,250	112.5	4,200	143
2,550	116	4,400	146
2,800	120	4,600	149
3,000	124	4,800	152
3,200	127		

The high surface temperature coupled with the higher-than-average rock temperatures presents a serious ventilation problem.

Underground water: The underground water varies from 350 to 565 gallons per minute, with a yearly average of approximately 400 g.p.m. The limestone and other sedimentary rocks act as a

⁵⁴By C. B. Foraker, Engineer, Magma Copper Company (Apr. 12, 1944); see also A.I.M.E. T.P. 979, Sept., 1938; A.I.M.E. Trans., vol. 141, pp. 253-67, 1941.

PLATE XV

- A.—Supergene chalcocite (light) replacing bornite (dark). Larger areas of bornite are traversed by tiny cracks of chalcocite. By progressive widening of these cracks and rounding of sharp edges of bornite areas, unreplaced bornite is left as formless masses surrounded by chalcocite. This type of replacement has been found only in enrichment zone and has not been observed in deep-level chalcocite. 900-foot level, E. 2½ stope, 20 feet below 800-foot level. (x 100)
- B.—Chalcocite replacing pyrite. Enrichment process is intense here and no hypogene copper sulfides remain unreplaced. Outline of a large pyrite grain in midst of chalcocite area in upper right quadrant of picture is very clear, but only a few small patches of pyrite along margins of former grain have escaped replacement by chalcocite; interior of grain is now all chalcocite. Strongly indicative of supergene replacement. 500-foot level, W. 1½ stope. (x 100)
- C.—Native silver replacing supergene chalcocite. This type of silver mineralization occurs only in the enrichment zone and accounted for the silver production in the early years of the mine. 650-foot level, E. drift at 7 raise. (x 290)
- D.—Pyrite oxidizing to limonite in presence of covellite. Pseudomorphs of limonite are sharp and clean-cut. Area between pseudomorphs is practically all covellite which has replaced chalcocite, only a few small triangular patches of chalcocite remaining unaltered. Different shades of covellite are due to its anisotropism. Process is indicative of strongly oxidizing conditions. Pseudomorphs of limonite after pyrite are probably due to the neutralization by calcium carbonate solutions of sulfuric acid generated by oxidation of pyrite. 500-foot level, 1 W. raise, 30 feet below 400-foot level. (x 100)

cc:—chalcocite
bn:—bornite
py:—Pyrite
Q:—quartz

Ag:—native silver
lim:—limonite
cv:—covellite

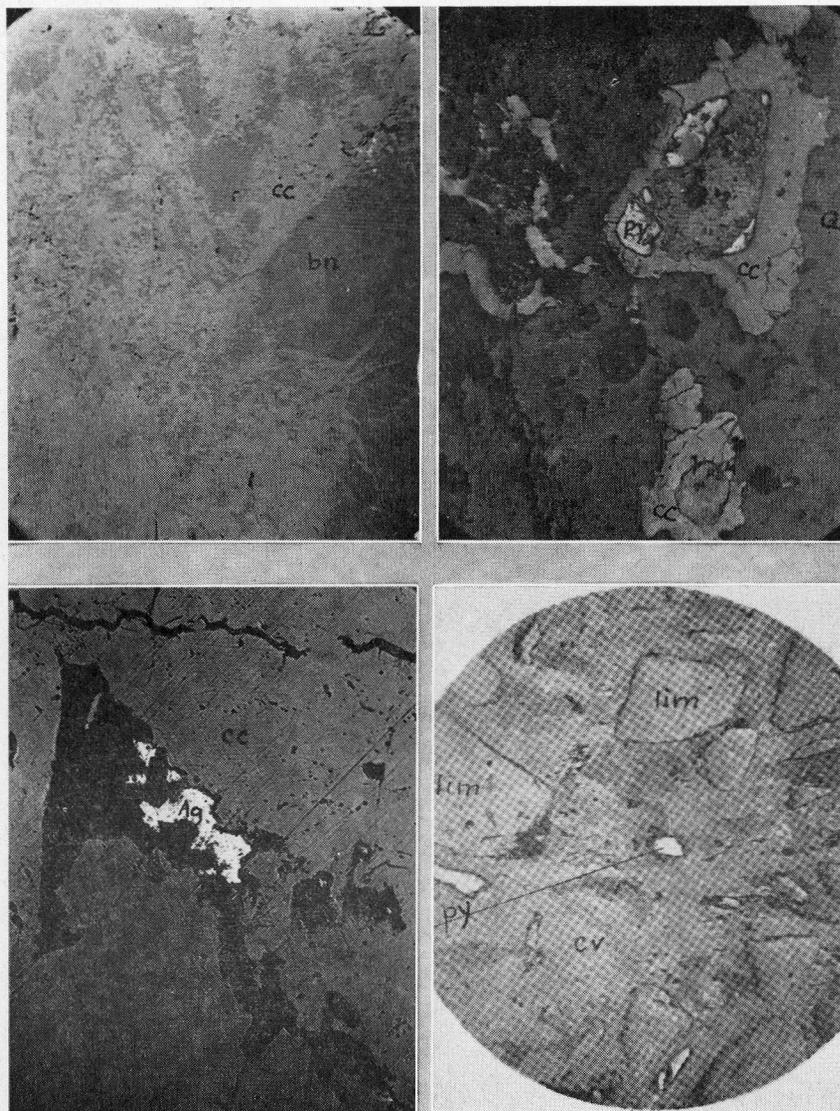


Plate XV.—Photomicrographs of Magma ore.

PLATE XVI

- A.—Chalcocite in veinlets cutting chalcocite-bornite intergrowth. Chalcocite in larger areas is of deep-level type. It shows "mutual" and sub-graphic patterns with bornite. Chalcocite in veinlets is distinctly later. It follows cracks in bornite. Where veinlets pass into larger chalcocite areas, chalcocite almost loses its identity, but continuity of veinlet is shown by scattered cracks. Cracks are later than deep-level chalcocite, and chalcocite in veinlets is later than cracks which it follows; hence, two generations of chalcocite are represented. It is believed that chalcocite in larger areas is hypogene and that in veinlets is supergene. 2,000-foot level, W. drift. (x 100)
- B.—Mottled chalcocite. Small darker patches in midst of lighter chalcocite are believed to represent "ghosts" of unreplaced bornite. When etched with HNO_3 these darker areas give etch tests like lighter chalcocite. This type of chalcocite is invariably supergene. 500-foot level, E. $1\frac{1}{2}$ stope. (x 175)
- C.—Blue chalcocite in bornite. This type of chalcocite has been subjected to oxidation. Wide parallel cracks are due to solution and removal of material. Covellite deposited along some of these cracks probably by direct oxidation. Origin of chalcocite in larger areas is obscure; it may be hypogene. Small veinlets of chalcocite near by are probably supergene. 3,600-foot level near main x-cut. (x 175)

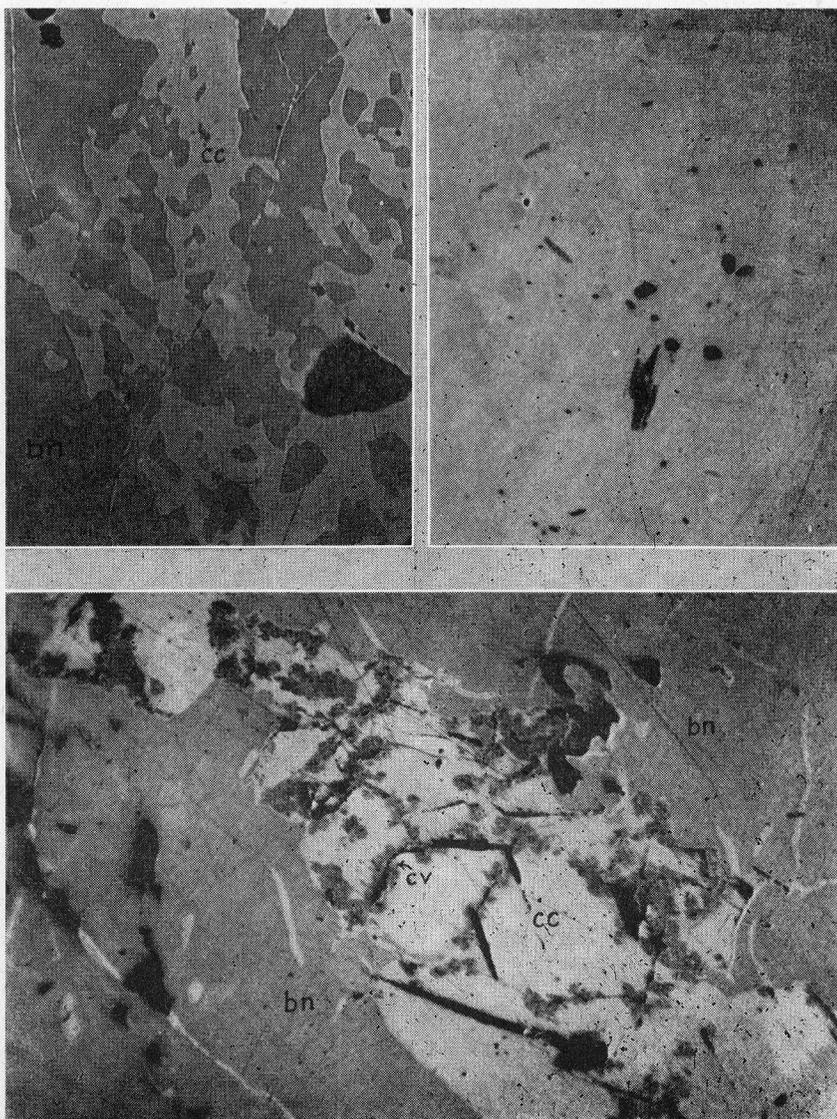


Plate XVI.—Photomicrographs of Magma ore.

PLATE XVII

- A.—Supergene covellite replacing bornite. This type of covellite has been found on the 3,600-foot but not on deeper levels. Amount of covellite in specimen is less than 1 per cent of total area. 3,600-foot level, 21 1/5 stope. (x 75)
- B.—Covellite replacing chalcocite. Covellite plates show different shades of gray due to anisotropism. This type of covellite is found only high in the enrichment zone where oxidation is superimposed on supergene enrichment. Covellite is due to direct oxidation of chalcocite. Malachite is found in the same specimen. 500-foot level, 1 1/2 raise. (x 100)
- C.—Supergene covellite replacing bornite along small cracks. From same specimen as A. (x 100)
- D.—Supergene covellite and supergene chalcopyrite replacing bornite. On upper right border of picture covellite follows boundary between quartz and bornite. Elsewhere covellite follows tiny seams in bornite. Chalcopyrite in part follows seam and in part replaces bornite along parallel, probably crystallographic directions. The quantity of covellite in specimen is about 5 per cent. 1,000-foot level, E. drift 30 feet from main x-cut. (x 160)

cp:—chalcopyrite

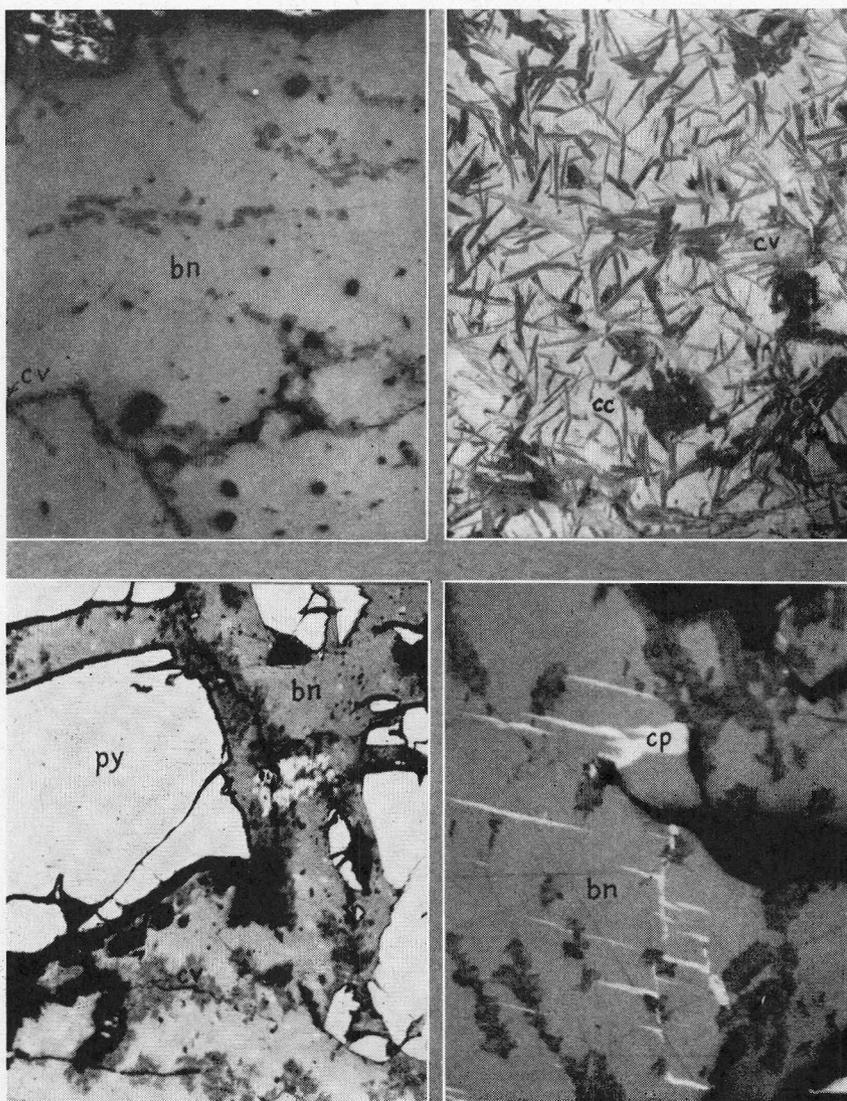


Plate XVII.—Photomicrographs of Magma ore.

PLATE XVIII

- A.—Supergene chalcopyrite replacing bornite along fracture. In the main, chalcopyrite follows fractures closely but shows a distinct tendency to go out into bornite as thin plates. Larger areas of chalcopyrite do not follow open spaces, have smooth boundaries with bornite, and are probably hypogene. 2,000-foot level, 15-W. x-cut. (x 75)
- B.—Chalcopyrite grating in bornite. Two possible causes for texture. In gratings of replacement origin, enlargements of chalcopyrite areas at intersections of chalcopyrite lamellae would be expected. Here they are lacking. The second and more probable theory for their origin is that the lamellae crystallized from an original solid solution on cooling. Both types of chalcopyrite-bornite grating have been produced by experiment. 3,600-foot level, 35 3/5 stope. (x 160)
- C.—Pyrite aligned along quartz. Pyrite occurs in small rounded grains which owe their position to quartz. Hence quartz is clearly earlier than pyrite. 3,400-foot level, W. 31 3/5 stope. (x 75)

sl:—sphalerite



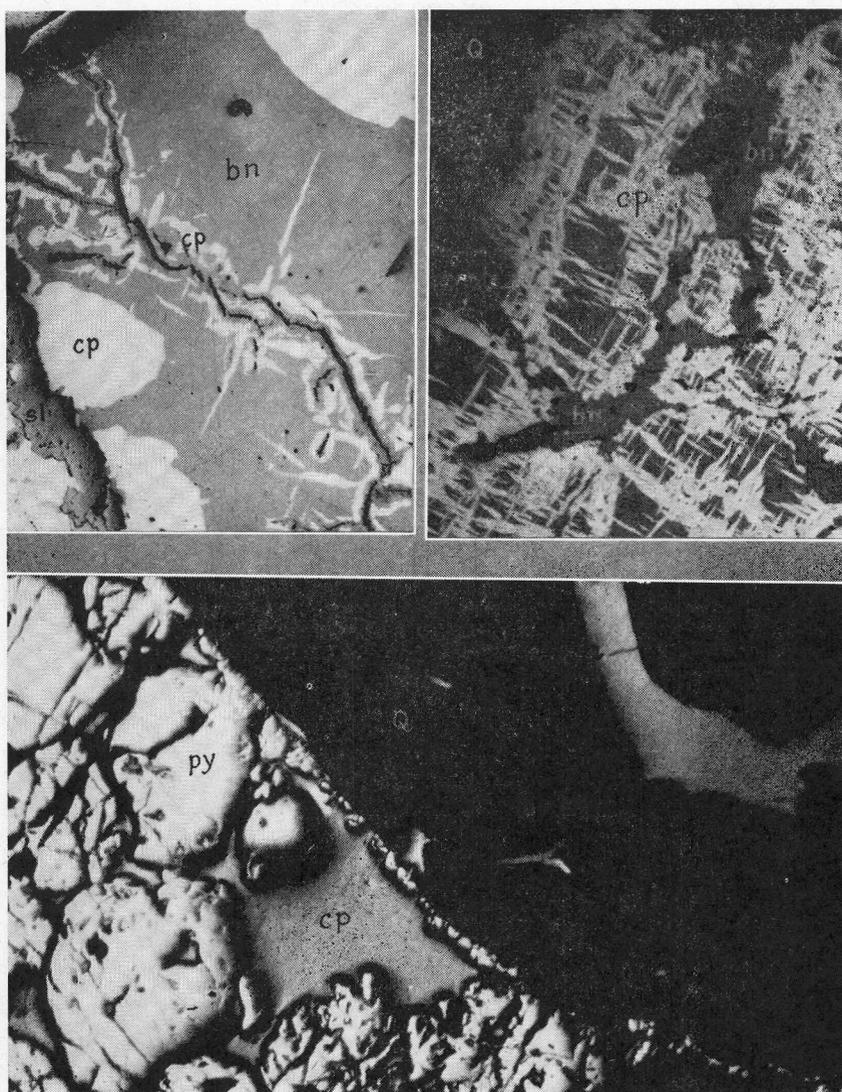


Plate XVIII.—Photomicrographs of Magma ore.

PLATE XIX

- A.—Subgraphic bornite-chalcocite intergrowth. In upper left corner parallelism of bornite lobes is sufficient to designate texture as "true graphic." Both minerals hypogene. 4,000-foot level, W. 34 1/5 stope. (x 75)
- B.—Digenite and chalcocite intergrown with each other and with bornite. Textures indicating replacement of one mineral by another are lacking. All minerals believed to be hypogene. 4,000-foot level, W. 34 1/5 stope. (x 75)
- C.—Patchy chalcocite-bornite intermixture. This is typical of lower levels of mine (below 3,400-foot level). Replacement textures not obvious. Some embayments in bornite areas may represent replacement. 4,000-foot level, 20 feet W. of 17 x-cut. (x 75)
- D.—Patchy chalcocite-bornite intermixture. Chalcocite is clearly replacing bornite. Larger chalcocite areas are joined by small veinlets of chalcocite. All chalcocite may be due to replacement of bornite, but it is more probable that most of chalcocite is contemporaneous with bornite, but chalcocite persisted after all bornite had been deposited. All chalcocite is hypogene. 3,600-foot level, W. 36 1/5 stope. (x 75)

wcc:—white chalcocite

bcc:—blue chalcocite (digenite)

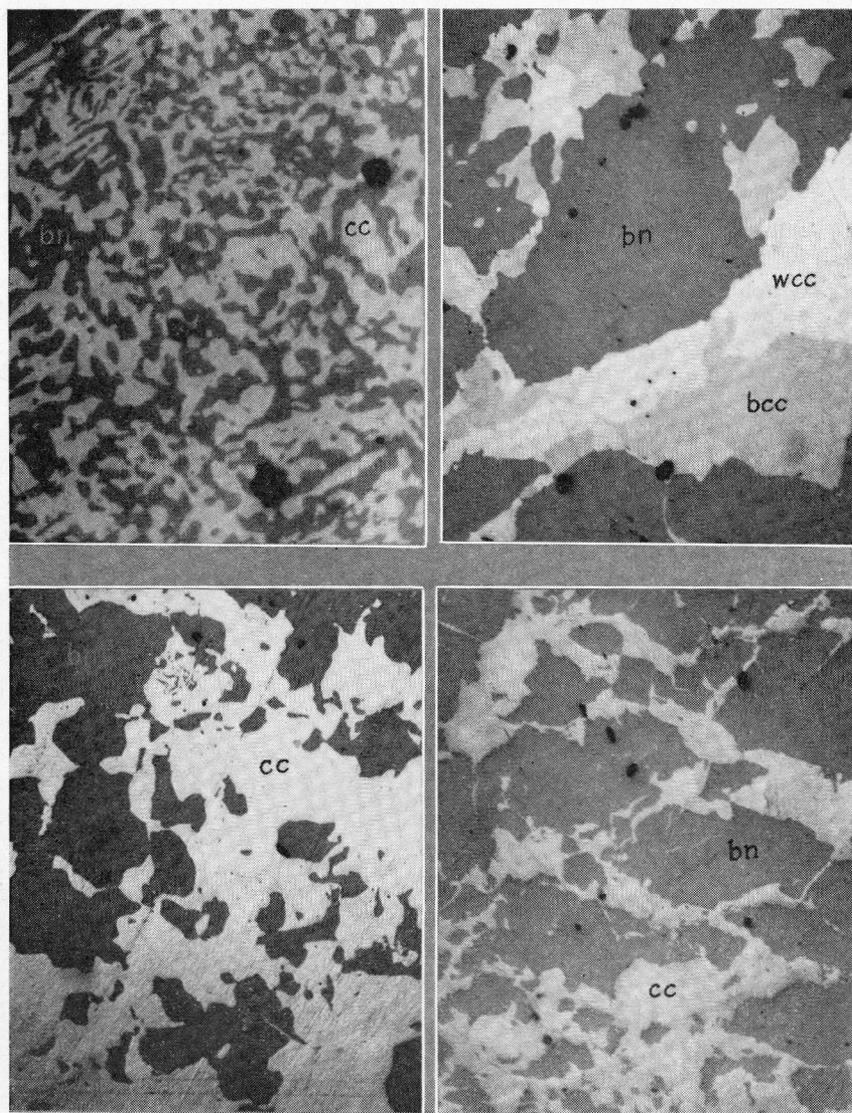


Plate XIX.—Photomicrographs of Magma ore.

PLATE XX

- A.—Blue chalcocite areas in bornite. Similar to Plate XVI, C. Parallel cracks in chalcocite show solution and removal of material. At least part and possibly all of chalcocite is hypogene replacement of bornite. Blue color of chalcocite is due to presence of covellite in solid solution. This may be due to direct oxidation of chalcocite. 3,600-foot level, W. 21 1/5 stope. (x 40)
- B.—Subgraphic chalcocite-bornite intermixture. Both are hypogene. 2,000-foot level, 8 x-cut. (x 100)
- C.—Graphic chalcocite-bornite intermixture. 3,400-foot level. W. 21 1/5 stope. (x 400)
- D.—Subgraphic chalcocite-bornite intermixture replacing pyrite in exploded bomb texture. 3,400-foot level, W. 24 stope. (x 100)

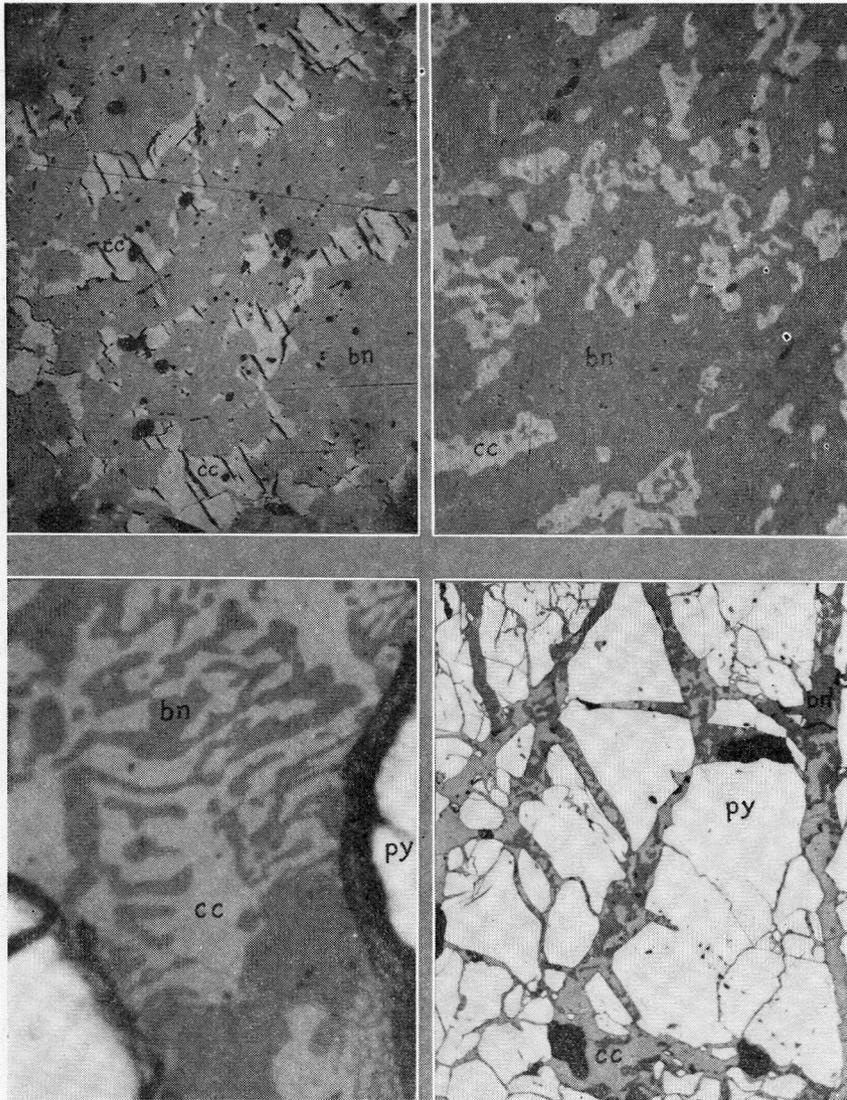


Plate XX.—Photomicrographs of Magma ore.

PLATE XXI

- A.—Deep-level chalcocite (light) and bornite (dark). Cigar-shaped bornite areas show pronounced parallelism. Between bornite "cigars" is a sub-graphic intergrowth of chalcocite and bornite. It is believed cigars are skeletal crystals of bornite which were first to form. With a decrease in iron content, remaining materials in solution separated out as simultaneous deposition of chalcocite and bornite. 2,000-foot level, 15 x-cut. (x 160)
- B.—From same section as A. Some of larger bornite areas are crossed by irregular veinlets of chalcocite. With further decrease in iron content, conditions were such that only chalcocite deposited. Solutions depositing chalcocite had some replacing action on bornite which took form of veinlets. These veinlets lack continuity and regularity of supergene chalcocite veinlets and do not follow cracks and gangue boundaries. All the chalcocite is believed to be hypogene. (x 330)

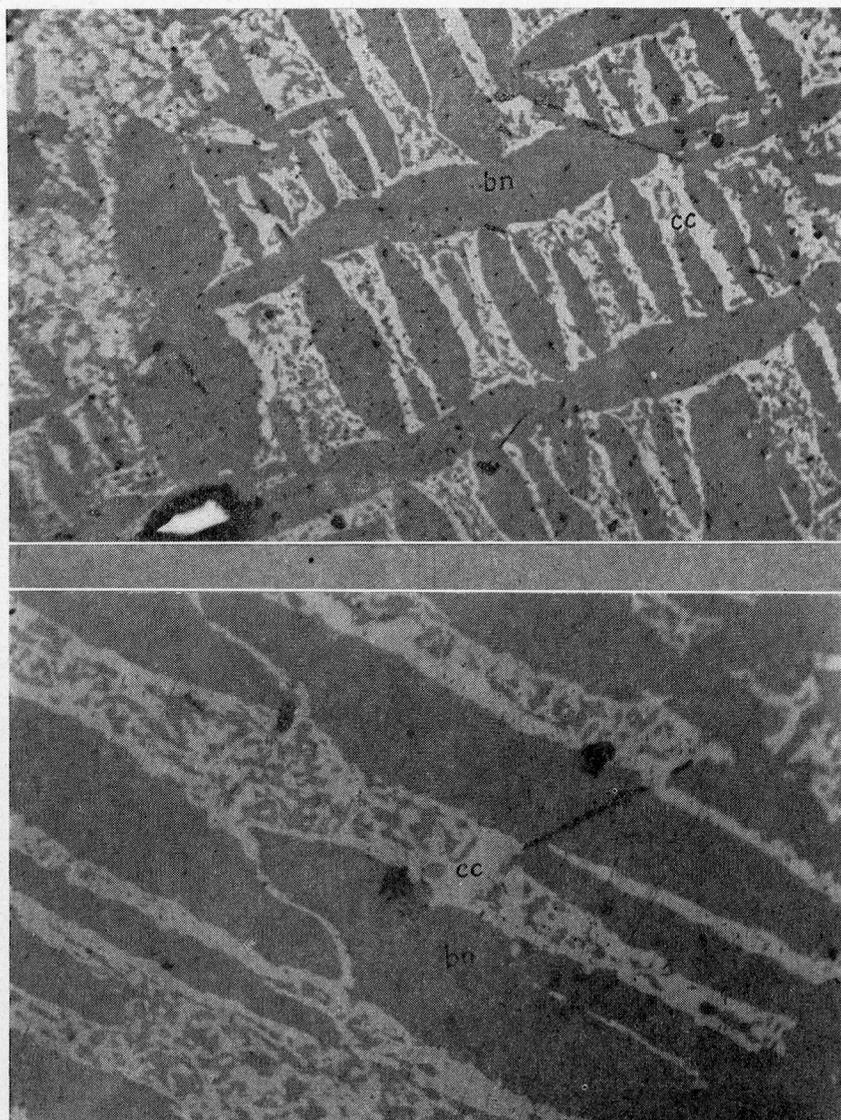


Plate XXI.—Photomicrographs of Magma ore.

PLATE XXII

A.—Stromeyerite-galena intergrowth in quartz vug. Metallic minerals show typical "mutual boundary" pattern. They are believed to be contemporaneous and hypogene. 1,200-foot level, W. drift near 8 raise. (x 100)

B.—Stromeyerite inclusions in bornite. These textures afford no proof of relative age of the minerals. Galena inclusions in upper right corner have same shape as those of stromeyerite. All minerals are hypogene. 1,400-foot level, W. drift, 8 x-cut. (x 160)

gn:—galena

str:—stromeyerite

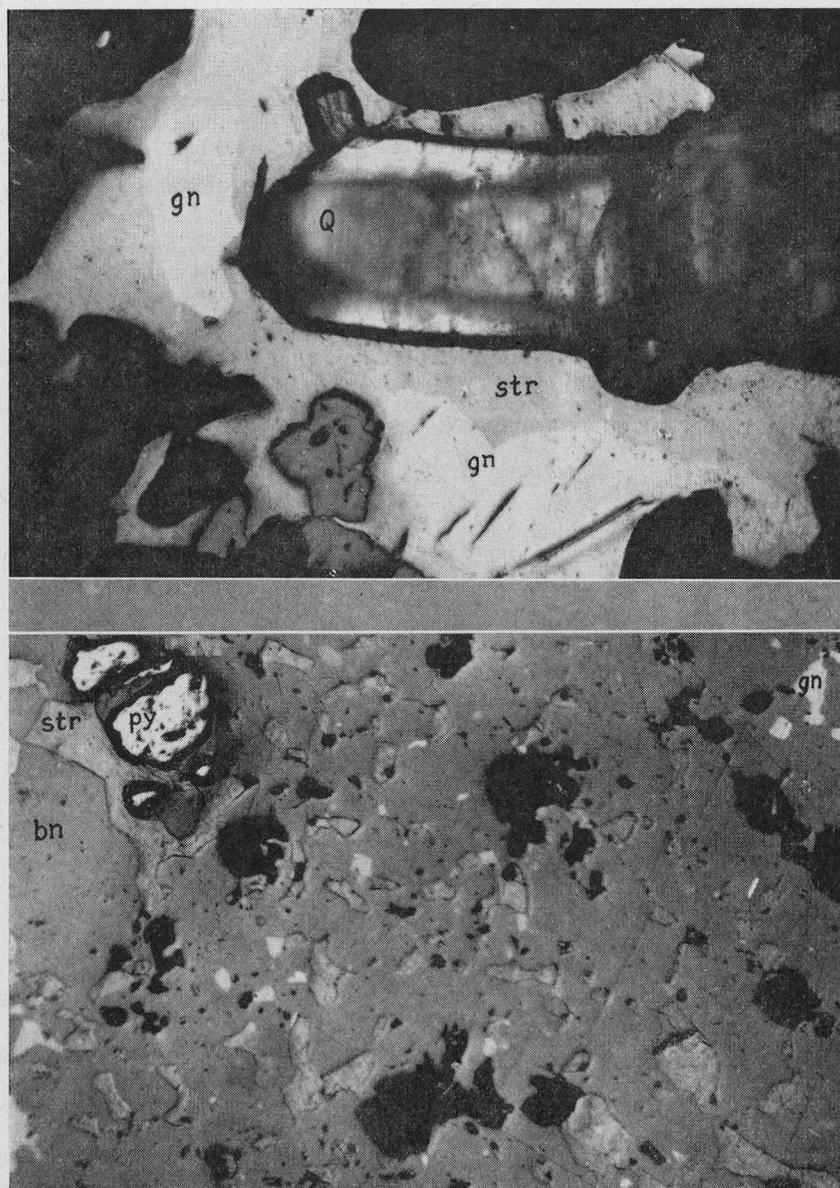


Plate XXII.—Photomicrographs of Magma ore.

PLATE XXIII

- A.—Graphic pattern of galena in bornite. Galena is invariably hypogene. Graphic texture is, in this case, a proved hypogene texture. Conclusion is that chalcocite in similar textural relation to bornite is likewise hypogene. 1,000-foot level, E. drift, E. ore body. (x 330)
- B.—Sphalerite replacing pyrite in exploded bomb texture. 1,800-foot level, E. drift at 20 raise. (x 75)
- C.—Bornite replacing enargite. Parallelism between bornite areas and enargite cleavage (short vertical cracks) indicates that enargite exerted a crystallographic control on bornite deposition. 4,000-foot level, 35 3/5 x-cut. (x 75)
- D.—Bornite and chalcocite in veinlets replacing enargite. Bornite and chalcocite have been etched with 1:1 HNO₃ to bring out contrast with enargite. Chalcocite in upper part of picture shows distinct etch cleavage. All minerals are hypogene. 4,000-foot level, W. 37 stope, 2nd floor. (x 80)

en:—enargite

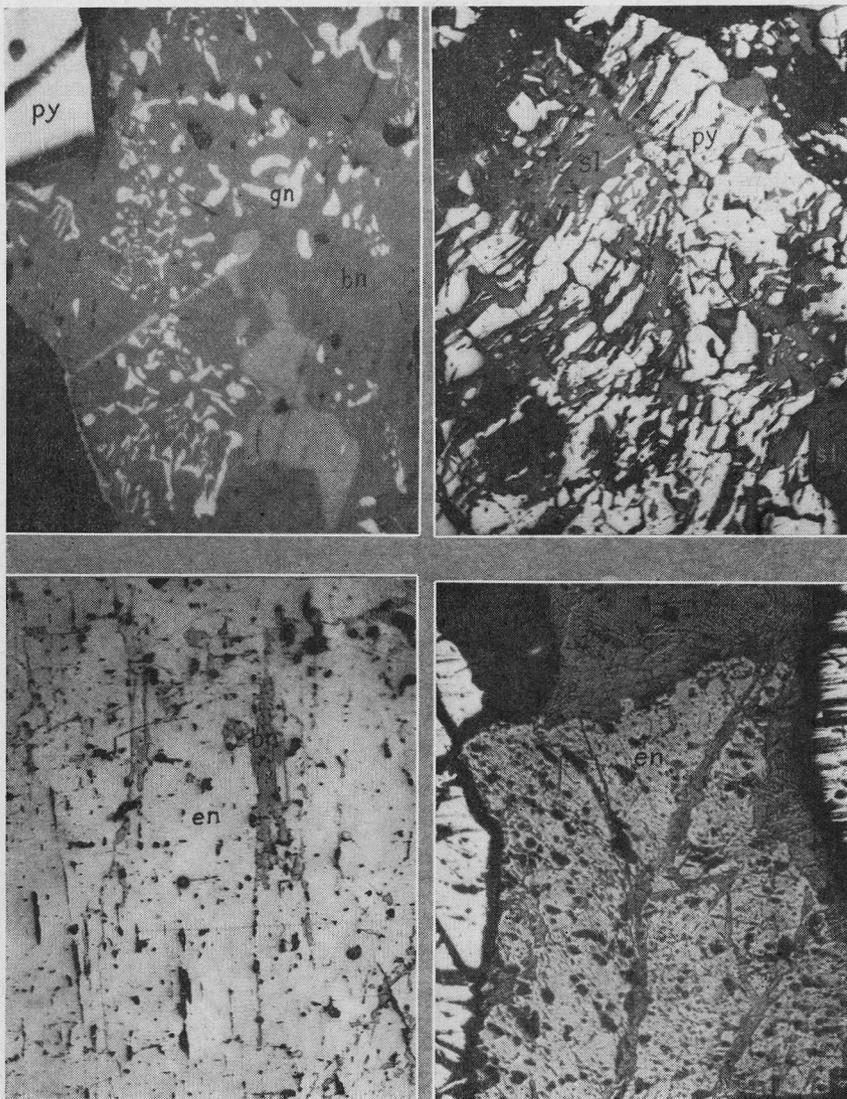


Plate XXIII.—Photomicrographs of Magma ore.

PLATE XXIV

- A.—Bornite-tennantite intermixtures. No texture indicative of replacement is seen and both minerals are probably nearly contemporaneous with bornite deposition overlapping that of tennantite slightly. Quartz is euhedral and is probably earlier than all sulfides. Either quartz crystals formed a vug which was filled by sulfides—the more probable explanation—or sulfides have replaced some gangue formerly surrounding quartz. 1,800-foot level, W. drift at 13 x-cut. (x 100)
- B.—Tennantite in veinlets cutting sphalerite. In part these veinlets fill cracks in sphalerite and in part tennantite occupies boundaries between quartz and sphalerite. Most geologists regard tennantite as always hypogene, but these textures point rather strongly to supergene origin. 1,800-foot level, W. drift at 18 x-cut. (x 75)

tn:—tennantite

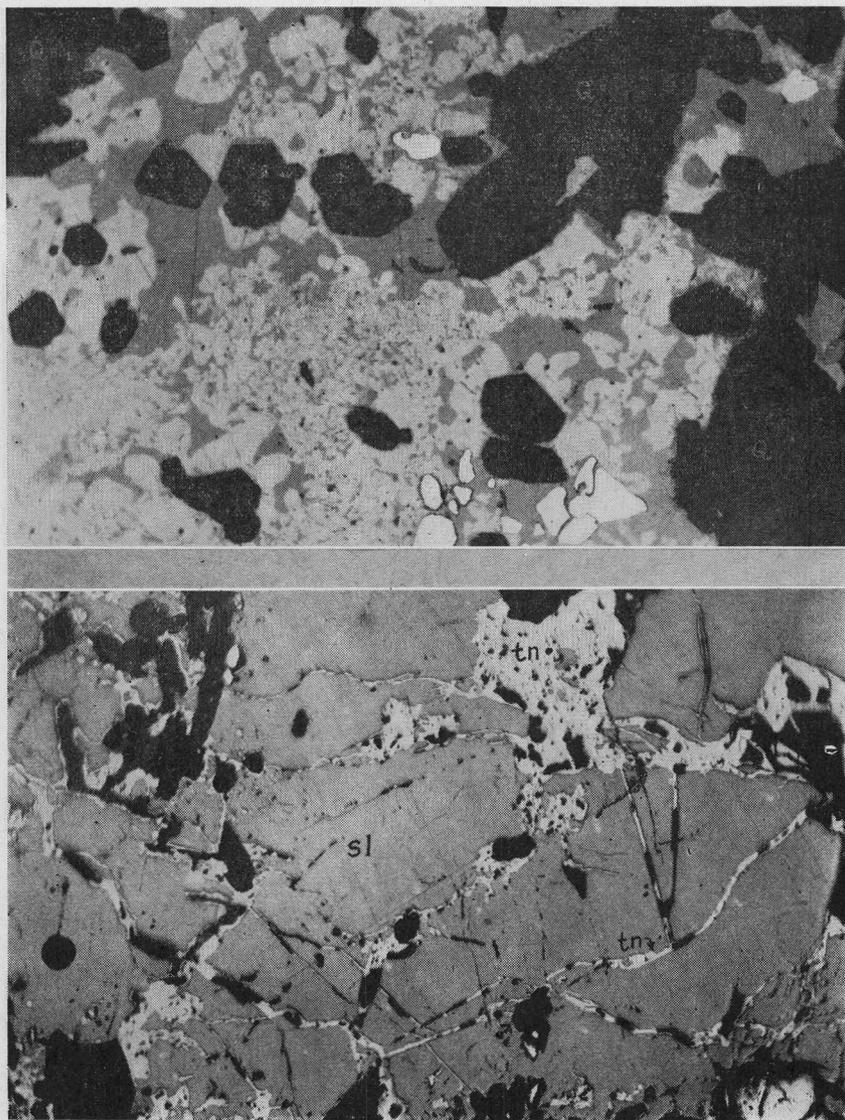


Plate XXIV.—Photomicrographs of Magma ore.

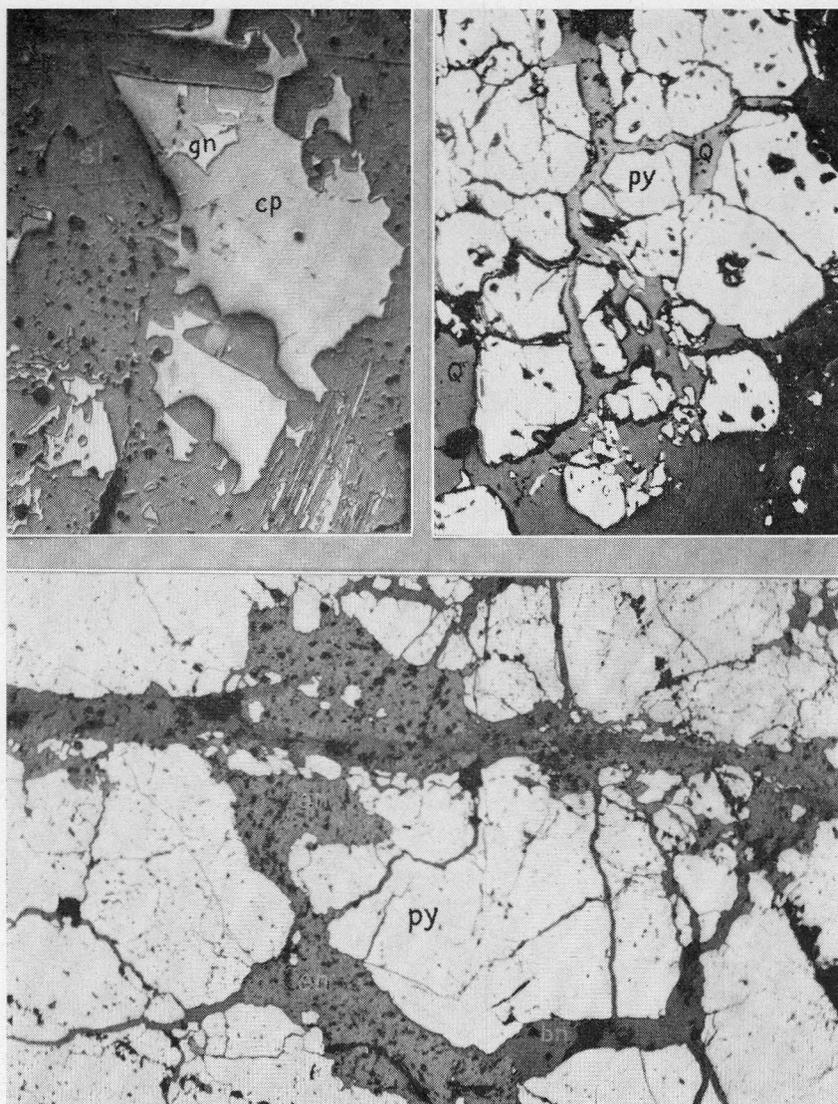


Plate XXV.—Photomicrographs of Magma ore.

- A.—Chalcopyrite and galena replacing sphalerite. Straight borders of sphalerite indicate that replacement was controlled by crystallographic directions in sphalerite. 1,500-foot level, 50 feet west of main x-cut. (x 55)
- B.—Quartz replacing pyrite. 1,800-foot level, main x-cut. (x 36)
- C.—Enargite and bornite replacing pyrite. 4,000-foot level, W. drift at 23 $\frac{4}{5}$ raise. (x 75)

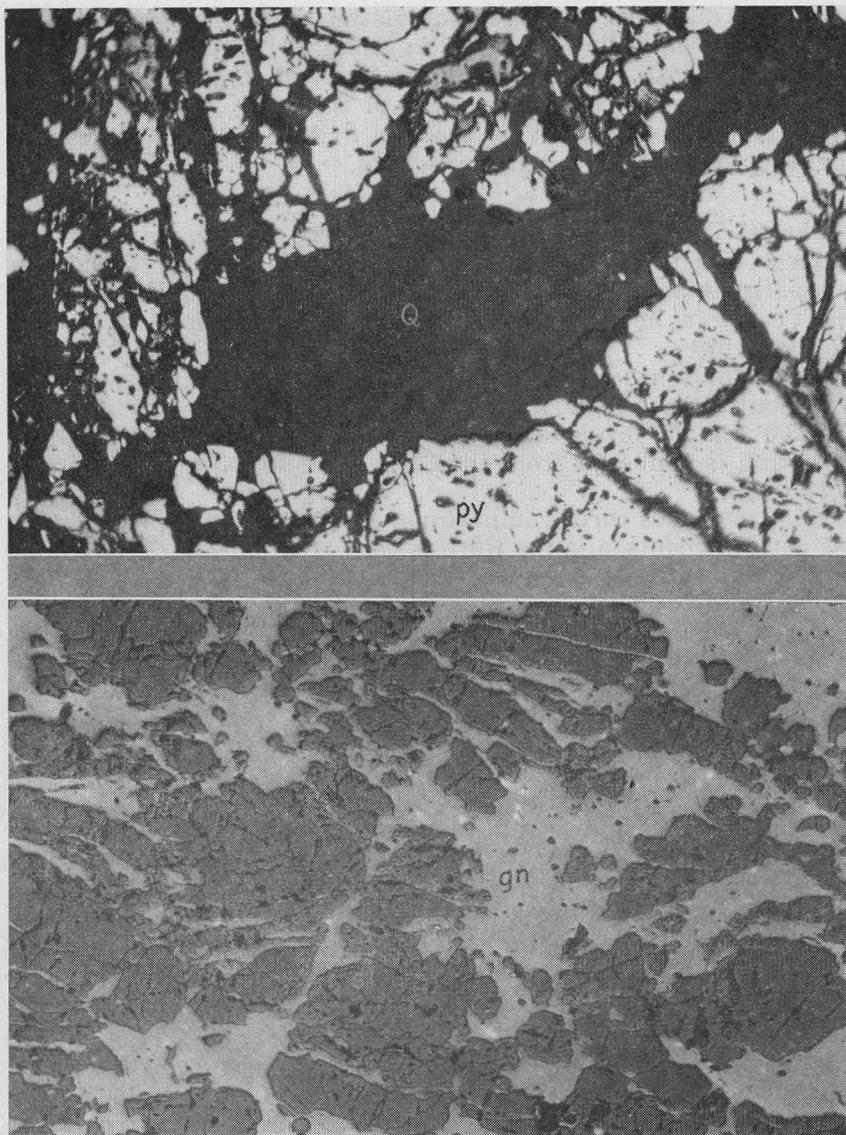


Plate XXVI.—Photomicrographs of Magma ore.

A.—Quartz replacing pyrite. 3,600-foot level, 29 ³/₅ stope. (x 75)

B.—Galena replacing sphalerite. 1,100-foot level near main x-cut. (x 35)

reservoir, and the rain water percolating through them from the surface is eventually found underground, principally on contacts between diabase and sedimentaries. For this reason most of the water is found in the eastern part of the mine, where the sedimentaries have been cut by the east drifts. The vein, however, for the most part, is damp throughout.

Shafts: The eight shafts on the property are numbered in the order in which they were sunk. No. 1, the original shaft, had two compartments 4 by 4 feet and was sunk to the 800 level. This shaft caved in, in 1927, and does not enter into the present ventilation system.

No. 2 shaft has one compartment 3 by 4 feet and two compartments 4 by 4 feet. It is connected to the surface by adits on the 200 and 500 levels. It is concreted from the 200 to the 3,200 level and is timbered from the 3,200 to the 3,600 level, where it bottoms.

No. 3 shaft has three compartments 4 by 5 feet. Two are hoisting compartments, and the third is used as a manway; it also contains pipe columns and power cable. It extends from the surface to the 4,800 level.

No. 4 is the exhaust shaft for the east section of the mine and is concreted throughout. It has two compartments 5 by 5½ feet, one 5 by 6 feet down to the 1,000 level, and two compartments 5 by 5½ feet from the 1,000 to the 1,500 level. On the 1,000, 1,200, and 1,500 levels it is connected by a system of raises to the lower workings.

No. 5 shaft was sunk to prospect the faulted part of the vein on the west end and is southwest of the main stoping area. It extends from the surface to the 4,800 level and has four compartments 4 by 5 feet, of which two are for hoisting, one is for a manway, cable, and pipe, and the fourth is open for ventilation.

No. 6 shaft was sunk from the surface to the 2,550 level approximately 4,550 feet east of shafts No. 2 and 3 for the purpose of ventilating the eastern part of the mine. It has three compartments 4 by 5 feet and is timbered throughout.

No. 7 shaft is west of No. 5 and was sunk from the surface to the 2,550 level to ventilate the stoping area southwest of No. 5. It has three compartments 4 by 5 feet and is timbered throughout.

No. 8 shaft has been sunk from the surface to the 4,800 level, and it is the exhaust shaft for the west section of the mine. It has four compartments 5 by 5 feet square. The sets are of steel, and the shaft is smooth-lined with 2- by 2-inch lagging.

Nos. 2, 3, 5, 6, and 7 shafts are air intakes in the ventilation system, and Nos. 4 and 8 are exhausts. Nos. 2, 3, and 5 are operating shafts, and the others are for ventilation only.

Nos. 4 and 8 shafts are equipped with 8- by 4-foot Jeffery fans which exhaust 330,000 cubic feet of air per minute from the mine.

Drifts and crosscuts: Down to the 2,000 level all drifts and crosscuts were small, generally 5 by 7 feet. As depth and higher rock temperatures necessitated better ventilation, all drifts from the 2,000 level down have been driven 8 by 8 feet in the clear. This construction permits a greater volume of air to pass through and reduces the mine resistance. In general, the ventilation system consists of taking the fresh, or intake, air to the lowest possible level by using booster fans, then letting it ascend through the working places through the stopes and stope raises to the exhaust shafts.

The efficiency of air distribution is maintained by preventing recirculation by the use of air locks. Standard doors are built of a double thickness of 1-inch boards, one layer vertical and the other at 45 degrees to the vertical, with a thickness of roofing paper between the layers. Leakage around the doors is kept at a minimum by skirts, or flaps, made of discarded belting.

The Magma Copper Company has standardized on the No. 8 American high-speed fan for booster service. This fan has been found to be adequate, is easily handled in small shafts, and requires very little rock excavation to install. It is of the double-width, double-inlet type, with backward-curving blades which give a nonoverloading characteristic. Power is supplied by a 40-h.p., 2,200-volt, 25-cycle, 3-phase induction motor directly connected to the fan and operating at 720 r.p.m.

Auxiliary ventilation: In all drifts, raises, and other dead-end working places, auxiliary ventilation is necessary. For this ventilation two types of fan are used: No. 3 Troy Sirocco with forward-curved blades and 10-h.p. motors, and No. 4½ American H.S. with backward-tipped blades and 7.5-h.p. motors. Rubber-covered canvas ventilation tubing 24, 16, and 12 inches in diameter is used to deliver the air to working places. It has been the experience at Magma that efficiency of the workmen falls off rapidly when the wet-bulb temperature is above 85 degrees and the relative humidity is above 85 per cent. Wet drilling and sprinkling of muck piles to allay the dust is of course necessary, even though it tends to raise the relative humidity. Every effort is made to keep drifts and tunnels as dry as possible by confining water to ditches and keeping it from contact with the air.

In the past, occasional, but not serious, cases of heat cramps were caused by the loss of body salt through excessive perspiration. In 1936, 15-grain salt tablets were made available to all the workmen. Swallowed, usually with a drink of water, they maintain the proper saline balance in the body. Since this practice was instituted, no workman who has used the salt tablets has had an attack of heat cramps.

None of the fans have reversing facilities, as it is thought best to maintain all air currents as nearly normal as possible in case of fire. For giving underground fire alarms, ethyl mercaptan is

used. This chemical is injected into the compressed-air lines, from which it passes into the ventilation currents and reaches all working places. Ethyl mercaptan is extremely volatile and vaporizes immediately when liberated. It has a very unpleasant odor, and by its use a fire warning can be given to everyone in the mine in twenty minutes.

Air conditioning: After the 3,200 level had been developed, it became evident that artificial cooling and air conditioning would be necessary to assist ventilation. Early in 1936 it was decided to install a refrigeration plant on the 3,600 level to air-condition the 3,600 and 3,400 levels.

For this plant, two Carrier centrifugal compressor units, each of 140 tons refrigeration capacity, were selected. The refrigerant used is Carrene No. 2, monofluorotrichloromethane (CMCl_3), which is a colorless, odorless liquid at ordinary temperatures. It is nontoxic, noncombustible, noninflammable, and has a boiling point of 74 degrees. Each unit is powered with a 200-h.p., 2,200-volt, 3-phase, 25-cycle induction motor operating at 1,440 r.p.m., which is stepped up by speed-increasing gears to drive the compressor at 6,750 r.p.m.

When each unit is furnished with 200 g.p.m. of 90 degree condenser water, it will cool 350 g.p.m. of chilled water to 60 degrees. This chilled water is circulated in closed circuit to Aerofin cooling coils. Fans of 30,000 c.f.m. capacity draw air through the cooling coils, and the coils are designed to cool this volume of air 12 degrees on the wet-bulb temperature.

A third unit was installed on the 4,000 level in 1939, and three more were installed in 1941. The six units are almost identical, the only difference being in the condensing temperature. For the purpose of making this difference clear, we shall refer to three machines as Type A, in which the condenser water enters the unit at 93 degrees and leaves at 117 degrees, and the other three as Type B, in which the condenser water enters at 117 degrees and leaves at 135 degrees.

Each machine requires 200 g.p.m. of condenser water. Many unsuccessful attempts have been made to develop water both underground and from the surface. The only water available is that made by the mine, an average of about 400 g.p.m.

Most of the underground water comes from the east section of the mine where its temperature coming from the rock is from 109 to 130 degrees. This water is sprayed into No. 6 shaft where it is cooled to about 90 degrees; it then flows by gravity through ditches and pipe lines to a gathering sump on 3,600 level. It picks up some heat enroute and arrives at the sump at 92 or 93 degrees.

By using machines having different condensing temperatures, it is possible to pass the same water through two machines. The water enters an "A" type machine at 93 degrees and leaves at 117 degrees, and enters a "B" type machine at 117 degrees and leaves

at 135 degrees. With six machines this still leaves the problem of furnishing 600 g.p.m. from 400 g.p.m.

To solve this problem a five-stage spray pond was built on the 3,600 level near No. 8 shaft in the exhaust air. This cooling pond cools 200 g.p.m. from 135 to 92 degrees. The spray pond consists of separate ponds, each 8 by 20 feet, and 936 spray nozzles (3/16-in.), divided among the five stages. Condensing water from one condenser enters the first stage at 135 degrees and is sprayed. It is then picked up from this stage and sprayed into the next by a 10-h.p. Byron Jackson "Bilton" pump. This is repeated in the other stages, and the water flows from the last stage back to the original condenser water-gathering sump.

The present arrangement of the machines is two units, one Type A and one Type B on the 3,600 level, with cooling coils and fans placed to serve the 3,600 and 3,800 levels; two units, one of each type on the 4,000 level, to serve two locations on this level; and two, one of each type on the 4,400 level, to serve two different locations on this level.

The circulation of the condenser water is as follows: From the storage sump, 600 g.p.m. is pumped through the three Type A machines on the three different levels and then on through the three Type B machines, on the same levels. The condenser water from the units on 3,600 and 4,400 is returned to the 3,600 level where it is pumped out of the mine by the main mine pumps. The water from the 4,000 level units returns to the 3,600 level and goes through the spray pond for recooling.

The main mine pumps are two horizontal duplex Prescott pumps, 600 g.p.m. capacity against a 3,300-foot head, which are powered with 600-h.p. motors. One is used for pumping, while the other is a stand-by.

MINING METHODS USED AT THE MAGMA MINE⁵⁵

The mining methods used at the Magma mine prior to about 1929 have been described in several articles. Only the methods now being used are outlined in the following brief description.

Recent changes in the mining methods have related chiefly to stoping. The modifications resulted from a change in the type of wall rock, more variation in the vein width on the lower levels, and the desire for more selective mining.

Most of the Magma ore bodies occur as shoots in a strong east-west fault zone which has an average dip of about 75 degrees south. A few ore bodies occur on branches of this fault and on adjacent parallel breaks. The ore shoots range from a few feet

⁵⁵By B. Van Voorhis, Engineer, Magma Copper Company (May 2, 1944). See also E. D. Gardner and C. H. Johnson, *Copper Mining in North America*: U.S. Bureau of Mines Bull. 405, pp. 234-7, 1938.