

readily with the formation of two nitrobenzylamine bases (m. p. 128° and 169°). The nitrosate of the terpene melts at 148° – 149° and forms with benzylamine a nitrol base (m. p. 128°), which is identical with the one obtained from the nitrosochloride. Contrary to the statement in the literature, the authors were able to prepare a crystalline hydrochloride of caryophyllene (m. p. 69° – 70°). Cadinene forms a nitrosate, which melts with decomposition at 105° – 110° , and a nitrosochloride which melts at 93° – 94° .

A Preliminary Communication upon Ricinine. By THOMAS EVANS. *J. Am. Chem. Soc.*, 22, 39–46.—The author shows that the substance obtained by Tuson by extracting castor beans with boiling water, to which the name ricinine was given, is identical with ricidine, which was isolated by Schulze from the same source by a different method of extraction. The compound crystallizes in small colorless prisms, melts at 193° , and gives results on analysis agreeing closely with the formulæ $C_{16}H_{16}N_4O_4$ and $C_{16}H_{15}N_4O_4$. The formation of a chlorplatinate, described by Tuson, was probably due to the presence of an impurity in the ricinine investigated, since the pure compound does not form such a salt. It does not give a precipitate with silver nitrate, mercuric nitrate, or mercuric chloride, but on long standing a concentrated solution of ricinine and mercuric chloride yield a feathery crystalline product. By brominating ricinine a substitution-product (m. p. 232°) containing two bromine atoms was obtained. On oxidation with alkaline potassium permanganate a crystalline acid, which melted at 279° – 280° , was obtained.

Benzenestearosulphonic Acid and Other Sulphonic Acids Containing the Stearic Radical. By E. TWITCHELL. *J. Am. Chem. Soc.*, 22, 22–26.—Benzenestearosulphonic acid, $C_6H_5SO_3H$. $C_{18}H_{35}O_2$, was prepared by treating a mixture of oleic acid and benzene with concentrated sulphuric acid. The acid is a sticky, semisolid mass, soluble in water, and forms alkaline salts which are readily precipitated by acids, bases, or salts. Analogous compounds were prepared from naphthalene and phenol.

METALLURGICAL CHEMISTRY.

H. O. HOFMAN, REVIEWER.

The Ultimate and Rational Analysis of Clays and Their Relative Advantages. By H. RIES. *Trans. Am. Inst. Min. Eng.*, 28, 160–166.—The rational analysis of a clay which in its simplest form gives the three mineral constituents, kaolinite, feldspar, and quartz, consists in decomposing and dissolving the

kaolinite with sulphuric acid followed by caustic soda, weighing the residual feldspar and quartz, determining the alumina in it, and then calculating the feldspar. Its value, especially with fire clays containing a small percentage of fluxing impurities (iron, lime, and magnesia) lies in the fact that it gives much information as to the behavior of a clay in shrinking and melting, which the ultimate analysis does not give.

A Modification of Bischof's method for Determining the Fusibility of Clay, as Applied to Non-refractory Clays, and the Resistance of Fire-clays to Fluxes. BY H. O. HOFMAN.

Trans. Am. Inst. Min. Eng., 28, 435-440.—There are two leading methods of testing clays, the direct one of Seger and the indirect one of Bischof. In the former, a clay to be tested is compared with the standard series of Seger; in the latter a clay is toned up with a silica-alumina mixture until it shows in the fire the same behavior as the most refractory clay known, that of Saarau. Instead of the Saarau clay the author chose the Seger cone No. 26, which forms the line of separation between refractory and non-refractory clays. He toned up a number of non-refractory clays until they behaved in the fire in the same manner as did the Seger cone No. 26, and tested the resistance of fire-clays to fluxes by mixing weighed samples of 1.5 grams severally with increasing amounts of fluxes until they yielded to the heat at which the sample cone bent over. By this modification of Bischof's method two things can be accomplished, the finding out, first, how much high-grade material is to be added to one of an inferior character to bring it up to a certain standard, and, second, how the resistance of acid or neutral materials to the corroding influences of bases, or vice-versa, can be determined.

Does the Size of Particles Have Any Influence in Determining the Resistance of Fire-clays to Heat and Fluxes. BY H. O. HOFMAN AND B. STOUGHTON.

Trans. Am. Inst. Min. Eng., 28, 440-444.—In testing fire-clays and fire-brick in the laboratory, the first step is to grind them to an impalpable powder. The objection made to this procedure is that the behavior of a brick which is a mixture of coarse and fine material in the fire must be different from that of the same material when finely ground. The experiments made with the material just as it is going to be used in making the brick and after fine grinding, prove that in the laboratory practically the same results are obtained with respect to resistance to heat as well as to fluxes, whether the material used be finely ground or not.

Testing Lead-tin and Lead-antimony Alloys. BY J. RICHARDS. *J. Franklin Inst.*, 147, 398-400; *Iron Age*, 63, 12.—The author prepared alloys of lead with 1-24 per cent. antimony, the largest amount that will alloy with lead, cast them into bullets to be used as standards for determining by direct weight the percentage of antimony in hard lead, cast in the same mold into bullets. The anomalous fact that 2 per cent. or less antimony makes the lead-antimony alloy heavier than pure lead, was corrected by preparing a separate set of bullets of lead with 2 per cent. antimony down to pure lead, the amount in each bullet diminishing by 0.1 per cent. These bullets have a frosty appearance and are therefore readily distinguished from the others. The author prepared a similar series of lead-tin alloys in which the tin diminishes by 0.1 per cent.

A Furnace for Smelting Zincose Lead Ores. BY L. KLOZ. *Eng. Min. J.*, 68, 279-280.—The peculiarities of this furnace are that the shaft and boshes consist of cast-iron plates in which water circulates through wrought-iron pipes and that the crucible, made of cast-iron plates, is detached, being brought beneath the furnace on a truck and raised into position by jack-screws.

Notes on Silver Losses in Smelting. BY M. W. ILES. *Eng. Min. J.*, 68, 307-309, 340-341, 367-369.—This paper is replete with data from the author's experiments as metallurgist of the Globe Smelting and Refining Co., Denver, Colorado.

Notes on Lead Smelting and Gold and Silver Refining—Slag Settling Furnaces. BY M. W. ILES. *Eng. Min. J.*, 68, 667-669, 729.—The paper discusses briefly the apparatus used for separating lead-matte from slag outside of the blast-furnace, and gives some facts relating to the reverberatory settling furnace patented by the author, and also several complete analyses of fumes collected by filtering devices.

Potash and Soda Salts from Natural Decomposition of Lead Slags. BY M. W. ILES AND C. F. SHELBY. *Eng. Min. J.*, 67, 650.—At or near the bottom of the slag-heap of the works of the Globe Smelting and Refining Co., at Denver, the authors found a white, efflorescent salt which was formed by the decomposition of the slag through atmospheric agencies. The ultimate analysis gave: SO_3 , 48.45; K_2O , 28.44; Na_2O , 13.48; CaO , 1.80; MgO , 0.80; Fe_2O_3 , 0.29; MnO , 0.26; Cl , 0.47; loss on ignition, 2.40; total, 96.97. These results show that the bases were combined mainly with sulphuric acid, which had been formed by the oxidation of the sulphides held in igneous solution by the slag.

Potash and Soda Salts in Lead Slags. BY R. C. CANBY. *Eng. Min. J.*, 68, 3.—Referring to the preceding paper the author notes the formation of similar potash and soda salts on heaps of some copper matte after exposure to snow, followed by bright, warm sunshine.

Notes on Lead Smelting and Gold and Silver Refining—Fans for Handling Lead Fume. BY M. W. ILES. *Eng. Min. J.*, 68, 788–789.—The larger part of this paper deals with the details of construction of the fans used at the silver-lead works of the Globe Smelting and Refining Co., at Denver. Here the dust-laden gases from the blast-furnace, the zinc retorts, and the silver-melting furnaces, are drawn off by fans and forced through cotton filter-bags.

A Liquid from the Fume of Lead Furnaces. BY M. W. ILES. *Eng. Min. J.*, 68, 576.—The author collected at the works of the Globe Smelting and Refining Co., at Denver, yellow icicles which had formed on a very cold day at the extreme end of the flue which carries the blast-furnace gases and dust into the filtering department. They showed, after dissolving in water, the following composition in grams per liter: FeSO_4 , 88.40; MnSO_4 , 0.68; ZnSO_4 , 24.56; As_2O_3 , 1.45; NaCl , 7.26; Na_2SO_4 , 3.69; K_2SO_4 , 2.22; $(\text{NH}_4)_2\text{SO}_4$, 33.00; H_2SO_4 , 1.22; H_2SO_3 , 15.00; organic matter, 31.94; total, 209.42.

Ammonia in Flue-dust. BY R. C. CANBY. *Eng. Min. J.*, 68, 633.—The author's attention was called to the presence of ammonia in flue-dust from blast-furnaces, when mixed with milk of lime previous to bricking.

Blakemore's Results in Refining Lead by Parkes' Process. BY W. JONES. *Eng. Min. J.*, 68, 273.—Blakemore found that the drosses obtained in the softening furnace and the desilverizing kettle from base bullion rich in gold and copper, were heavy and contained large amounts of gold. The author advocates the method, common in American refineries, of matting the dross in the softening furnace (see Hofman, "Lead," 5th edition, page 435) which leaves nearly all the gold in the lead where it belongs, and greatly reduces the amount of intermediary product requiring retreatment. The author also states that he found little difficulty in separating gold from silver by zincking with base bullion that contained three times as much gold as silver.

Refining Products. BY M. W. ILES. *School Mines Quart.*, 20, 397.—This paper is mainly a table containing forty-eight analyses and assays by H. Alexander of products from the desilverization of base bullion by the Parkes process, at the works of the Globe Smelting and Refining Co., at Denver, Colo.

Silicon and Copper Silicide. BY O. J. STEINHART. *Eng. Min. J.*, 67, 710.—The paper briefly points out the uses to which silicon and copper silicide made by the Cowles Electric Smelting and Aluminum Co., at Lockport, N. Y., can be put. In making iron castings, silicon can take the place of silvery irons; in steel castings low in carbon it does away with the use of ferrosilicon or ferromanganese, which increases the carbon content. Silicon-copper with 10, 15, and 30-35 per cent. silicon is sold in the form of ingots having notches; they weigh 15-18 pounds. It can be used instead of phosphide of copper to promote the soundness of copper and of bronze castings. Small quantities of silicon, up to 0.1 per cent., make the copper stronger than pure copper without materially reducing its electrical conductivity. The following table gives some interesting facts bearing on this matter:

Per cent. silicon.	Tensile strength. Lbs. per square inch.	Electrical conductivity.
None (pure copper wire)	39,825-49,780	100
0.02-0.05	64,000	98
0.05-1.1 (with 1.10 Sn)	118,000	43
0.50	71,116	28
3.5	135,120	7

The Influence of Antimony on the Cold-shortness of Brass.

BY E. S. SPERRY. *Trans. Am. Inst. Min. Eng.*, 28, 176-190.—As a basis for the experiments, the author chose a high brass containing 60 per cent. copper and 40 per cent. zinc, which would roll both cold and hot, and added varying amounts of antimony in the form of copper-antimony alloy. The molten material was poured into a mould $\frac{5}{8} \times 2\frac{3}{8} \times 24$ inches with the necessary precautions; the plate was cleaned by scraping and then rolled. The experiments showed that brass should not contain over 0.01 per cent. antimony if it is to roll satisfactorily.

The Influence of Bismuth on Brass and Its Relation to Fire-cracks.

BY E. S. SPERRY. *Trans. Am. Inst. Min. Eng.*, 28, 427-435.—It is usually believed that bismuth is more deleterious even than antimony in casting brass. For the experiments a high brass of 60 per cent. copper and 40 per cent. zinc was chosen, and varying percentages (0.50, 0.25, 0.09, 0.05, 0.02) of bismuth were added, the alloys were cast into an iron mould to an ingot $\frac{5}{8} \times 2\frac{3}{8} \times 24$ inches and rolled when cold. The conclusions arrived at are that bismuth renders brass cold-short as well as hot-short, and is the cause of fire-cracks. High brass to be cold-rolled should not contain over 0.01 per cent. bismuth.

A New Form of Ingot Mould for Casting Brass or Bronze Ingots, with Remarks on the General Form of Ingots. BY E. S. SPERRY. *Trans. Am. Inst. Min. Eng.*, 28, 246-253.—The

author illustrates a mould made of soft gray iron weighing about 35 pounds which he found suitable for casting ingots weighing 15-25 pounds, also tongs suited both for dumping the mould and handling the ingot while hot.

Reducing-Roasting.—Its Value for Arsenic—Expulsion from Copper Ores and Mattes. BY E. W. SCHERR. *School Mines Quart.*, 21, 66-74.—The paper gives the results obtained by an oxidizing followed by a reducing roast of a mixture of arsenopyrite and a copper matte (Cu, 50.16 per cent.; Ag, 16.82 ozs.; Au, 0.18 ozs.), averaging 4.56 per cent. As. The furnace was an 18 × 24 inch American Gas Co. annealing furnace. Two lots of $3\frac{3}{4}$ kilos were brought in $4\frac{3}{4}$ hours approximately to a dead-roast in shallow iron trays, 7 × 23 inches; 10 per cent. of finely pulverized charcoal was added to one of them, and the roast continued for 1 hour and 55 minutes, when all the carbon was consumed. The temperature at the beginning, 460° C., was slowly raised to the end, when it reached 767° C. The tabulated and plotted results show that during the first two hours the arsenic had been reduced to 0.8 and 0.9 per cent., and $2\frac{3}{4}$ hours later to 0.7 per cent., when charcoal was added to one lot, while at the end of the roast both samples retained 0.5-0.6 per cent. arsenic. Copper began to be converted into sulphate one hour after starting; after 3 hours 5 per cent. was rendered soluble, the percentage of sulphate then diminished in one lot to 3 per cent.; as soon as charcoal was added to the other lot (5 hours after starting) the sulphate was quickly decomposed, leaving only 0.2 per cent. soluble copper. The author believes that during the first period arsenic was eliminated mainly as arsenious oxide, and to some extent as disulphide. The arsenious oxide was formed by oxidation of arsenide and by the reduction of arsenic oxide by sulphur, sulphur dioxide, and ferrous oxide, having been separated from its base by sulphur trioxide present in excess. The arsenic remaining in the ore is believed to be in combination with copper as arsenate, in which form it is decomposed with great difficulty by a reducing roast.

Roasting Copper Ores at Keswick, California. BY TH. NEILSON. *Eng. Min. J.*, 68, 457.—The paper gives interesting data about roasting sulphide copper ores in heaps, stalls, and the Ropp and McDougal mechanical furnaces. The ore averages: Fe, 38; Cu, 8; Zn, 4; S, 46; SO₂, (?) and Al₂O₃, 3 per cent.; Ag, 1.5 ozs.; and Au, 0.03 ozs. per ton. Stalls 14 × 7 feet and 6 feet high, 170 in number, with a capacity of 6,000 tons of ore, were first used, but were replaced by heaps as the matting of the ore and the expense of discharging made the work too costly. The ore coming from the mine is charged into breakers with a grizzly with 3-inch spaces and a screen with $\frac{3}{4}$ -1 inch

openings, thus furnishing coarse, medium, and small-size ore, which is collected in separate bins. From these it is run on a trestle, with bents 15 feet apart, over the roast-yard, where the heaps 20 feet wide, 8 feet high, and, varying with the nature of the ground, from 500-1,500 feet long, are erected. Fines are spread to a depth of 6-12 inches, then follows the bed of dry free-burning light wood (pine, fir, cedar, manzanita) 6 inches square in four-foot lengths. Longitudinal flues 4 inches wide, and, at eight-foot intervals, similar cross flues, are left open to be fired with kindling. At their intersections chimneys $8\frac{1}{2}$ feet high are erected of old boards or of cord wood, and are filled with kindling. The fines are covered with cord-wood, and small wood is piled around the intersections of the flues. Coarse ore is now dumped onto the bed, and a wall six inches high of lump ore, or waste rock, is built outside of the wood, with arches over the flues. Coarse and medium ore follow, until the heap has nearly reached its normal height; it is followed by a cover of medium ore and then by one of fine ore about two inches in thickness. The heap being erected, the stringers (6×8 inches and 16 feet long), ties (4×4 inches), and rails (16-20 feet long), are removed to be used for another heap, while the poles (4-6 inches in diameter) are left, as the cost of recovering them is too great. The heap is fired by igniting one-half pint of coal oil poured into each of the chimneys. In one hour after starting the ore begins to burn. A heap burns about six months, reducing the sulphur to 12 per cent., of which 2 per cent. is present as sulphate. One cord of wood is sufficient for 120-150 tons of ore. When roasted, the ore is loaded by means of a $\frac{3}{4}$ -yard steam shovel into cars and goes to the blast-furnace. Drilling holes 10 feet apart under the heap and blasting loosens up the roasted ore and helps the work of the steam-shovel. Building the heaps as described costs 17.9 cents per ton; but this has been reduced to 9.31 cents by doing the screening at the mine. Discharging the heap by hand costs 23 cents, by steam shovel 4.92 cents per ton. The two McDougal furnaces, 15 and 18 feet in diameter, roast 20 and 35 tons of fines (under one-half inch size) in 24 hours at a cost of 12-15 cents per ton for labor, at \$1.85 per 10 hours. To the labor cost in the Ropp furnace, which is about the same as in the McDougal, 30 cents has to be added for cord-wood at \$3.00 per cord. Granulated matte is treated in both furnaces.

The Bretherton Blast-heater. BY S. E. BRETHERTON. *Eng. Min. J.*, 68, 689.—An illustrated description is given of an apparatus utilizing the waste heat of a movable forehearth for heating the blast.

A Study of the Elimination of Impurities from Copper Matte in the Reverberatory and the Converter. BY E. KELLER. *Trans. Am. Inst. Min. Eng.*, 28, 127-159; DISCUSSION, 816-840.—In bringing forward 60 per cent. copper matte to metallic copper in the reverberatory furnace, the first step is to crush the matte fine and to roast it in a hand reverberatory furnace. As in large-scale work the complete clean-up necessary to arrive at satisfactory data as to losses in roasting is not practicable, the author made some small-scale experiments in dead-roasting a matte of the following composition: Cu, 60.89; S, 23.22; Fe, 12.28; Fe_2O_3 , 0.34; Zn, 1.70; Pb, 0.568; Bi, 0.0501; Sb, 0.1010; As, 0.0481; Se, Te, 0.0101; Ag, 61.1 ozs.; Au, 0.20 ozs. He found by comparing raw and roasted matte that 5 per cent. of the arsenic and 15 per cent. of the antimony had been volatilized. By bringing the analyses of flue-dust and raw matte to the common basis of Cu = 100, and dividing the former by the latter, he found the relative elimination of the elements to be Cu, 1.00; Zn, 1.86; Pb, 1.88; Bi, 1.60; Sb, 8.15; As, 27.16; Se, Te, 2.80. Upon the assumption that the elimination of arsenic is proportional to that of sulphur, and that in actual roasting the sulphur content is reduced one-third, there will be found in the flue-dust the following percentages of the original quantities in the matte: Cu, 0.18; Zn, 0.34; Pb, 0.34; Bi, 0.29; Sb, 1.47; As, 5.00; Se, Te, 0.50. In smelting the roasted matte with silicious and calcic fluxes for bottoms, there are also formed regulus, slag, and incidentally some flue-dust. The author finds that arsenic concentrates three times as much as antimony or bismuth in the bottom, while lead, selenium, and tellurium concentrate in the regulus. The order in which the foreign constituents are eliminated from the regulus by slagging, volatilization, and draining into the bottom is the following: zinc, lead, bismuth, antimony, arsenic, selenium, tellurium, gold, and silver. In smelting the regulus and bottoms together in a reverberatory furnace for blister copper, the order of elimination of impurities by slagging is antimony, lead, selenium, tellurium, bismuth, and arsenic. In refining the blister copper, the impurities, slagged on the basis of 7 units slag to 100 copper, were Pb, 0.0678; Bi, 0.0009; Sb, 0.0115; As, 0.0020, while those remaining in the refined copper were Pb, 0.0093; Bi, 0.0320; Sb, 0.0651; As, 0.0586; and Se, Te, 0.0098. In refining converter copper, the order of slaggability was lead, antimony, arsenic, bismuth, and selenium-tellurium. After discussing the reduction of slags from the reverberatory process in the reverberatory furnace and of refinery slags in the blast-furnace, the author sums up the elimination of impurities in the total reverberatory process of copper smelting by comparing the values of the raw matte with those of the refined copper and

shows that the following percentages are eliminated by it: Pb, 99; Bi, 54; Sb, 50; As, 21; Se, Te, 60. In converting matte of a composition similar to that brought forward in the reverberatory furnace, the author shows that the following percentages of impurities are eliminated: Pb, 95-99.4; Bi, 94-96; Sb, 62-73; As, 73-91; Se, Te, 57.71; varying with the size of the converter and the extent to which the blowing was carried, a few minutes' overblowing being more efficient than hours of flapping in the reverberatory furnace. In the discussion of the paper the author tells of his laboratory experiments on the elimination of impurities by the James Direct Process. His results are unfavorable to it.

The Jackson System of Treating Ores. BY A. PHILLIPS. *Eng. Min. J.*, 68, 34; BY VAN ZWALUWENBURG, *Ibid.*, 68, 363-364.—A silver-gold ore is to be dead-roasted in a mechanical roaster with stationary hearth; at the discharge it is to drop into a heated chamber, being at the same time supplied with salt and iron sulphide, which are to do the chloridizing. After passing through the chamber, the chloridized ore is to be cooled by passing through the open air, then fed into tanks, leached with brine and sodium hyposulphite, and the gold precipitated by some electrical means. Van Zwaluwenburg, who experimented at a large lixiviation plant in Mexico with the electrical precipitation, found it unsatisfactory as regards the deposition of silver and that, in addition, the hypo salt was decomposed.

The Alluvial Deposits of Western Australia. BY T. A. RICHARDS. *Trans. Am. Inst. Min. Eng.*, 28, 490-537.—This paper, mainly of a geological character, has a number of interesting illustrated descriptions of the separation of placer gold from sands by dry-blowing.

Water Required for Stamp-milling. BY G. W. RITER. *Eng. Min. J.*, 68, 278.—The author gives some data from his experience with ores from Eureka Hill, Tintic District, Utah, on the water required by the combination process, *i. e.* stamping, concentrating, and treating of settled tailings by pan amalgamation. The amounts required per ton of ore were $1\frac{3}{8}$ tons for crushing, concentrating, amalgamating, and retorting; and, if the steam required for power, drying concentrates, heating pans, etc., be included, this figure was raised to nearly 2 tons.

Silver-Plating of Amalgam Plates for Gold Mills. BY A. J. CLARK. *School Mines Quart.*, 21, 48-56.—The paper is a description of a method used for replating the silver-plated amalgam plates (54 inches wide and 8-12 feet long) of the mills of the Homestake and Highland Mining Cos., at Lead City, S. D., after these

had shown a decrease in recovery of gold as soon as the original plating had begun to wear off. In order to have the silver adhere firmly to the copper plate, this must be absolutely clean. On a new plate, any scale rolled into the surface would cause the silver to form a rough deposit easily rubbed off in the daily collection of amalgam. A thin film of oxide or grease would give a somewhat uneven deposit, quickly washed off by the stream of pulp. On an old plate, after scraping off the remaining amalgam, the quicksilver has to be removed. Scraping by hand with acid treatment proved unsuccessful. Stripping (solution of any gold, silver, mercury, by the anode action of an electric current using iron plates as cathodes and potassium cyanide as electrolyte) proved successful as far as the cleaning was concerned, but it left the copper plate pitted, and while it took a good silver deposit, when used in the battery it did not work satisfactorily, the heavier sands settling in the hollows. Stripping had to be supplemented by a power scraper. A buff wheel with an emery-cutting surface was used, a small fan sucking off the cuttings and discharging them into a box filled with water. The scrapings were amalgamated in a Knox pan yielding a bullion with Au, 5-14, Ag, 14-23, and Cu, 72 per cent. Two men clean three plates in ten hours. The plates were painted twice on the reverse side with a solution of asphaltum in turpentine and then brought to a slightly inclined working table having cross strips of wood covered with rubber and a gutter to carry off the washings. The wood was painted with asphalt, and the whole placed under a hood connected with a fan. The scraped side of the plate may require scouring with sand or pumice stone to remove scale (the sand going to the mill), acid treatment (equal amounts of nitric and sulphuric acids with 1-5 per cent. of hydrochloric acid) to remove remaining oxides or mercury, a hot concentrated potash or soda wash to remove grease, a painting with potassium cyanide followed by one with cyanide of mercury, which tends to improve the adherence of the deposit. The depositing vat was $15\frac{1}{2}$ feet long by 30 inches wide; it was made of 3-inch red-wood, heavily strapped, and lined with a layer of asphaltum by melting and pouring it in and ironing it out. The tanks were found to spread $2\frac{1}{2}$ inches when filled, necessitating loosening the nuts of the straps. The depositing vat was set in a cement vat 3 feet deep to catch any leakage. There was a tank of pine wood, $10 \times 5 \times 5$ feet, painted with asphalt into which the solution could be siphoned when it might require to be renovated. A plate was suspended by long hooks (passing through holes punched 2 feet apart along the edge) from a longitudinal bar held at the ends by a differential pulley and supported in the center by a lever. The electrolyte was a solution of silver chloride in potassium cyanide carryin

2.5 ozs of silver and one-half pound of potassium cyanide to the gallon. From 15-20 silver plates, 3 feet by 6 inches and one-fourth inch thick, were used as anodes for one copper cathode. With a single plate and an electrode distance of 16-17 inches, an electromotive force of 1.5 volts was ample. With a pair of plates hung back to back with anodes on either side and an electrode distance of 10-12 inches, 2-2.25 volts were necessary. The rate of deposition was 0.35-0.50 oz. per square foot per hour. A new solution did not work so well as one that had been used for some time. Every day the solution had to be stirred to correct a tendency of layering. After the deposition has been going on for 10 or 15 minutes the plate is drawn out and any imperfections corrected by scouring. The composition of the electrolyte has to be checked by determining the silver and free potassium cyanide.

De LaMar's Mercur Mines. By H. L. J. WARREN. *Eng. Min. J.*, 68, 754-756; 787; 788.—The paper discusses the occurrence of the ore, the mining and the metallurgical treatment. There are two varieties of ore, oxidized and base; a third class called mixed ore, a talcose oxidized material, is recognized in the mill. The gangue consists of quartzite, limestone, shale, and clayey talcose material; it contains 20-30 per cent. SiO_2 . The oxidized ore is yellowish to brownish, the base ores carry 1-5 per cent. realgar and orpiment, cinnabar is occasionally found. The value in gold shows a range of \$2.00-\$60.00, and usually varies from \$3.00-\$12.00 per ton. The ore is passed through crushers, roughing rolls, and finishing rolls with intervening screens, leaves the last reduced to a limiting sieve with one-eighth inch holes. The oxidized ores are leached raw, the mixed ores are calcined in two Holthoff-Wethey mechanical furnaces (100 × 12 feet), passing through them in six hours. Each treats 145 tons in 24 hours, and consumes 8 tons of coal. The object of calcining is to remove the combined water and thus render the ore readily leachable. The base ores with 2-5 per cent. sulphur and 0.95-2.50 per cent. arsenic are roasted in 4 Jacklin mechanical furnaces (120 × 12 feet) passing through them in 8 hours. A furnace roasts 70 tons in 24 hours, consuming 7 tons of coal, and reduces the arsenic to 0.10-0.15 per cent. The oxidized, calcined, and roasted ores are collected together in a general charging bin whence they go to the leaching vats. There are 26 steel leaching tanks (50 × 25 feet and 5 feet deep), each holding 250 tons of ore. On the bottom are spread 50 tons of cherty material, making a 10-inch filter bed. A tank is filled from side dump-cars by 3 men in 8 hours. First a strong 0.4 per cent. solution of cyanide is turned on from the bottom through 10 2-inch openings; it takes 8 hours to rise to the sur-

face, stands 16 hours, and then leaching by percolation is begun and continued for 24 hours. The first leaching is followed with a weak (0.3 per cent.) solution until the gold has been extracted, which takes two or three days, after which the wash-water is turned on. The rate of washing is one inch per hour, the weight of solution and wash-water used is about twice that of the ore. The leached ore is shoveled through openings in the bottom into cars by ten men in four hours. Solutions and wash-water flow into two collecting tanks (20 feet in diameter and 12 feet deep), whence they are pumped into three precipitating tanks (16 feet in diameter and 8 feet deep). The gold is precipitated by means of zinc dust. While the solution flows into a tank it is kept stirred by compressed air passing at a pressure of 20 pounds through a one-half inch pipe. When a tank is filled with its first solution, 30 pounds of zinc dust are given, with the second charge 20 pounds, and with the third and the rest 10 pounds, 15 pounds being required on the average for a full charge of liquor, which weighs 30 tons. When the precipitation is finished, the solution is allowed to settle and is passed through ten filter-presses (Stillwell-Bierce and Smith-Vaile pattern, 24 inches square) into a sump 24 feet in diameter and 8 feet deep. The solution before precipitation contains \$3.00 gold per ton, and after precipitation \$0.20; the zinc consumption is $1\frac{1}{3}$ pounds per ounce of gold. Before bringing the low solution to the normal standard by the addition of fresh potassium cyanide, one pound of caustic soda is added to a ton of solution so as to preserve its alkaline character. At the monthly clean-up, the precipitate from the vats and filter-presses is dried and heated in shallow pans in a muffle furnace to oxidize the zinc, and then crushed to pass a one-fourth inch screen. It contains about 20 per cent. gold; there is always present 1 part silver to 100 parts of gold. The crushed material is treated with sulphuric and nitric acids and brought up to 60 per cent. gold. It is now filtered in a press, dried in a muffle, broken to one-half inch size, mixed with flux (1 potash, 2 soda, and 2 borax), and melted down in a graphite crucible. The gold is 950 fine.

Mill Practice of the Utica Mills, Calaveras County, California.

By W. J. LORING.—*Trans. Am. Inst. Min. Eng.*, 28, 553-565. —The paper gives full practical details of the manner of working at the three mills containing 120 stamps in all. Of special interest is the use of shoes of manganese-steel, 8.5 inches in diameter of face and 10 inches high, which last on the average 296 days, weighing 177 pounds when new and 28 pounds when used up so that they have to be exchanged. Manganese-steel has also done better work in the crushers than any other kind of iron. A stamp weighs 835 pounds, drops 7-8

inches 100 times a minute, crushes per day 4.8 tons through a fine, round-punched screen (corresponding to 30-mesh), requires per battery of five stamps 8.5 gallons of water per minute, and 2.5 gallons per minute for the two concentrators belonging to it. The consumption of quicksilver per ton of ore is 0.076 ounce. Concentrates from the vanners and the Gates canvas-plant are chlorinated and the slimes finally treated with potassium cyanide. The cost of milling, exclusive of power, is 13.8 cents per ton. For further details the reader must be referred to the original paper.

Cyaniding Slimy Ores and Tailings. BY M. W. ALDERSON. *Min. Sci. Press*, 78, 584, 612, 636, 665; 79, 4.—The author discusses first the early attempts at cyaniding slimes in Montana. He then takes up the treatment by agitation and illustrates the working of a vertical agitator with steel propeller blades, rotating at the rate of 16 revolutions per minute in a vat 16 feet in diameter and 5 feet deep, holding 20 tons sands or 12 tons of slimes. With the ore are charged 12 pounds of lime to neutralize any active acidity and to aid in settling the slime. For a ton of ore there is required usually one ton of solvent, rarely 1.5–2 tons. The agitation is continued until a sample of solution shows the enrichment it ought to have, this being ascertained by preliminary tests. When agitation has been stopped, the charge is allowed to settle. After 1–2 hours, decantation can be begun by removing the plugs one after another from the 3-inch auger-holes bored in the sides. Decantation is continued until there remains not more than one-half inch of clear solution on top of the slime. The decantation of the strong solution is followed by two washes of 2 or 3 tons of water per ton of ore. The whole operation takes about 30 hours. The tailings are discharged at the bottom into a launder. The gold solution is collected in a vat with floating drain and passes through the zinc boxes. Trouble is here encountered on account of the very weak solutions, the original solution being greatly diluted by the decantations. The author overcame the difficulties by periodically allowing the zinc shavings in the extractor to stand in contact with a solution very much stronger than the normal (one-half pound cyanide to 1 ton solution) in order to remove incrustations formed by the weak normal solution. He did this by placing, every 6 hours, 1.5 pounds potassium cyanide in the head compartment. The extractor, 3 feet wide by 12 feet long has 7 compartments, the first four are 18 × 18 inches, the last three 12 × 18 inches, and treats 30 tons solution in 24 hours. Most of the cyanide works send their precipitated gold to a United States assay office to be refined. The author, after experimenting in vain with sulphuric acid and hydrochloric acid,

tried nitric acid, which worked satisfactorily. The dissolved silver was precipitated by salt and the chloride decomposed by fusion, etc.

Cyaniding at Gilt Edge, Montana. BY M. W. ALDERSON. *Min. Sci. Press*, 79, 408.—The gangue of the ore is porphyry and altered limestone averaging 20 per cent. SiO_2 . The porphyritic ore assays \$3-\$10 and the limestone as high as \$300 per ton, average ore assay \$7.00. Everything down to \$3.00 goes to the mill, which has a capacity of 160 tons per day. The ore reduced to one-fourth inch size is charged into six steel vats 28 feet in diameter and 6 feet deep, each holding 70 tons of ore. It is first treated with a solution containing 4 per cent. cyanide (one-quarter ton solution to 1 ton ore), which remains 8 hours in contact with it. This is followed by the leaching solution, 0.24-0.25 per cent. cyanide, which percolates in 52 hours, and the leaching solution by two wash-waters of 6.5 tons each. The ore when charged contains 4 per cent. water, when drained, 13 per cent. In the leaching room there are besides the ore vats, two solution tanks (12 feet in diameter and 4 feet in height), one sump tank (14 X 4 feet) and one water tank (14 X 12 feet). In the precipitating room are six zinc boxes, the precipitated gold is treated with sulphuric acid, the refined bullion is 900-1000 to 950-1000 fine. The tailings average \$1.00. The cost of treatment in July, 1899, was \$1.32 per ton, a total of 2800 tons being handled. There was consumed about one-eighth pound potassium cyanide per ton of ore.

The Pneumatic Cyanide Process. BY J. WEBB. *Eng. Min. J.*, 68, 4.—The idea of the process is to stir with compressed air a warmed cyanide solution, thus permitting a quick treatment of slimes or ores crushed very finely, and a degree of aeration otherwise not attainable. The process is still in the experimental stage.

The Pneumatic Cyanide Process. BY A. DEL MAR. *Eng. Min. J.*, 68, 94.—The author criticizes the process on account of the greatly increased consumption of cyanide which must follow from the continuous air-stirring of the warmed solution.

Notes on Tuyeres in the Iron Blast-furnace. BY J. M. HARTMAN. *Trans. Am. Inst. Min. Eng.*, 28, 666-673, 902-907. BY B. F. FACKENTHAL. *Ibid.*, 28, 673-678, 858-872.

Note on the Use of the Tri-axial Diagram and Triangular Pyramid for Graphical Illustration. BY H. M. HOWE. *Trans. Am. Inst. Min. Eng.*, 28, 346-355, 894-901.—In representing graphically the properties of slags the tri-axial diagram

serves an excellent purpose. To illustrate it, the author has plotted the total heats of solidification of an extended series of lime-alumina-silicates examined by Prof. Åkerman. The lines joining the points of equal total heat of solidification, called "isocals," tell a clear story of the relation between composition and total heat of solidification. The author further outlines the use of a triangular pyramid for the graphical representation of four variables.

Analysis of Blast-furnace Gas while Blowing In. By R. H. SWEETSER. *Trans. Am. Inst. Min. Eng.*, 28, 608-613.—The paper is a record of analyses of the gas made during the blowing-in of the furnace "C" of the Maryland Steel Co., at Sparrow's Point, Md., the time covered being from 12.26 P.M. of one day to 2.45 P.M. on the following day. With the analyses are tabulated the revolutions of the engine, the pressure and the temperature of the blast, and the height of the stock. These figures, as well as the conditions of normal work, are graphically represented to facilitate comparison.

Note on the Forms Assumed by the Charge in the Blast-furnace, as Affected by Various Methods of Feeding. By F. FIRMSTONE. *Trans. Am. Inst. Min. Eng.*, 28, 370-395.—This is an extended illustrated paper on the effects of different methods of feeding on the distribution of ore, flux, and fuel in the blast-furnace, and on the working of the furnace itself.

Note on Slips and Explosions in the Blast-furnace. By F. B. RICHARDS. *Trans. Am. Inst. Min. Eng.*, 28, 604-608, 911-919.—Two illustrations accompanying the note represent the Claire furnace of Sharpsville, Pa., one while in normal condition and the other 5 seconds after an explosion or slip had taken place. The second picture shows the tremendous force of the accidents caused by the use of fine Menominee and Mesabi ores. The furnace, 75 feet high, with a 16-foot bosh, a 10-foot 6-inch hearth, and 13 tuyeres, was working an ore mixture containing 37.5 per cent Mesabi ore, 25 per cent. of which would pass through a 100-mesh sieve, the balance coming from the Menominee and Marquette ranges.

The Manufacture of Sulphur-free Mineral Wool. By A. D. ELBERS. *Eng. Min. J.*, 68, 248-249.—The paper gives a review of the manufacture of mineral wool and discusses the process invented by the author for freeing it from calcium sulphide, a substance which has prevented mineral wool from being used as much as it deserves. The process consists in remelting iron blast-furnace slag with moderate quantities of gypsum or other alkaline earth sulphates, when the reaction $3\text{CaSO}_4 +$

$\text{Ca}(\text{Mg}, \text{Fe})\text{S} = 3\text{CaO} + \text{Ca}(\text{Mg}, \text{Fe})\text{O} + 4\text{SO}$, takes place, and the sulphur dioxide is carried away by the gases, the atmosphere in the cupola being sufficiently oxidizing to prevent its reduction. Some of the gypsum charged cannot but be superficially reduced to calcium sulphide, and has to be reoxidized by the remaining sulphate; thus more gypsum is required than is called for by the formula.

The Utilization of Blast-furnace Slag. BY A. D. ELBERS. *Eng. Min. J.*, 67, 649-650, 708.—The author discusses the possible uses to which iron blast-furnace slags have been and can be put, and then takes up their desulphurization.

The Kytchym Medal. BY P. FRAZER. *Trans. Am. Inst. Min. Eng.*, 28, 613-617, 848-854.

Blast-furnace and Cupola Practice. BY E. S. COOK. *J. Am. Foundrymen's Assoc.*, 6, 295-313; *Iron Age*, 63, May 11, 1899.—The paper compares in a general way the work in the blast-furnace and the foundry cupola. The following points are not generally known: 1. Iron run into chilled molds contains more graphitic carbon than when run into sand beds. Thus cast No. 81, of the Warwick furnace, showed in chilled molds: graphitic C, 2.87; combined C, 0.48; total C, 3.35; Mn, 0.42; P, 0.36; S, 0.017; Si, 1.50; and in sand molds graphitic C, 2.57; combined C, 0.42; total C, 2.99; Mn, 0.41; P, 0.36; S, 0.010; Si, 1.52, the reason being that the iron molds are dry and frequently warm. 2. In casting by a machine, the cold iron molds will have a chilling effect on the iron, and disturbing the crystallization, will interfere with the usual grading by fracture, hence a new method of grading (by composition) becomes necessary. The leading components being silicon and sulphur, the grading ought to be as follows:

	No. 1.	No. 2x.	No. 2, soft.	No. 2, plain.	No. 2, strong.	No. 2 foundry.
Silicon....	2.50-3.00	2.00-2.50	2.50-3.00	2.00-2.50	1.50-2.00	1.50-2.50
Sulphur ..	0.02 and less	0.02 and less	0.05 and less	0.05 and less	0.05 and less	over 0.05

3. The furnaceman as well as the foundryman finds a guide for the quality of the pig in casting chill-cup test-pieces, as iron of certain grades will not take a chill. At Warwick, for example, iron low in manganese with 1.5 per cent. Si and that with 0.03 per cent. S will not take a chill; with 1.00 per cent. Si and 0.02 per cent. S there may be a slight chill; with 0.08 per cent. S the chill will be one-half inch deep. Pig with 2-3 per cent. Si and 0.05 per cent. S will not chill; with 0.10 per cent. S a chill will be seen in spite of the high percentage of silicon. The chill produced by sulphur is very different from that resulting from a lack of silicon.

Modern Cupola Practice with Special Reference to the Discussion of the Physics of Cast-iron. BY B. S. SUMMERS. *Trans. Am. Inst. Min. Eng.*, 28, 396-413, 769-796, 884-893.— In foundry practice it is held by many that the silicon is the governing element; the author disagrees with this opinion, believing that graphitic carbon controls the value of a casting. While the most prominent function of silicon in foundry iron is that it promotes the formation of graphitic carbon, it also lowers the saturation of iron for carbon. Machinery castings of close texture should contain between 1.50 and 2.00 per cent. silicon and may reach 2.25 per cent.; light hardware may contain as much as 3 per cent. silicon; with more than 3 per cent. silicon, castings become brittle; for ordinary castings sulphur, which interferes with some of the good effects of silicon, should not exceed 0.10 per cent., and had best be kept below 0.08 per cent. While pig-iron rarely contains a harmful amount of sulphur, it is often introduced into the mixture through ferrosilicons, some well-known brands of which the author found to contain 0.17-0.18 per cent. The limit of phosphorus for most cases is about 1.00 per cent; where great strength is required the percentage ought to be much lower. As to manganese, 0.80 per cent. will hardly be noticed and it may reach 1.00 per cent. On account of the property of manganese to counteract the formation of blow-holes or the iron becoming spongy, the use of manganese is desired so long as it does not harden the iron. As to the forms of carbon, the author is inclined to attribute a considerable importance to amorphous temper carbon, and believes that the closer and denser fracture of charcoal-iron over coke-iron may be attributed to it. The reviewer would point out that the microphotographs of Glinz, published in *Stahl und Eisen*, 1899, page 1062, show that this difference in fracture is due rather to the more uniform distribution of graphite than to the presence of temper carbon. As to the relation of silicon and graphitic carbon, the author shows by analysis that an increase of silicon does not have a corresponding increase in the graphitic carbon; in fact, records for months in works show that silicon varying between two and three per cent. has much less effect on the separation of carbon as graphite than is usually supposed. The effect of silicon seems to be governed more by the temperature at which it is allowed to act. The higher the coke consumption up to a certain limit, and with it the larger the volume of blast, the greater will be the total carbon and the percentage of graphite. Rusty scrap-iron, especially if much corroded, has a tendency to make iron spongy.

The Manufacture of Car Wheels. BY G. R. HENDERSON. *The Foundry*, 15, 48-53.—The paper discusses the control by

mechanical tests and chemical analyses of the materials used in the manufacture of the finished product. Bars of wheel-iron 2 inches square by 24 inches should stand, with supports $21\frac{1}{2}$ inches apart, a center load of 12,000-14,000 pounds before breaking, showing a deflection of 0.2 inch at center; in testing, the amount of chill, the shrinkage, and the fracture should be noted. Car wheels should show the following limits in chemical composition: graphitic C, 2.75-3.00; combined C, 0.50-0.75; Si, 0.50-0.70; Mn, 0.50-0.75; S, 0.05-0.07; and P, 0.35-0.45 per cent.

The Malleable Iron Industry: Its Development in the United States. BY G. C. DAVIS. *J. Am. Foundrymen's Assoc.*, 6, 59-67.—This paper is a historical one. In 1853 there were five foundries in this country making malleable castings. At present there are ninety with a capacity varying from one to eighty tons per day.

The Basic Furnace for Malleable Iron. BY E. C. WHEELER. *The Foundry*, 14, 247-250.—The author advocates the use of an 8-10 ton basic open-hearth furnace for foundries having a tonnage of 60-100 tons per day. A heat would be tapped into a ladle attached to an overhead traveling crane, and its contents poured into small ladles when the moulder's floor had been reached. While pouring, the melting furnace would be recharged and the fusion well under way before the pouring was finished.

Suggested Improvements in the Manufacture of Bar-iron. BY S. PETERS. *Proc. Eng. Soc. Western Pa.*, 15, 222-230; *Iron Age*, 63, May 11, 1899, 14-15.—In puddling gray-forged iron, 500-600 (sometimes 1,000) pounds of iron ore are required per ton of muck-bar, exclusive of tap-cinder and squeezer-cinder. This forms a very large amount of slag which is liable to retard somewhat the process; a lack of cinder is perhaps still more harmful, as the impurities will not be sufficiently diluted in the cinder, and will with the slag reenter the puddled ball; further, the refining will be retarded, and hence the welding of the granules. The improvement suggested is to melt down the pig in a cupola, transfer to a Bessemer converter, and desiliconize and mostly decarbonize (Si, 0.05; C, 0.5; P, 0.5; S, 0.05), transfer the liquid metal to a rotary reverberatory furnace lined with bauxite or magnesite, remove with small addition of raw ore, carbon and phosphorus making as acid a slag as is permissible (say 15 per cent. SiO_2). By delivering to the balling furnace metal in definite quality and of definite composition, and by regulating the quality and quantity of the slag, more uniform

results will be obtained than at present; moreover, more work can be done on a large scale.

The Relations between the Chemical Constitution and the Physical Character of Steel. BY W. R. WEBSTER. *Trans. Am. Inst. Min. Eng.* **28**, 618-665, 876-883.—The author gives a review of the subject accompanied by a general discussion and tables, to which the reader is referred.

Notes on the Bertrand-Thiel Process. BY J. HARTSHORNE. *Trans. Am. Inst. Min. Eng.*, **28**, 254-264.—The paper is a report on the progress made with this process since 1896. It gives the titles of the papers published on the process, discusses them briefly, and summarizes the results obtained at Kladno, using pig and scrap, cold pig iron alone, and molten metal from the blast-furnace.

An Explanation of the Rapidity of the Bertrand-Thiel Process. BY H. M. HOWE. *Eng. Min. J.*, **68**, 276-277.—In this process, which is a modification of the basic open-hearth process, cast-iron is melted down in one furnace and the elimination of phosphorus and silicon started; scrap and ore are melted down at the same time in a second furnace situated at a lower level, where the iron becomes somewhat oxidized; now the metal from the upper is tapped into the lower furnace and its slag diverted during the flow. The two charges react quickly, and the carbon and phosphorus are rapidly removed. By the removal of silicon and the partial elimination of phosphorus in the upper furnace, with a slag that is not very basic relatively little lime has to be added with the result that a smaller quantity of slag per unit of iron is produced than in the regular practice, which aims to remove the whole of the phosphorus in one operation. This smaller amount of slag leaves more room in the furnace for the metal charge. The partly treated metal charge free from slag meets, in the lower furnace, iron oxide containing very little slag, hence the reaction is vigorous and the amount of slag formed small, which again leaves more room for the metal. The higher temperature over that in the ordinary process is explained by the higher initial temperature of the substances brought together. The frothing of the charge in the lower furnace during the boiling period is less, on account of the smaller amount of slag and its higher temperature.

ASSAYING.

H. O. HOFMAN, REVIEWER.

Experiments in Sampling Silver-Lead Bullion. BY G. M. ROBERTS. *Trans. Am. Inst. Min. Eng.*, **28**, 413-427.—The