United States Patent Office

3,456,792

Patented July 22, 1969

3,456,792

METHOD FOR RECOVERING CHALCOPYRITE AND PYRITE FROM COMPLEX MAGNETITE ORES

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No Drawing. Filed Mar. 7, 1966, Ser. No. 532,084

Int. Cl. B03d 1/02, 1/08

U.S. Cl. 209—167 4 Claims

ABSTRACT OF THE DISCLOSURE

A two-phase froth flotation process wherein chalcopyrite and pyrite are separated as a froth product from complex magnetite ores in a first froth floation step and the chalcopyrite is separated from the pyrite as a froth product in a second froth-flotation step, said separation being made possible by conditioning the slurry prior to the second froth-floation step with lime and air.

This invention in general relates to an improved method of recovering metal sulfides by froth flotation and to the recovery of chalcopyrite and pyrite from complex ores containing the aforementioned sulfides. In particular it relates to the recovery of such sulfides from complex magnetite ores by a two phase bulk froth flotation and selective froth flotation method.

Certain complex magnetite ores contain economically recoverable quantities of chalcopyrite ($CuFeS_2$) and pyrite (FeS_2). In the prior art practice magnetite and a portion of chalcopyrite are separated from the ore slurry by magnetic means. The chalcopyrite in turn is separated from the magnetite by froth flotation. The chalcopyrite remaining in the magnetite tailings and the pyrite therein are removed by a froth flotation process which comprises first treating the slurry with lime and cyanide to depress the pyrite and float the chalcopyrite and then cleaning the surface of the pyrite with copper sulfate. The cleaned pyrite is then floated by adding a collector for example sodium ethyl xanthate and a frother such as pine oil to the slurry.

The above selective froth flotation practice of magnetite tailings is complicated, cumbersome and requires large amounts of reactive agents such as collectors, depressants and frothers. The satisfactory separation of chalcopyrite and pyrite is dependent upon the use of a toxic cyanide compound. The process is not only expensive but requires a large labor force for the recovery of chalcopyrite and pyrite.

It is the primary object of this invention to effect the recovery of chalcopyrite and pyrite from complex ores by a method which utilizes bulk flotation of the raw ore to remove all the sulfur bearing particles therefrom and selective flotation of the chalcopyrite and pyrite.

It is another obpect of this invention to effect an economical recovery of chalcopyrite and pyrite from complex ores by a method which is simple, efficient and safe.

It is also an object of this invention to effect the recovery of chalcopyrite and pyrite from a complex magnetite ore by a method which requires a minimum amount of collectors and depressants and does not require the use of a toxic cyanide compound.

Broadly the improved method of recovering chalcopyrite and pyrite from complex ores includes subjecting a slurry of the finely ground raw ore to a bulk flotation step in which all the chalcopyrite and pyrite is caused to float out of the slurry by the addition of a collector agent thereto, thereafter subjecting the slurry to a conditioning

step to render the collector agent ineffective as a collector for pyrite and thereafter separating the chalcopyrite from the pyrite by selective flotation of the chalcopyrite.

In a more detailed description of the invention a complex magnetite ore analyzing less than 50% in iron content and containing chalcopyrite and pyrite is subjected to a wet grinding operation in the usual rod and ball mills to obtain a slurry of finely ground particles of ore. The bulk ore is screened to remove the large pieces of rock and is then classified in conventional classifiers. The overflow from the classifiers which contain the finer particles in the slurry is passed to cyclones for further separation of the relatively coarser particles from the relatively finer particles. The large particles or sand in the underflow from the classifiers are passed to conventional rod mills for regrinding.

The overflow from the cyclones, usually referred to as slimes, is substantially all magnetite and gangue material and is passed directly to magnetic separators for separation of the magnetite particle from the gangue material. The upgraded magnitite concentrate which may be as high as 66% in iron content may be passed to a plant to be pelletized.

The underflow from the cyclones contains a portion of the magnetite and gangue material and some chalcopyrite and pyrite. The said underflow is combined with the reground sand product from the rod mills. The mixture is further ground in conventional ball mills so that substantially all the particles in the slurry will pass a 100 mesh Tyler sieve.

During grinding of the slurry in the ball mills a collector agent, for example sodium sec-butyl xanthate, is added to the slurry. The treated slurry which may contain about 20% to about 35% solids is then passed to a series of air cell units for bulk flotation of all the chalcopyrite and pyrite. A frother, for example MIBC (methyl isobutyl alcohol), is added to the slurry in the said air cell units. Substantially all the chalcopyrite and pyrite are caused to float in the froth formed on the surface of the slurry and are recovered as the float product while the magnetite and gangue material therin are recovered as the sink product. The chalcopyrite and pyrite are then passed to a bank of froth flotation cleaner cells and thence to a conventional thickener. The slurry is there dewatered and thickened so as to contain up to about 65% solids. The slurry at this point is almost neutral or slightly alkaline and has a pH of between about 6.5 to 8.0. The thickened slurry is diluted to about 50% solids and pumped to Denver conditioners and super agitators. Hydrated lime is there added to the slurry in an amount slightly in excess of that sufficient to raise the alkalinity of the slurry to at least pH 12. Air is caused to be bubbled upwardly through the slurry for a time sufficient to render the xanthate ineffective as a collector agent for the pyrite. The slurry is treated with the lime and air for approximately one hour or more to obtain the aforesaid results. We have found that selective flotation of the chalcopyrite and pyrite is enhanced when the alkalinity of the slurry is above about pH 12 and we prefer to use an alkalinity of pH 12.3.

The slurry is then pumped to a series of flotation cells for selective flotation of the chalcopyrite and pyrite. The lime in the slurry acts as a depressant to the pyrite preventing the pyrite from floating to the surface of the slurry in the flotation cells. The chalcopyrite, on the other hand, floats out and is contained in the froth atop the surface of the slurry and is recovered as the float product. The pyrite is removed as the sink product from the flotation cells and is prepared for shipment by calcining. The chalcopyrite is dewatered by filtration prior to shipment.

The improved bulk flotation of the raw magnetite ore which includes the bulk flotation of substantially all of the chalcopyrite and pyrite in one step from a 30% to 35% solids slurry with no control of the alkalinity of the slurry, followed by selective flotation of the chalcopyrite from an approximately 50% solids slurry having a controlled alkalinity of at least pH 12 has resulted in increased recovery of copper as chalcopyrite and sulfur as pyrite over the old method of recovery as shown in the following table:

COMPARISON OF RAW ORE FLOTATION AND PRIOR PRACTICE

	Prior Practice—selective flotation of magnet tailings		New Practice—bulk flotation of raw ore	
Product	Percent Cu	Percent S	Percent Cu	Percent S
Ore	0.34	1.38	0.31	1.40
Chalcopyrite	26.6	29.4	28.6	32.3
Pyrite	0.40	48.5	0.06	48.4
Magnetite	0.05	0.14	0.018	0.06
Tailings	0.06	0.46	0.05	0.32
Percent copper Recovery	80% of a 26.6% Cu		88.5% of a 28.6% Cu	
Percent pyrite Recovery	72.5% of a 48.5% S		85.0% of a 48.4% S	

Since a greater percentage of chalcopyrite and pyrite is removed from the magnetite ore by our method, the magnetite concentrate is lower in copper and sulfur content than heretofore. Metallurgically, this is a distinct advantage because the pellets formed from the magnetite concentrate contain less copper and less sulfur and consequently deliver less of the same to the blast furnace and to the refining furnaces in the steel mill.

It should be understood, in this specification, that wherever percentages are referred to, such percentages are by weight.

In a specific example of the invention, 178 tons of a complex magnetite ore containing essentially 45.6% iron, 1.00% chalcopyrite and 2.2% pyrite and the remainder gangue material and incidental impurities was screened. The large gangue material or rock was passed to waste while the particles less than ¼" in size were classified in two 30-inch spiral classifiers.

Th overflow from the classifiers contained substantially all the particles smaller than −100 mesh Tyler sieve while the classifier underflow, generally called sand, contained substantially all the particles over −100 mesh Tyler sieve but under ¼" x 100 mesh Tyler sieve. The classifier overflow was passed to six-12 inch diameter cyclones for further separation of the relatively coarser particles and relatively finer particles. The classifier underflow was passed to two-9 feet diameter by 13 feet long rod mills for regrinding.

The overflow from the cyclones contained substantially all particles smaller than 500 mesh Tyler sieve and consisted of less than 5% of the raw ore. The particles analyzed less than 0.10% copper and 0.50% sulfur and contained 8.9 tons of magnetite and gangue material. This was passed to magnetic separators for separation of the magnetite from the gangue. The cyclone underflow contained substantially all particles which passed a 100 mesh Tyler sieve but remained on a 500 mesh Tyler sieve. These particles were mixed with the classifier underflow which had been reground.

The combined particles were ground in two-11½ feet diameter x 14 feet long ball mills to a fineness so that 93% of the particles passed a 100 mesh Tyler sieve as shown below.

Tyler sieve size:	Percent particles passing through sieve
+65	2
+100	5
+200	18
+325	32
−325	68

During regrinding in the ball mills, 0.07 pound of the collector agent, sodium sec-butyl xanthate, per ton of ground material was added to the slurry. After regrinding, the slurry which contained 30% solids was passed to a second series of two-30 inch cyclones where the coarser particles as underflow were separated from the finer particles as overflow. The underflow was recycled to the ball mills for regrinding. The above overflow was next passed to two-No. 30 air cell units for bulk flotation of the chalcopyrite and pyrite. A frothing agent, MIBC (methyl isobutyl alcohol), in an amount of 0.4 pound per ton of ground material was added to the slurry in the air cell units. The aforementioned collector, sodium sec-butyl xanthate, coated the chalcopyrite and pyrite thereby enabling them to float atop the slurry and be recovered as the float product from the air cell units. The magnetite and gangue material did not float and were removed from the air cell units as the sink product. This was passed to magnetic separators where the magnetite, amounting to about 119 tons, was separated from the gangue. Approximately 90% of the chalcopyrite and pyrite in the charge ore was recovered as the float product.

The chalcopyrite and pyrite float product was passed to froth flotation cleaner cells for removal of any incidental gangue material. The slurry formed from the froth contained 18% solids made up of 1.97 tons of chalcopyrite and 4.02 tons of pyrite. The slurry was then passed to a 20 feet diameter thickener to dewater the slurry and produce a slurry containing 60% solids. The slurry, which had a pH of 7.0, was diluted to 50% solids and was subjected to a conditioning treatment in a series of 6 feet diameter x 7 feet high Denver conditioners containing super agitators. Sufficient lime, ten pounds of hydrated lime per ton of sulfides, was added to the slurry to obtain a pH of 12.3. Air was caused to bubble upwardly through the lime treated slurry. The slurry was subjected to the action of the lime and air for one hour and 48 minutes during which time the xanthate was rendered ineffective as a collector agent for the pyrite. The 50% solids conditioned slurry was piped to a series of eight-No. 24 Denver flotation cells. The slurry was diluted in the cells to a 25% solids. The excess lime acted as a depressant for the pyrite preventing it from floating in the froth, whereas the chalcopyrite floated in the froth. The sink product of the slurry contained 90% of the pyrite while the float product contained 83% chalcopyrite. About 89% of the pyrite in the original raw ore was recovered as 48.4% S grade, while 91.6% of the chalcopyrite in the original raw ore was recovered as 28.6% grade copper.

While the foregoing description and examples have been directed to the treatment of a complex magnetite ore, the invention is equally effective in the treatment of other ores containing pyrite and chalcopyrite.

We claim:

1. In a method for recovering chalcopyrite and pyrite from a complex magnetite ore which has been crushed, ground, screened and classified to form a slurry having an alkalinity of from about pH 6.5 to about pH 8.0, and including the steps of adding a xanthate collector agent thereto and subjecting said slurry to bulk froth flotation to produce a first float product containing substantially all the chalcopyrite and pyrite and a first sink product containing substantially all the magnetite and subjecting the first sink product to a magnetic separation to recover substantially all the magnetite therein, the improvement comprising:

(a) treating the first float product with lime to increase the alkalinity thereof,

(b) aerating the treated first float product to render the collector ineffective for pyrite, and

(c) subjecting the treated first float product to a selective froth-flotation to recover chalcopyrite as a

second float product and pyrite as a second sink product.

2. The method of claim 1 in which the amount of lime used in step (a) is sufficient to produce a pH not less than 12 in the first float product.

3. The method of claim 1 in which the time of step (b) is not less than one hour.

4. The method of claim 1 in which the amount of lime used in step (a) is slightly in excess of the amount sufficient to produce a pH of not less than 12 in the first float product.

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U.S. Cl. X.R.

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