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3,380,666

PROCESS FOR OBTAINING POTASH VALUES
FROM POTASH ORE

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ABSTRACT OF THE DISCLOSURE

Potash is obtained from potash ore by a beneficiating procedure in which coarser and finer fractions are reagentized separately and the reagentized finer fraction is subjected to froth flotation to give a concentrate which is combined with the reagentized coarse. The combination is then subjected to another froth flotation procedure to give a highly pure product without "middlings."

This invention relates to a process for obtaining desired mineral values from ore by froth flotation and especially to obtaining potash from potash ore.

Froth flotation of potash from potash ores such as sylvinite, which is predominantly potassium chloride and sodium chloride, is generally accomplished by crushing the ore to the point of KCl-NaCl liberation, treating the ore to remove insoluble clays, reagentizing the treated ore with flotation reagents which have an affinity for potash, and finally separating the KCl from the NaCl in a conventional rougher-cleaner flotation circuit.

The coarse potash particles, generally in the +20 Tyler mesh size range, are far more difficult to float than are the finer particles. Therefore, in order to obtain an acceptable process recovery, it has been found to be necessary to add excessive amounts of flotation reagents to the ore resulting in waste and uneconomical practice. Various methods of overcoming this have been suggested, the most successful of which is separation of the coarse and fine fractions of the ore and addition of all the reagent to the coarse ore fraction. The separate fractions are then recombined and introduced to a "rougher" flotation cell where the KCl is floated and the salt is discarded to waste. The flotation product (concentrate) from the rougher circuit is then refloated in a "cleaner" flotation circuit designed to produce a concentrate of an acceptable grade and an underflow stream called "middlings" which contains too much KCl value to be discarded as waste from the rougher flotation system. The production of the middlings stream in the cleaner flotation introduces metallurgical problems for recovery of the KCl values in this fraction. It is the practice to screen the middlings and to add the coarse or oversize directly to the concentrate. The fines are then returned to the rougher flotation circuit.

The present invention provides a much simpler process for recovering potash from potash ore. According to this invention, the ore is comminuted to provide a coarser and a finer fraction which are sized and separated, such as by screening or classifying, and both fractions are separately reagentized with the flotation collector reagents. The finer fraction is submitted to a rougher flotation circuit to give a potash concentrate mixed with the collector reagent as a float product which is then combined with the coarse fraction, and the combined stream is then introduced to another rougher flotation circuit. The collector reagent in the finer flotation fraction is thus added to the reagent on the coarse fraction and, in effect, extends the beneficial effects of the reagent on the coarse fraction. In addition, the flotation action of the finer potash helps flotation of the coarser particles and results in a potash concentrate which does not require the conventional cleaner flotation

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step. Also of importance is the fact that a middlings fraction, which is normally associated with a cleaner flotation step, is therefore not formed and this eliminates the requirement of middlings fraction treatment steps, such as screening and recycling.

Thus, the present invention provides an improved process which is much simpler and, as a result, more economical than conventional procedures heretofore used.

In preparing the ore for the flotation procedure, it is comminuted, such as by use of conventional ore-crushing equipment, to a particle size preferably in the range of about -8 to -10 mesh. The comminuted ore is then pulped by slurrying it in a saturated brine composed of the soluble constituents of the ore. If the ore contains clay contaminants, it is preferred to deslime using conventional procedures such as by screening the slurry through, for example, a 100- or 200-mesh screen to remove the clay and fines. The deslimed feed is then split into coarser and finer fractions such as by screening through a screen or classifier. The mesh size at which the splitting takes place is not critical, it only being important that the ore is split into two fractions, one of which is coarser than the other.

After splitting, the fractions are reagentized separately and submitted to froth flotation circuits as described above. Conventional flotation temperatures, such as in the range of from about 15° to about 35° C., are employed.

If desired, a "blinding" agent such as starch can be added to the finer portion prior to flotation to act as a depressant for the control of slimes.

The coarser and finer fractions are reagentized separately with the conventional amine flotation reagents well known to those skilled in the art. The saturated or unsaturated long-chain aliphatic amines, generally in the form of their water-soluble addition salts, can be used as collector reagents. Typical amines suitable as reagents are those having at least eight carbon atoms per molecule, such as n-octadecyl amine, n-octadecenyl amine, n-nonylamine, n-decylamine, preferably as the water-soluble salts thereof such as the acetates, and the like. The amine collector reagents may be employed either individually or as a mixture containing more than one amine. The amine collector reagent may be employed in the form of an aqueous solution, emulsion, or solution in an organic solvent. In the flotation of the finer portion, I prefer to employ from about 0.05 to about 0.5 pound of the amine reagent per ton of ore. The coarser fraction is preferably reagentized with from about 0.2 to about 0.8 pound of the amine reagent per ton of ore. Of course, greater or lesser amounts of reagents can be used if desired, but the above amounts are presently preferred.

Also, to aid in the action of the amine collector reagents, it is preferred to employ auxiliary reagents such as alkylnaphthalenes, the chloro and hydrogenated derivatives thereof, alkylmercaptans, and dialkylsulfides and their aryl derivatives, dialkyl disulfides, dialkyl polysulfides, indane, alkylindanes and polycyclic aromatic compounds, or mixtures thereof, such as described in U.S. Patents 3,059,774 and 3,149,788, as well as co-pending U.S. patent application Ser. No. 397,036 filed Sept. 16, 1964, now Patent No. 3,341,135, by Martin Wilson and assigned to the assignee of the present application.

The following examples are presented for illustration, but the invention is not to be considered limited to the specific examples given.

EXAMPLE I

Sylvinite ore (500 grams) from the Carlsbad, New Mexico potash deposits, having a K₂O analysis of 22.2%, was crushed to -8 mesh and slurried in a saturated brine composed of the soluble constituents of the ore. The

slurry was deslimed twice on a 100-mesh screen to remove the fines and insoluble matter, and the deslimed ore then wet screened and divided into two fractions; one fraction containing the +20 Tyler mesh particles, and the other fraction containing the -20 Tyler mesh particles. The -20 mesh fraction was reagentized with 0.4 pound of starch per ton of ore and 0.13 pound of alkylamine (consisting of tallow amine acetates, of which 75% was saturated) per ton of ore. Pine oil (consisting of mixed terpenes) in the amount of 0.05 pound of oil per ton of ore was added to the fraction, which was then floated in a Fag flotation cell. The floated fine concentrate was collected for combination with the +20 mesh fraction. The +20 mesh fraction was reagentized with 0.32 pound of alkylamine (consisting of the same tallow amine acetates) per ton of ore and with 0.5 pound of Provalent 4A (an aromatic oil containing about 40% polycyclic compounds having at least 3 rings in the molecule, 7% alkylbenzenes and 34% alkylnaphthalenes and alkylindanes) per ton of ore. The reagentized coarse fraction was then combined with the floated fine concentrate and the resultant combination submitted to a Fag flotation cell. The resultant potash concentrate was recovered and analyzed and the tailings were discarded as waste salt. The concentrate analyzed 60.63% K_2O (95.8% KCl) and represented 95.88% K_2O recovery. The tailings contained 1.41% K_2O and represented 4.22% K_2O recovery.

EXAMPLE II

The procedure of Example I was repeated. The potash concentrate obtained analyzed 60.49% K_2O (95.6% KCl) and represented a 96.86% recovery of K_2O . The tailings analyzed 1.08% K_2O and represented a 3.14% K_2O recovery.

EXAMPLE III

The procedure of Example I was repeated. The potash concentrate obtained from the flotation procedure analyzed 60.35% K_2O (95.4% KCl) and represented a recovery of 97.46% K_2O . The tailings analyzed 0.88% K_2O and represented a 2.54% K_2O recovery.

Thus, as can be seen from the above examples, the novel process of this invention provides excellent recovery of potash from Sylvinitic ore, giving a high-grade product. The process eliminates the undesirable "middlings" product and consequently the required middlings

screening and additional processing steps. Also, a further advantage is the fact that the coarse and fine tailings are collected separately, which eliminates a possible later waste salt split for retreatment of the coarse fraction such as by leaching or regrinding.

Various changes and modifications of the invention can be made, and to the extent that such changes incorporate the spirit of this invention they are to be included within the scope of the appended claims.

I claim:

1. In the froth flotation of sylvinitic ore to recover potash values, the steps which include comminuting said sylvinitic ore, to about -8 mesh, pulping said comminuted sylvinitic ore with brine and desliming said pulp, sizing and separating said comminuted and deslimed sylvinitic ore to provide a coarser fraction and a finer fraction, separately reagentizing said finer fraction with amine collector reagent, and submitting the reagentized finer fraction to froth flotation to provide a finer potash concentrate mixed with amine collector reagent, separately reagentizing said coarser fraction with amine collector reagent, combining the reagentized coarser fraction and said finer potash concentrate mixed with amine collector reagent and submitting the resultant combination to froth flotation to provide a clean potash concentrate without formation of a middling fraction.

2. The process according to claim 1 in which said finer fraction is reagentized with from about 0.05 to about 0.5 pound of amine collector reagent per ton of ore and said coarser fraction is reagentized with from about 0.2 to about 0.8 pound of amine collector reagent per ton of ore.

3. The process according to claim 1 in which said finer fraction is about -20 mesh and said coarser fraction is about +20 mesh.

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