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Optimization of an Integrated Flowsheet for Barite Processing

R. CICCU, L. CURRELI, S. GIULIANI, P.P. MANCA and G. MASSACCI

Department of Mining and Minerals Engineering, University of Cagliari, Italy

This paper examine the optimization of a beneficiation plant for recovering marketable barites from crude ores of different characteristics. The plant, part of a mining complex in Sardinia, consists of a jig section integrated with a flotation line. The problem has been studied using a suitable model of the process, based upon experimental and field data pertaining to both the characteristics of the feed material and the performance of machinery. Data have been automatically processed with the aid of a computer to find the optimum setting of each section, in order to maximize profit for each kind of ore separately fed to the plant. Moreover, the advantages of blending the various concentrates — not all of them individually marketable — are also demonstrated.

The overall economic result can be improved provided suitable proportions of each kind of ore are fed to the plant. Such information is fundamental for the long-term planning of exploitation of available reserves, and provides guidelines for further prospecting.

The paper illustrates the washability characteristics of the ores and describes in detail the model adopted. The results of data processing are then presented in the form of computer graphs and discussed. Finally, conclusions are drawn regarding the advantages of resorting to Operations Research as an aid to management.

Introduction

At the Barega mine, in Sardinia, ore reserves are contained in a number of orebodies of varying characteristics. They are mined either opencast or underground, and the resulting run-of-mine can be distinguished as follows, according to $BaSO_4$ grade and intergrowth features:[1]

(a) a crude ore easily washable with gravity methods yielding relatively good recoveries (ore A, from Litopone stope);

(b) a crude ore of lower grade containing finely disseminated quartz which yields a poor quality concentrate at the expense of considerable losses in the coarse waste (ore B, from Gianni stope).

The plant consists of three main sections:

(a) a three-stage crushing station where the R.O.M. is reduced from a top size of about 600 mm down to - 20 mm;

(b) a two-stage jigging section with interstage grinding whereby a gravity concentrate is produced;

(c) a flotation line for the recovery of barite values still contained in the fine fractions discarded from the preceding section.

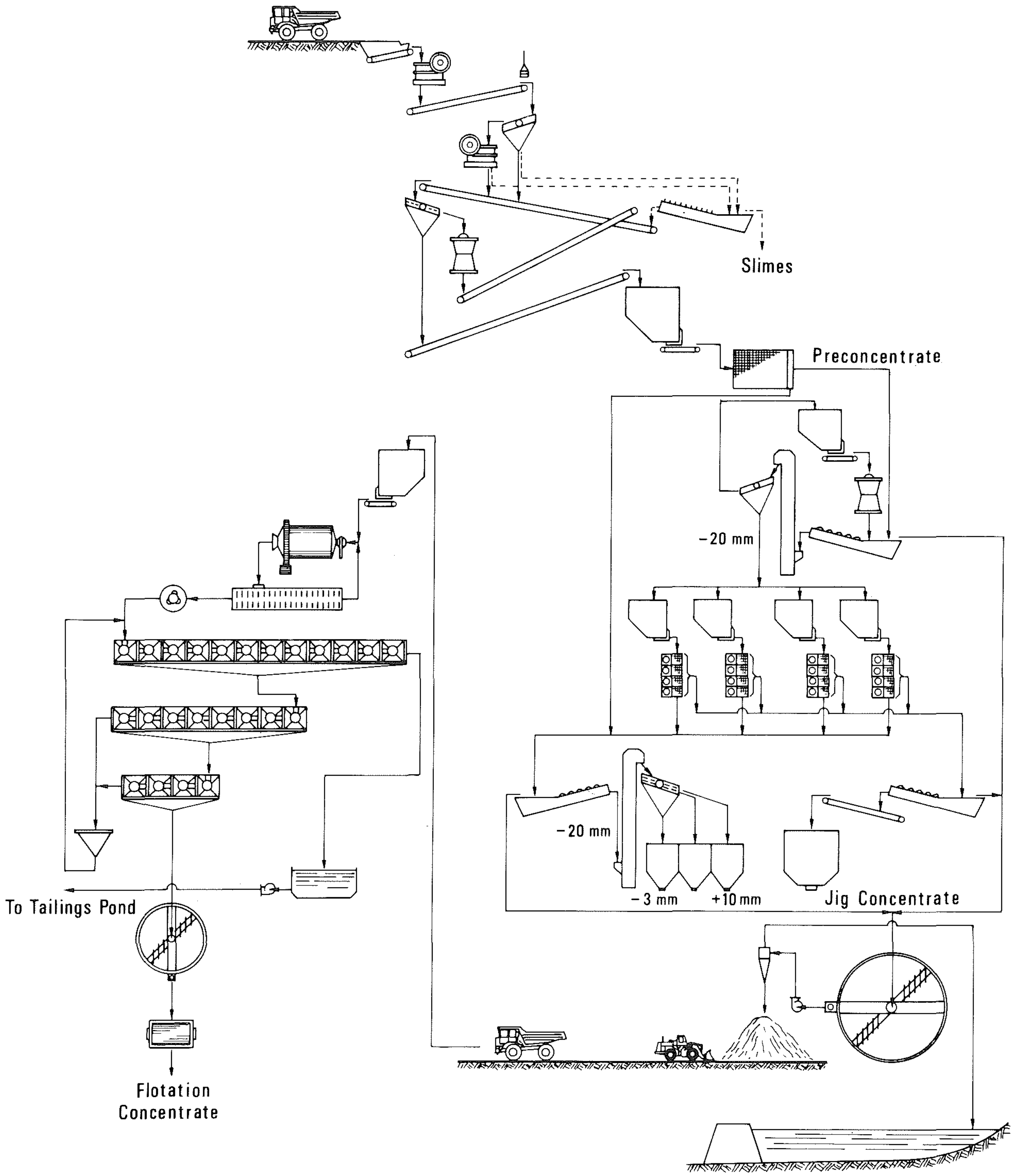

FIGURE 1. Flowsheet of the Barega plant

The flowsheet of the plant is sketched in Figure 1.

Each kind of ore is stocked in different stockpiles and can be fed separately to the plant. Crushing is done with the same machine setting, irrespective of the ore being processed.

The reject of the screen-controlled cone crusher is collected in a 1200 tonnes bin at the head of the upgrading plant.

Gravity treatment is carried out in two stages. In the first stage the ore is preconcentrated in two parallel lines, each consisting of a vibrating bed jig with a capacity of 60 t/h. The coarse waste is separated, screened and sold as an aggregate for concrete. The finest size class (-3 mm) may be suitable for flotation, provided it contains enough recoverable barite.

The preconcentrate and the fines filtered through the jig bed are dewatered in a spiral classifier: the overflow is sent to a Dorr thickener whereas the coarse fraction is screened at 8 mm, which is the liberation size for the final gravity concentration. A short-head cone crusher reduces the screen reject to below that size. The crushed preconcentrate is split and stored in four bins, each feeding a stationary-bed jig with a capacity of 7 t/h for final cleaning. Both concentrate and middlings are dewatered; overflow fines are thickened and sent to flotation together with preconcentration fines while middlings follow the same route as the preconcentration waste.

Settled pulp from the Dorr thickener is fed to the flotation line. The material is first cycloned; this operation generally produces a 10 to 15 $BaSO_4$ points increase in the underflow grade with tolerable barite losses. This underflow is ground to - 0.5 mm in a ball mill in closed circuit with a rake classifier; after conditioning with Na-Cethylsulfate, the reject is processed via flotation with two cleaning stages obtaining a commercial filter cake assaying 94 - 95% $BaSO_4$ suitable for barium chemicals.

The optimization problem

The barite market is today undergoing a worldwide slump. In fact, owing to the recent fall in oil prices, the demand for drilling muds applications, by far the largest outlet for barite concentrates, has suddenly diminished.[2]

Consequently, the market structure has undergone a major change compared to the previous period. This chiefly concerns the more stringent quality requirements for the different utilizations: oil service companies, in addition to OCMA (Oil Companies Materials Association) standards, tend now to refuse blends with products containing flotation reagents, whereas manufacturers of barium salts accept only high-grade concentrates with very low pollutants. Moreover, over the last few years market prices have fallen considerably in real terms.

Under these circumstances, the production schedule had to be adjusted; presently, the plant output consists of the following products:

(a) a high-grade, low silica gravity concentrate for barium chemicals;

(b) a 4.20 S.G. jig concentrate suitable as weighing agent for drilling muds (after grinding) or heavy concrete;

(c) a flotation filter cake for barium chemicals;

(d) classified aggregates for common concretes or road construction

Therefore the main problem is how to optimize the production schedule in order to achieve the maximum profit while satisfying market demand.

As regards the ore, the basic data to be included in the process model are the washability characteristics and the proportion of each kind of ore being beneficiated. This proportion must be consistent with the level

TABLE 1. Washability characteristics of crude ore A, by size

SIZE CLASS	DENSITY													
	2.56	2.60	2.65	2.71	2.78	2.91	3.00	3.10	3.20	3.70	4.20	4.30	4.40	4.45
	MASS													
-20.000 +7.925	.0289	.0520	.0748	.1178	.0632	.0245	.0146	.0087	.0233	.0291	.0146	.0437	.0874	
- 7.925 +4.699	.0071	.0089	.0108	.0226	.0059	.0004	.0002	.0005	.0010	.0017	.0015	.0103	.0340	
- 4.699 +2.362	.0067	.0077	.0137	.0168	.0061	.0019	.0014	.0001	.0035	.0038	.0037	.0119	.0106	
- 2.362 +1.168	.0062	.0034	.0087	.0120	.0038	.0020	.0010	.0007	.0007	.0010	.0012	.0065	.0206	
- 1.168	.1567													
	GRADE													
-20.000 +7.925	.0239	.0267	.0259	.0208	.1053	.1314	.1866	.2447	.4190	.7095	.8838	.9419	.9855	
- 7.925 +4.699	.0346	.0282	.0313	.0296	.1318	.1572	.2107	.2671	.4362	.7181	.8872	.9436	.9859	
- 4.699 +2.362	.0440	.0243	.0331	.0304	.1076	.1337	.1887	.2467	.4205	.7103	.8841	.9421	.9855	
- 2.362 +1.168	.0251	.0226	.0331	.0246	.1445	.1695	.2223	.2778	.4445	.7222	.8889	.9444	.9861	
- 1.168	.4982													

ORE GRADE = .3810
FEED GRADE (FINES EXCLUDED) = .3593

TABLE 2. Washability characteristics of crude ore B, by size

SIZE CLASS	DENSITY													
	2.56	2.60	2.65	2.71	2.78	2.91	3.00	3.10	3.20	3.70	4.00	4.20	4.30	4.40
	MASS													
-20.000 +7.925	.0293	.0454	.1134	.0792	.1765	.0412	.0203	.0102	.0190	.0319	.0305	.0475	.0339	
- 7.925 +4.699	.0065	.0074	.0223	.0185	.0212	.0020	.0009	.0011	.0020	.0034	.0018	.0053	.0122	
- 4.699 +2.362	.0050	.0085	.0150	.0101	.0151	.0039	.0021	.0005	.0010	.0007	.0021	.0066	.0058	
- 2.362 +1.168	.0062	.0043	.0071	.0053	.0071	.0024	.0016	.0007	.0005	.0005	.0007	.0032	.0064	
- 1.168	.0894													
	GRADE													
-20.000 +7.925	.0195	.0207	.0153	.0369	.0473	.0751	.1339	.1958	.3814	.6288	.7835	.8763	.9381	
- 7.925 +4.699	.0227	.0198	.0141	.0279	.0336	.0618	.1215	.1842	.3725	.6235	.7804	.8745	.9372	
- 4.699 +2.362	.0273	.0152	.0129	.0277	.0448	.0727	.1316	.1937	.3797	.6278	.7829	.8759	.9380	
- 2.362 +1.168	.0196	.0137	.0100	.0261	.0435	.0714	.1305	.1926	.3789	.6273	.7826	.8758	.9379	
- 1.168	.3680													

ORE GRADE = .2293
FEED GRADE (FINES EXCLUDED) = .2157

of known reserves available at the mine. Washability characteristics have been determined by sink-float analysis using heavy liquids or FeSi suspensions in TBE (tetrabromoethane) for the higher densities.[3]

The results of the experimental investigation are summarized in Tables 1 and 2 and by the graphs of Figure 2.

As regards the features of the plant, the basic information on machinery performance has been collected after carefully sampling the material passing through the various sections of the industrial operations, and in particular the feed and reject of the jigs. Accordingly, the following data have been assumed:

(a) imperfection of primary jigs: 0.15;

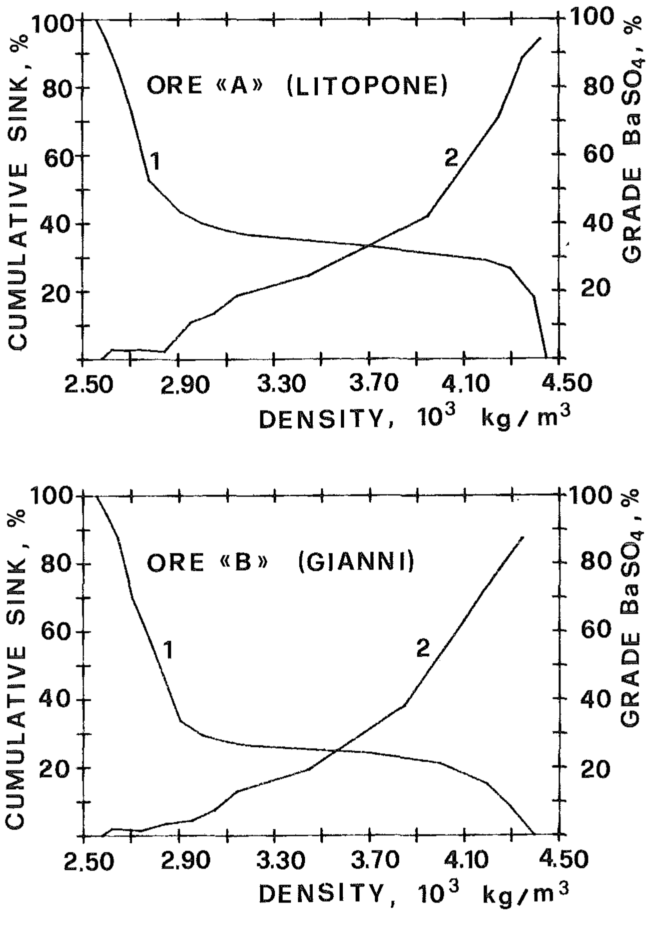

FIGURE 2. Washability curves for ores A (Litopone stope) and B (Gianni stope), fines excluded

(b) imperfection of final cleaning jigs: 0.10;

(c) flotation yield: varying linearly with the feed grade from 0.16 at 18% $BaSO_4$ up to 0.36 at 36% $BaSO_4$.[4]

Running costs have been estimated on the basis of the records available at the mine. For each operation the unit variable costs have been determined with reference to each tonne of ore treated. In fact only the variable costs are meaningful for optimum machine setting.

The main components of variable costs are energy, spare or wear parts, water supply, maintenance and flotation reagents, whereas manpower is to be considered as independent of throughput on account of the relatively narrow range of variation assumed for the setting parameters.

Development of the model

As a first approach to the problem of optimization of the whole plant, the sub-system 'Jigging + Flotation' is taken into consideration here.

In the computer program, the relevant information for process simulation concerns the distribution, for each size class, of the unit mass among the different density fractions of the full range (2.65 - 4.45 kg $\cdot$ dm^{-3}). These data are separately stored in matrix form for each kind of ore.

The other input data are loaded from the keyboard with interactive operations. They consist of the variables and constraints of economic nature such as unit costs, market prices and demand, as well as of the main technical features of the preparation plant (imperfection parameters for the preconcentration and final cleaning stages, plant capacity, average daily throughput).

The main setting parameters of the process are the values of the separation densities d_1 and d_2 at which the preconcentration and final upgrading jigs are adjusted.

With reference to one tonne of crude ore fed to the plant, the model developed allows to obtain, for each combination of d_1 and d_2 within the assigned fields, the following results:

(a) the ore mass circulating through the various plant sections and the corresponding operating costs, for that part proportional to the amount of ore treated (variable cost). In fact, as already mentioned, only this part of the processing cost, given the average daily feed rate, is relevant for optimum regulation of the plant;

(b) the total variable cost of the whole process per tonne of ore;

(c) the yield, recovery and grade for the various final products; the characteristics of the materials entering or leaving the intermediate stages of the process are also simulated;

(d) the overall production and market value of the commercial products which can be sold at maximum profit, their optimum mix and the total sales income;

(e) the resulting margin of contribution of the processing plant.

The peak value of this latter parameter corresponds to the most advantageous adjustment, i.e. the optimum combination of d_1 and d_2.

The computer program consists of a main program and five subroutines written in a generalized form, suitable for a variety of similar cases.

The characteristics of the float and sink products resulting from the first jigging stage (preconcentration) are given in matrix form by a suitable subroutine; for each size class, the fractions of the unit mass relating to each density range are obtained by assigning to the corresponding mass of the feed the collective probabilities α and $(1 - \alpha)$ to belong to either alternative product of the operation, respectively. The value of these probabilities is calculated according to the log-normal distribution law as a function of the separation density (d-1) relative to the density of water, given the imperfection parameter I_1. The log-normal distribution has been approximated by a 5th degree polynomial.

Owing to the non-linearity of the curve, a sufficient degree of accuracy in the calculation is achieved by reducing the width of the elementary intervals (0.01 $kg \cdot dm^{-3}$) by which the variable density is increased from that of the lightest gangue (2.60 $kg \cdot dm^{-3}$) up to that of pure barite (4.45 $kg \cdot dm^{-3}$).

The mass of the feed assigned to each unit interval is calculated by linear interpolation of the available experimental data. The screening of the preconcentrate which precedes the interstage grinding is regulated by the mesh size. However, in the case at hand the model does not imply any decision since both ores are sufficiently liberated below about 8 mm. Actually, an imperfection factor affects the dispersion of the Gauss probability distribution law which describes screen performance; but for the sake of simplicity and on account of the fact that screening is wet, a perfect size classification was assumed, with negligible loss of accuracy.

Prior to the final concentration stage, oversize is coarsely ground in a short-head cone crusher, with the discharge aperture set at around 8 mm. This operation produces a proportion of fines below 1.168 mm, weighing on average one-tenth of the feed to the crusher; in practice this figure varies slightly with the preconcentrate since the waste rock (Cambrian limestone) is tougher than barite; however, the simplification adopted can be considered to have a negligible effect.

It is also assumed that no substantial concentration of barite in the fines takes place. This hypothesis is corroborated by some 'spot' laboratory tests using a crushing machine of performance comparable to that installed in the plant.

The new fines produced are combined with the fines separated during the preconcentration stage and sent to the grinding mill for flotation.

The final jigging operation is simulated working on the washability characteristics attributed to the ground preconcentrate after removal of fines, combined with those of the screen undersize, given by the simulation of the preconcentration stage.

The washability curve of the ground preconcentrate is constructed by means of a mathematical algorithm; the results obtained agree fairly well with the corresponding operation data.[3]

To check the above assumption, spot samples have been taken from the belt feeder and the discharge chutes of the preconcentration jig. The -20 +8 mm and the -8 +1.168 mm size fractions of the feed sample have been analyzed with heavy liquids up to 4.20 $kg \cdot dm^{-3}$; the washability curve of the ground preconcentrate calculated with the model is reported in Figure 3 together with the experimental points of the direct analysis of the actual preconcentrate sample, ground to -8 mm. As it can be observed, the agreement is fairly satisfactory.

It has been assumed that the washability characteristics of the ground product are similar to those of the same size class -8 +1.2 mm of the ore fed to the plant, experimentally studied by heavy liquid analysis.[3,5]

Accordingly, the ground product, after

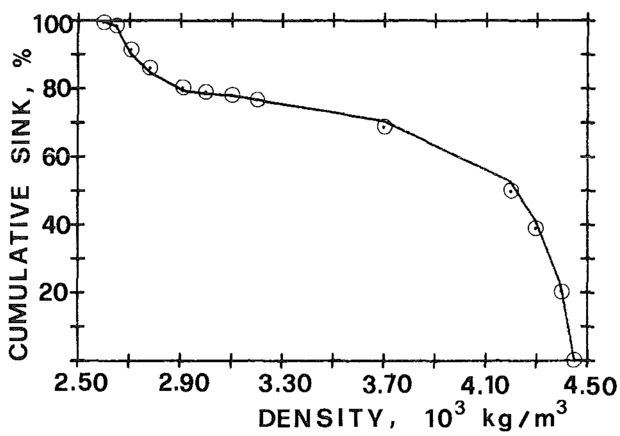

FIGURE 3. Comparison of calculated washability curve of ground preconcentrate fed to final cleaning jigs (solid line) with the corresponding experimental data of heavy liquid analysis (circled points) for ore A.

screening off the -1.168 mm fines, will contain, in the intermediate density range of the washability curve, the same proportion of intergrown middlings as the corresponding fraction of the feed, i.e. the two curves have the same gradient.

This assumption is justified in the case being examined, since crude ore A consists of two prevailing components (barite and limestone, with minor associations of other gangue minerals) sufficiently liberated at sizes below 8 mm, whereas in the case of ore B the polluting gangue (quartz) is evenly intergrown with the barite matrix in the form of fine associations or disseminations. However, the two products have different $BaSO_4$ grades, generally higher in the preconcentrate. Therefore suitable adjustments must be introduced as regards the mass of density fractions outside the 'middlings' range, i.e. lighter than 3.20 $kg \cdot dm^{-3}$ or heavier than 3.80 $kg \cdot dm^{-3}$. In fact it has been observed that within this range the washability curves of the Barega ore are almost flat, with minimum constant slope.

According to the model, the amount of mass to be subtracted from the lighter and added to the heavier fractions is kept proportional to the mass contained in the corresponding density fraction of the -8 +1.2 mm size class of the feed.

The following formulas have been used:

$$m_z(i) = m_y(i) \cdot \left(1 - \frac{q}{L_y}\right) \qquad \text{for } 1 \leq i \leq n_1$$

$$m_z(i) = m_y(i) \qquad \text{for } n_1 \leq i \leq n_2$$

$$m_z(i) = m_y(i) \cdot \left(1 + \frac{q}{H_y}\right) \qquad \text{for } n_2 \leq i \leq n_3$$

$$L_y = \sum_{i=1}^{n_1-1} m_y(i); \qquad H_y = \sum_{i=n_2+1}^{n_3} m_y(i);$$

where $m_z(i)$ and $m_y(i)$ are the shares of the unit mass contained in the i-th density fractions of the ground preconcentrate and of the 'reference' feed, respectively. The n_1-th and n_2-th density fractions correspond to the extremes of the 'middlings' range whereas the n_3-th is bound by the density of the heavier mineral component. In our case the n_1-th fraction is that just above 3.20, the n_2-th is that just below 3.80 and the n_3-th is that just below 4.45 $kg \cdot dm^{-3}$ (volume mass of barite).

The amount of mass to be calculated from the above formulas depends on the value of the parameter q which is determined under the constraint that the grade t_z of the product leaving the crusher is exactly the same as the grade t_x of the screen undersize entering it.

Therefore:

$$q = \frac{\sum_{i=1}^{n_3} m_x(i) - m_y(i) \cdot t_x(i)}{\frac{1}{H_y} \sum_{i=n_2+1}^{n_3} m_y(i) t_x(i) - \frac{1}{L_y} \sum_{i=1}^{n_1-1} m_y(i) t_x(i)}$$

where $m_x(i)$ and $t_x(i)$ are the mass and the grade of the i-th density fraction of the material to be ground, respectively.

In practice, satisfactory results are easily met if the 'middlings' interval is reasonably limited.

The characteristics of the ground product are given by a subroutine in vectorial form since the subdivision of the masses into size classes is no longer necessary nor useful for the subsequent development of the model.

Finally, the last subroutine enables the characteristics of the material feeding the jigs to be calculated by appropriately combining the undersize with the ground product obtained from the comminution of oversize.

The final cleaning stage is simulated using procedures similar to those followed for the preconcentration simulation.

The product of this operation is a barite concentrate and middlings to be sent to flotation together with the primary fines and the fines obtained after interstage comminution. Moreover, the model makes allowance for sending to flotation the fraction -3.0 +1.168 mm screened from the preconcentration reject. For this purpose the computer program verifies whether the recoverable values still contained therein are at least equal to the marginal costs of flotation.

The above-described procedures are repeated for each pair of densities, d_1 and d_2, within the programmed fields of variation.

The data base generated is suitable for automatic plotting and for further processing.

Blending operations are simulated through the last subroutine: this allows identification of the set of admissible solutions for variables t_A, t_B and λ (mutual proportion of ores A and B) at which marketable products can be obtained.

Discussion

First of all, the treatment of each kind of ore individually fed to the plant has been simulated. The setting parameters for the gravity section are the separation densities d_1 and d_2 of preconcentration and final cleaning, respectively.

In this case the objective to be pursued was that of achieving from each ore the maximum margin of contribution, i.e. the largest difference between the market value of all the saleable products obtainable from one tonne of ore (gravity and flotation concentrates, classified aggregates) and the corresponding unit variable processing cost. The existence of a peak value of the margin of contribution for varying combinations of the separation densities is highlighted by the curves of Figure 4. The upper family of

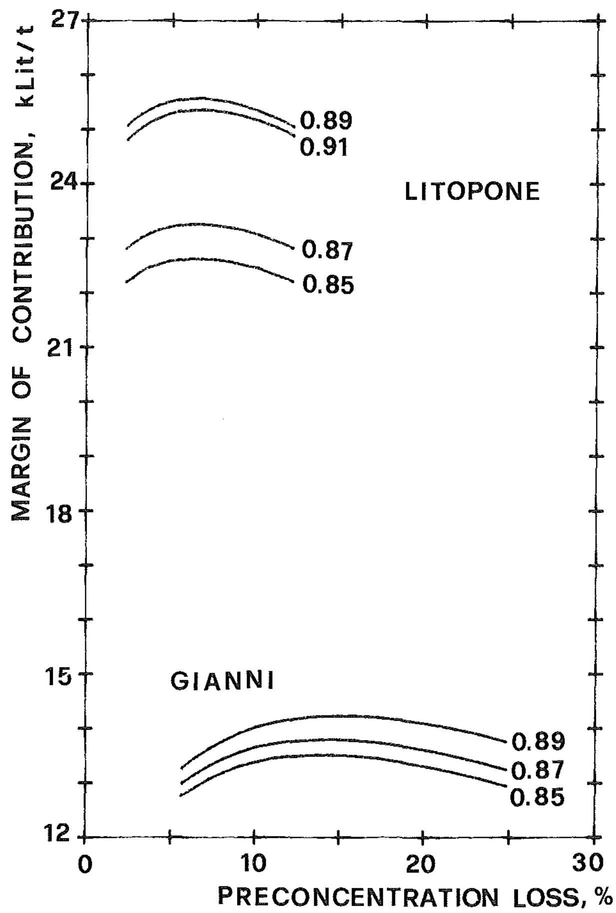

FIGURE 4. Processing margin of contribution (M.O.C.) for ores A (Litopone tope) and B (Gianni stope) at varying level of barite loss in coarse preconcentration waste. The parameter is the BaSO grade of final jog concentrate

curves refers to ore A, which is of higher quality and easily washable, the lower family to ore B, which is leaner and less amenable to treatment.

The M.O.C. (Margin of Contribution) is represented as a function of the percent loss of barite in the coarse preconcentration waste, which is the most significant variable of this operation. The parameter of the curves is the $BaSO_4$ grade of the final gravity concentrate.

The peak M.O.C. for ore A is reached at very low losses, between 5 and 10%. Every percent point lost in preconcentrate A recovery entails greater economic disadvantages than for ore B for which losses as high as 15 - 20% are still tolerable. As can be observed, the optimum M.O.C. is obtained for a concentrate grade of 89%: at increasing values above this level the higher market price paid by the chemical industry is neutralized by the deterioration of the overall yield.

The peak M.O.C. is a function of final jig concentrate grade, as clearly shown by the curves of Figure 5, upper part. The terminal point of each curve represents the highest $BaSO_4$ grade technically achievable with the present plant, given the grinding size: concentration of ore A can be pushed to 96%, whereas that of ore B is limited at 89%. Correspondingly, optimum concentrate yields vary according to the curves of Figure 5, lower part. Gravity yield deteriorates gradually with constant gradient up to 92% for ore A and 86% for ore B, then drops sharply with asymptotic values of 96% and 89%, respectively. Flotation yield increases almost symmetrically.

The most significant conclusion is that optimum jig concentrate grade is 89% for both ores, under the constraint that each is individually sold on the market. In fact, 89% $BaSO_4$ is the lowest acceptable limit for either drilling or chemical use; below this the material is rejected or heavily pen-

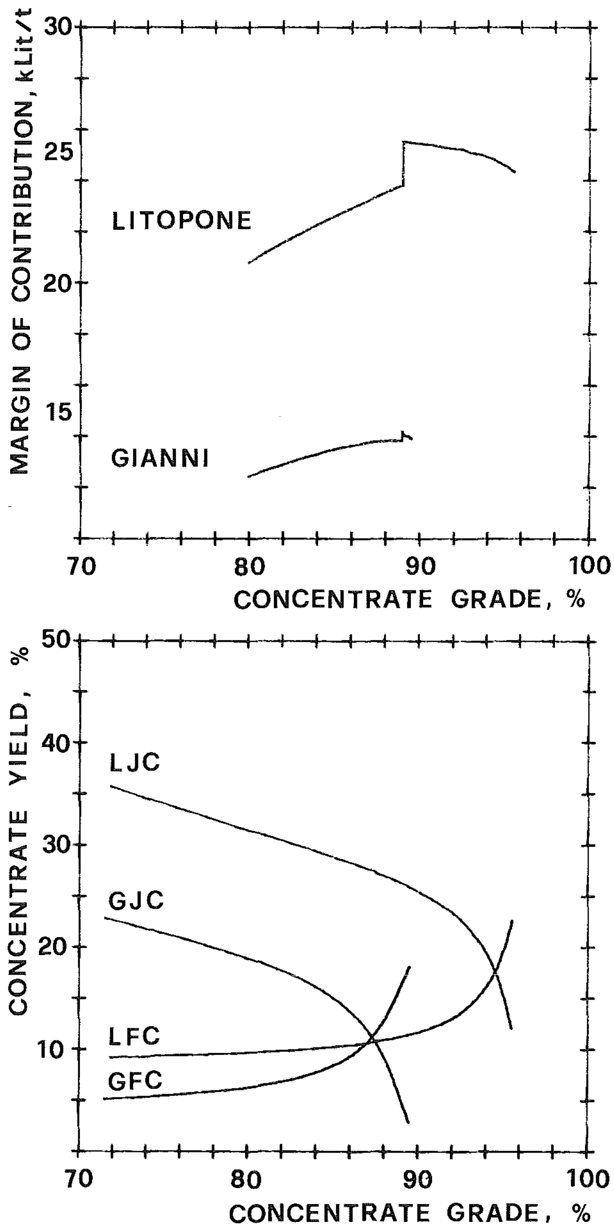

FIGURE 5. *Above*: Processing margin of contribution as a function of jig concentrate grade for ores A (Litopone stope) and B (Gianni stope). *Below*: Jigging and flotation yields of ore A (LJC and LFC, respectively) and ore B (HJC and GFC, respectively) versus jig concentrate grade

alized.

The fact that jigging yield of ore B progressively deteriorates well before reaching the saleability limit suggests that consistent economic advantages may be achieved by resorting to blending.

After this approach to the problem of optimum beneficiation of the whole R.O.M., upgrading of ore A can be pushed further, and that of ore B somewhat diminished, to the extent that a blended concentrate is obtained which meets market specifications.[6] The outcome of computer simulation is shown in Figure 6, where the difference between the M.O.C. after blending and the optimum M.O.C. without blending is plotted against percent proportion of ores A and B fed to the plant (with respect to the total). The parameters of the curves are the grades t_A and t_B of jig concentrates A and B, respectively.

Figure 6 consists of two graphs, one for each blending scheme. The first reflects the case where the jig concentrates are blended in their entirety; the second refers to the case where blending with the whole concentrate B is limited to that part of concentrate A in excess of the chemical industry

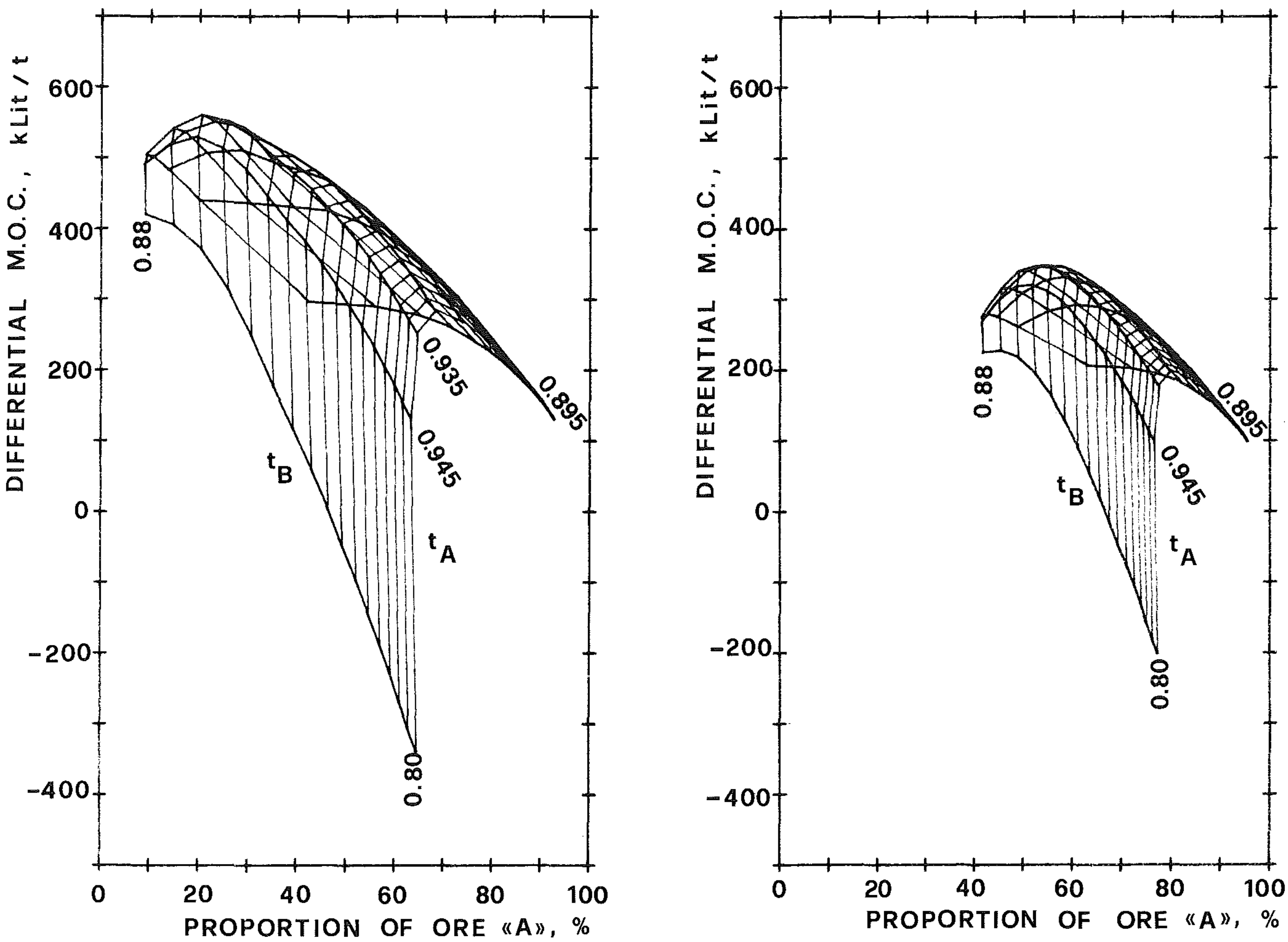

FIGURE 6. Additional economic advantage achievable through blending operations of concentrates A and B of varying quality with respect to the weighed M.O.C. for the single ores at optimum plant setting. Abscissa represents the proportion of ore A in the overall feed. *Left side*: Blending of concentrates A and B in their entirety. *Right side*: Blending of whole concentrate B with that part of concentrate A in excess of chemical industry demand.

demand. Flotation concentrate is not utilized in either case.

It is worthwhile noting that blending is always advantageous, except for the extreme cases when t_A is too high, let us say above 95%, and t_B is inversely too small, around 80% or less. Blending is again economically advantageous even for the highest values of t_A, provided that t_B is increased to a level such that smaller proportions of concentrate A are required for offsetting its poor quality.

Concerning the first scheme, the advantage of blending increases at higher t_B while adapting either the grade or the relative proportion of concentrate A, or both. The absolute peak value of the envelope curve indicates that blending is most convenient when ore A is upgraded to 93,5 and ore B to 87% $BaSO_4$, the proportion of ore A being around 20% of the total. This, of course, does not imply that the plant should be set at these conditions.

Obviously the larger the amount of ore A the higher the profits. The optimum processing M.O.C. for ore A alone is about 25.5 kLit/t whereas that for ore B is slightly above 14.2; the unit costs of mining and haulage are almost the same in both cases.

The proportion of the two ores is dictated by the situation of the reserves and by the mining constraints. Given this proportion, which may vary with time, the production schedule should be set at the corresponding point on the envelope curve. Similar considerations also hold for the blending scheme B, except that presently the curves are displaced towards higher proportions of ore A.

Scheme A is more advantageous, except when ore A prevails by far over ore B; in this case the two envelope curves almost overlap.

Conclusions

The advantages of applying Operations Research and in particular computer simulation for identifying optimum operating conditions of a processing plant are widely recognized.[7] The case discussed in this paper further supports this assumption.

In particular, the likelihood of the model here proposed for multi-stage jigging plants, eventually integrated with flotation, seems reasonably verified, at least for ores of simple composition.

Finally, the advantage of blending high-quality concentrates with poorer ones obtained from less amenable ores in order to make the latter marketable, are also shown. In the case here illustrated an additional profit of the order of 500 Italian Lire per tonne of ore can be achieved by resorting to blending techniques, also taking into account the cost of handling.

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List of symbols

d	Current separation density of the gravity process
d_1	Separation density of preconcentration
d_2	Separation density of final cleaning
$\alpha(d)$	Probability of collection in the heavy product of industrial jigging
i	Current index for the elemental density fractions in the simulation of industrial jigging
$m_x(i)$	Mass of the i-th elemental density fraction of the feed to the inter-stage crusher (preconcentrate undersize)
$m_y(i)$	Mass of the i-th elemental density fraction of the 'reference' feed (ore below the final grinding size, untreated fines excluded)
$m_z(i)$	Mass of the i-th elemental density fraction of inter-stage crusher output
t_x, t_z	Calculated grade of screening undersize of preconcentrate before and after inter-stage grinding (to be kept equal)
L_y	Cumulative mass of the light product of preconcentration up to the lower limit of middlings range
H_y	Cumulative mass of the heavy product of preconcentration down to the upper limit of middlings range
n_1	Number of elemental density fractions below the lower limit of middlings range
n_2	Number of elemental density fractions below the upper limit of middlings range
n_3	Overall number of elemental density fractions
t_A	Concentrate grade from ore A of easier washability
t_B	Concentrate grade from ore B of more difficult washability
λ	Proportion of ore A in the plant feed

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