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PERFORMANCE EVALUATION IN A SMALL SCALE GRAVITY CONCENTRATION PLANT

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In this paper, the results of a performance evaluation study carried out in a chromite concentration plant are presented. The plant is a typical of the similar kind of gravity concentrators which are not equipped with basic control and measuring devices. Therefore evaluation of their performance appears to be a difficult task and requires unique methodology. The study commenced with sampling. A week long sampling surveys were repeated twice. First, the data was mass balanced then conventional parameters of equipment efficiency were determined. After detailed evaluation of the performance data, some modifications at the plant operation, i.e. changing screen aperture from 6 mm to 2 mm, limiting capacity for the plant and the possibility of using two rod mills in parallel, were proposed. Their effects on the performance were investigated by using computer simulation techniques. The results show that the performance and capacity could be improved by applying the proposed changes in the flowsheet.

Key words: gravity concentration, performance evaluation, rod mill, modelling, simulation

INTRODUCTION

Performance evaluation has vital importance in mineral processing plants. It is necessary to determine the existing plant performance to diagnose problems in

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individual equipment or circuit configurations and to investigate the alternatives for improving metallurgy and/or reducing costs.

Since modern plants equipped with instruments and control devices, it is relatively easier to obtain large amount of good quality data from them. Developments in computers and softwares have made it possible to calculate complex mass balances and to carry out simulations for many the modifications in operating conditions and flowsheets. Consequently, this type of modern tools is emerging to have routine use in modern plants.

However, most plants of small to medium capacity are in general poorly instrumented. Most of chromite ores are processed in this type of plants. It is highly questionable that they are operated at optimum conditions. In order to take engineering based decisions to improve their profitability, detailed performance evaluation should be carried out time to time. In these circumstances, sampling and data collection is difficult since operating variables could be changed considerably during a short period due to the lack of measuring and control devices. Therefore, the data deserves a cautious treatment. The methodology to be applied in these circumstances should take the undesirable conditions into account.

In this paper, the methodology used in the performance evaluation of Sori Plant which represents a typical chromite processing plant in Turkey, as well as problems encountered and experiences gained are presented. After evaluating the data gathered, some modifications in the plant flowsheet were proposed for improving the performance. In contrast to its backward nature, modern tools such as modelling and simulation were used for estimating plausible improvements. The ideas for a better performance were also outlined.

DESCRIPTION OF PLANT

Performance evaluation studies were performed at the Sori Plant where shaking tables are used for beneficiation. The plant is designed for a tabling unit capacity of 10 t/h to produce a chromite concentrate of minimum 48 % Cr_2O_3. The ore is massive Alpine type and occurres as several small deposits. The gangue mineral of high grade (38–45 % Cr_2O_3) and relatively coarse grained ore is serpentine.

The simplified flowsheet of the plant is exhibited at Figure 1. Run of mine ore is, first, screened from 25.4 mm. Then, the oversize is transported to hand sorting unit, while undersize undergoes another screening through 6 mm (2 mm in original design).

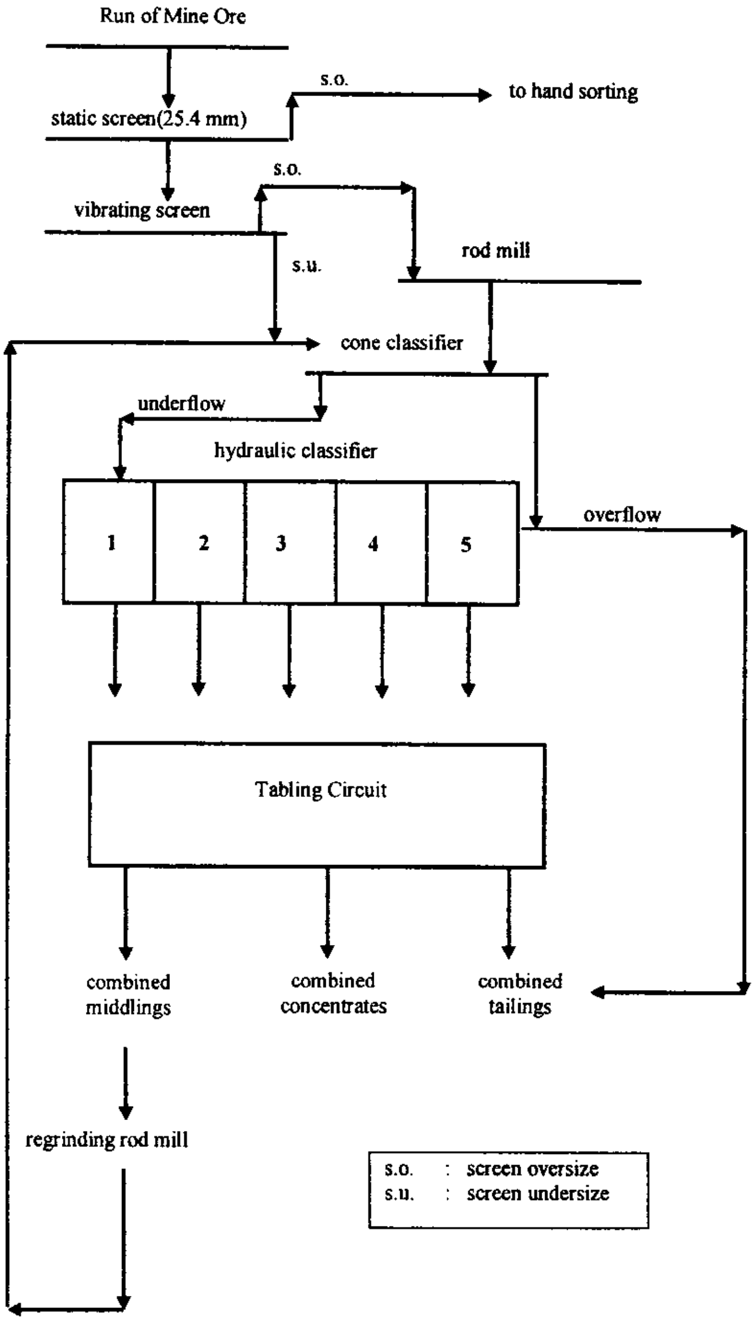


Figure 1. Simplified flowsheet of Sori plant

After rod mill grinding of -25.4+6 mm fraction, the rod mill product and screen undersize (-6 mm) are combined and fed to cone classifier for the separation of slime and excess water from the circuit. The slimes in the overflow of cone classifier is transported to tailing streams and the underflow is sent to a hydraulic classifier. Each classified product of hydraulic classifier is fed to a different set of shaking tables. The underflows from compartments 1, 2, 3, 4 and 5, are fed to 8, 3, 2, 2 and 1 shaking tables each having 8 m^2 deck surface, respectively. The middling obtained from the shaking tables concentrating the classified products of compartment 1 and 2 are sent to a regrinding rod mill and ground to -0.42 mm. The regrinding mill discharge is recycled to the cone classifier.

Apart from the belt scale weighing the feed flow, no instrumentation is available in the plant. Even the hand sampling points designed in the original flowsheet were obstracted by the modification made during the years. The daily metallurgical balance is based on the assays of three samples, namely feed, combined concentrate and tailings. Operators maintain the manual controls in the plant.

SAMPLING AND EXPERIMENTAL STUDIES

The lack of sampling devices in the plant rendered sampling very difficult. The overflows of the cone classifier and hydraulic classifier could not be sampled, since these two were transported in closed pipes to a thickener which could not be sampled either. The flow rates and solid ratios of rod mill overflows could not be measured, since there was not sufficient space for sampling cans at these points. Similarly, the physical conditions did not permit taking of reliable samples for flow rate and solid ratio measurements from the hydraulic classifier. All the remaining points were sampled.

Since the flow rates and operating parameters were possibly fluctuating in a very short period, it was considered that regular sampling throughout a relatively large period could provide samples representing the average behaviour of the plant. After examining the available data about the performance of the plant, it was decided that the length of such period should be about a week.

Two sampling surveys of one week long were carried out at the plant. Samples were taken from each point every hour during two shifts in a day. At the end of the week long survey they were combined and dried. Then, the amount of sample was reduced by coning and quartering. Consequently, they were transported to the laboratory where they were sieved and assayed for Cr_2O_3. The Bond Work Index of the ore was determined in the laboratory.

MASS BALANCING

It is well known that, whenever the excess data is collected, it contains some errors. These may be due to system dynamism, sampling equipment, operator, aging, bias etc. Therefore, it is necessary to balance the data before doing any further evaluation (Houdin et al. 1980). There are several mass balance algorithms presented in the literature (Richardson et al. 1987). They use same principles i.e., the least squares constrained optimization problem. At the time of the study, the commercially available mass balancing softwares were not in the reach of authors.To simplify the problem, a sequential methodology for mass balancing was followed. Starting from the first separator, i.e., screen, the best fit flow rates and corrected assays were calculated by using a house made program written for this purpose. For the following separators, the assays and flow rates of streams coming from previously calculated and adjusted streams were assumed to be the true values. The hydraulic classifier was the most problematic unit of the circuit in terms of calculations, since there were three incoming and five outgoing stream. A unique program was written for this unit. A constrained simplex approach (Carpenter et al. 1965) is used for the calculation of the flow rates around it. In the calculations, particle size of two unsampled stream, the overflows, was assumed to be 100 % -74µm. Although the reliability of the method used for mass balancing can be argued, it was the most practical approach available at the time of study. Later, the raw data were also inputted to a commercial mass balancing program and very similar results were obtained. However, the hydraulic classifier data from the first sampling period could not be balanced by either of the two probably due to large errors in assay values. Therefore the study was mainly based on the data obtained from the second sampling survey.

CALCULATION OF PERFORMANCE CRITERIA

After the calculation of best fit flow rates and adjusted assay values, the performance of individual equipments were examined by using conventional criteria. The equipments were screen, rod mills, hydraulic classifier and shaking tables.

The criterion used for measuring screen efficiency was imperfection (I). The partition curves were drawn and imperfection was calculated for each period using the equation below.

$$I = \frac{d_{75} - d_{25}}{2d_{50}} \tag{1}$$

d_{25}, d_{50}, d_{75}: sizes at which 25, 50 and 75 % of feed reported to the coarse product, respectively

The performance of rod mills was examined by taking account of feed and product size distributions, Work Index of the ore, the dimensions of the mills and the powers of their drives.

The hydraulic classifier was symbolized by a series of simple separators having one feed and two product streams. This enabled the calculation of partition coefficients for each compartment. From the partition curve, imperfection, I, was calculated and used as an indicator of the quality of classification at different chambers.

Although ideally heavy liquid tests are required to assess the performance of gravity separators, performance evaluation of shaking tables was mainly based on their grade and recovery figures due to inavailability of high-density liquids. The length and frequency of stroke were also measured, whereas the measurement of wash water addition to tables was not possible.

PLANT PERFORMANCE

For the evaluation of plant performance, the design principles and methodology outlined by Ottley (1986) and Wells (1991). The design principles were found to be an excellent quide for both evaluating the existing flowsheet and proposing flowsheet modification. The methodology, however, could only be applied where the instrumentation and laboratory facilities permit its use.

The average grade of the feed was 41.90 % Cr_2O_3 with average flow rate of 10 t/h. The flow rate calculations of the remaining streams based on the measured flow rate of the feed. The flow rates and adjusted assays of main streams are presented at Table 1. The extend of data adjustment needed for mass balancing is illustrated in Figure 2.

Table 1. Mass flow rates and Cr_2O_3 assays of the main streams

stream →	plant feed	rod mill discharge	table feed	combined concentrates	combined tailings	overflow
weight (t/h)	10.00	4.84	11.60	7.47	2.53	0.23
Cr_2O_3 (%)	41.90	41.90	41.06	50.35	16.92	23.00

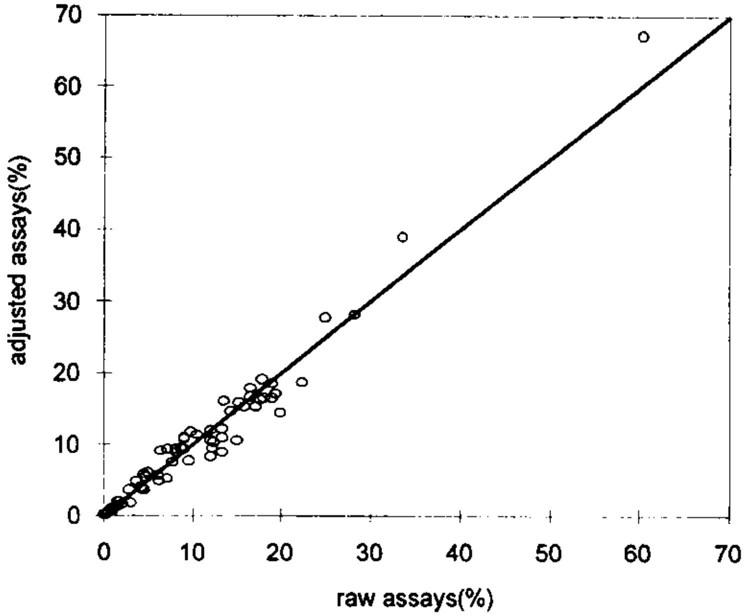


Figure 2. Raw vs adjusted assays

Partition curves of the screen for two sampling period are shown in Figure 3. The shape of the curves indicates screening operation was efficient. Imperfections and d_{50} values for the two sampling periods are presented at Table 2. These values also verify that screen was working properly. Significant difference between d_{50} values of the two periods aroused the question whether the screen aperture was different at each. Close examination of size distributions revealed that this was only due to the changes in feed size distribution.

Table 2. d_{50} and imperfection values of the screen

	period I	period II
d_{50}(μm)	2830	3813
Imperfection	0.226	0.189

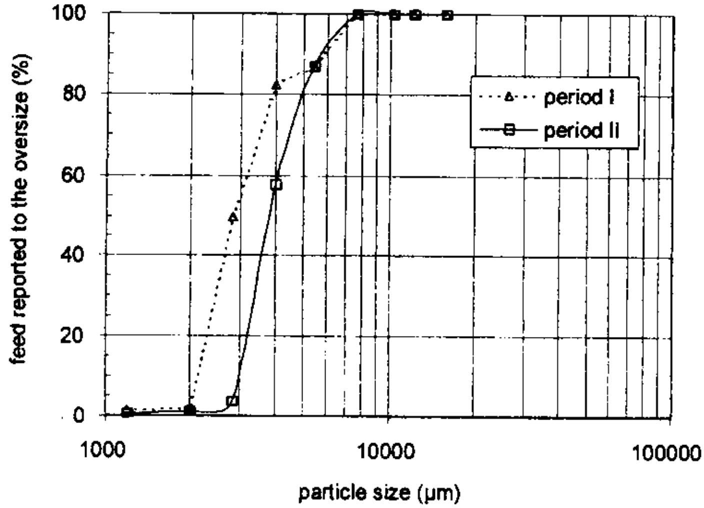


Figure 3. Partition curves of screen for two sampling periods

The partition curves for each hydraulic classifier compartment are presented in Figure 4. Imperfections and d_{50} values of each compartment are given at Table 3, excluding Compartment 5 whose product was finer than 100μm. Compared to the data available in the literature (Wells 1991; Burt 1984) both imperfections and d_{50} values indicate that the classification within hydraulic classifier was not efficient. d_{50} values are too close to each other and imperfections are too high. This was expected since the control system of the hydraulic classifier was not operational.

Table 3. d_{50} and imperfection values of hydraulic classifier compartments

	Compartment			
	1	2	3	4
d_{50}(μm)	330	135	125	80
Imperfection	0.58	1	0.76	0.72

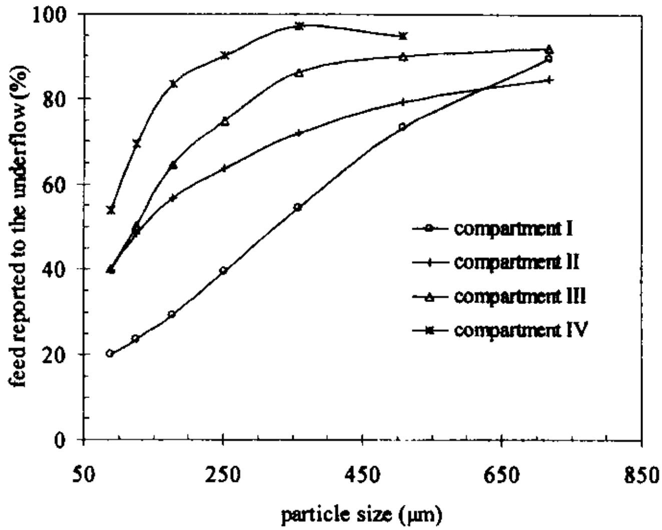
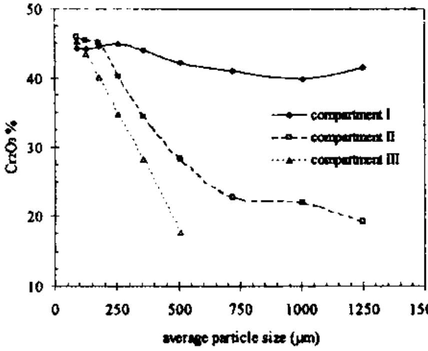


Figure 4. Partition curves of hydraulic classifier compartments

Variation of Cr_2O_3 contents with particle size for different chambers of hydraulic classifier are shown in Figure5. Preferential settling of chromite particles in the first chamber is clearly seen. This is also apparent in the second chamber, but in a much lesser extent. Since 95% of total Cr_2O_3 is in particles coarser than 74μm (Figure 6), very favourable metallurgical performance could be expected from tables treating the product of chamber I.



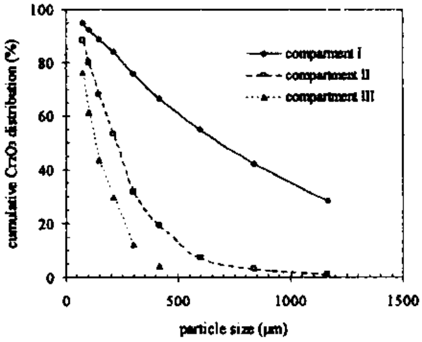


Figure 5. Variation of Cr_2O_3 contents with particle size at each compartment of hydraulic classifier

Figure 6. Variation of cumulative distribution of Cr_2O_3 with particle size at each compartment of hydraulic classifier

The properties and operating parameters of rod mills are presented at Table 4.

Table 4. Properties of rod mills and their operating parameters

	Rod Mill	Regrinding Mill
Diameter (cm)	91	61
Length (cm)	183	183
Rod Diameter (cm)	10	5
Number of Rods	36	28
Power (kW)	22	7.5
F_{80} (μm)	16470	865
P_{80} (μm)	1660	850

Feed and product size distributions of primary rod mill and regrinding rod mill for two sampling period are shown in Figure7 and Figure 8, respectively. Bond Work Index of the ore was determined to be 11.5 kWh/ton. From the available information, the following comments could be drawn:

a) Slime production is at acceptable levels.
b) The regrinding mill is not operating efficiently. The desired level of size reduction, 80% -0.42 mm, is not accomplished, despite the fact that only small fraction of available grinding power appears to be utilized. It is also arguable that a rod mill rather than a ball mill should be used for regrinding.

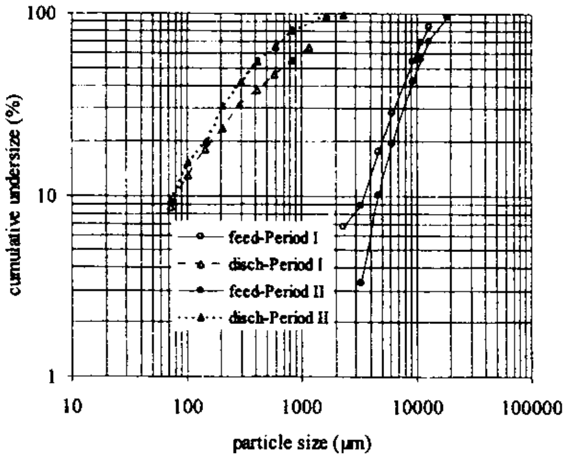


Figure 7. Particle size distributions of feed and discharge of primary rod mill

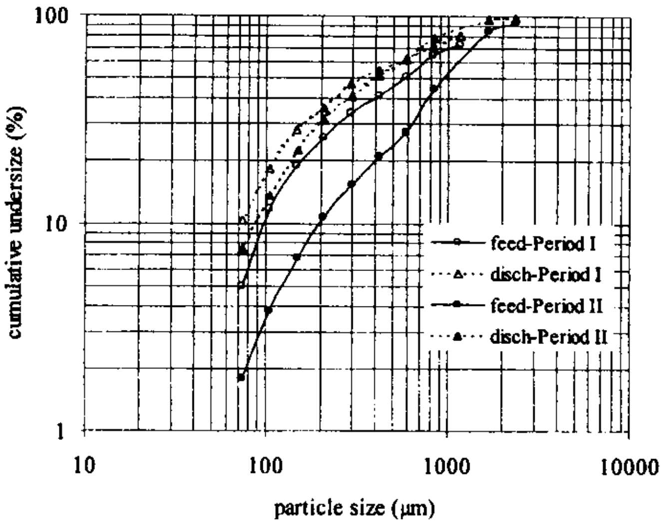


Figure 8. Particle size distributions of feed and discharge of regrinding mill

The performance values of shaking tables are presented at Table 5. The following conclusions were drawn:

a) While high grade concentrates with high recoveries were obtained from the first two sets of tables, the third, fourth and fifth sets did not perform so well both in terms of grade and recovery. However, the overall recovery and grade were satisfactory, mainly due to the fact that 87.8 % of total concentrates were recovered from the first set of tables.
b) The differences between the performance values of tables with and without middling product indicates that the performance of the third, fourth and fifth sets of tables could be improved by taking middlings from them and circulating directly to the cone classifier.
c) The measured capacities varied between 0.2 and 0.875t/h/deck indicating that, particularly at coarse size range the full capacity of tabling circuit is not utilized (Kelly et al. 1982, Burt 1984).
d) Microscopic examinations indicated that middlings are made of mainly liberated particles. Therefore, the necessity of regrinding was found to be questionable. The middling from the first set of tables may be combined with its concentrate. Then, the combined concentrate grade would only be reduced to 48.86 % Cr_2O_3. The circulation of middling from the second set of tables to cone classifier would be more suitable treatment for this product.

e) The length of stroke was varied between 30–60mm with frequencies between 160–180 rpm. They do not appear to be the suitable values for this type of operation (Burt 1984).

Table 5. Mass balance of tabling circuit

name of the product	Cr_2O_3 (%)	flow rate(t/h)	recovery (%)
the first set of shaking tables			
concentrate	51.30	5.31	84.01
tailing	11.30	0.53	1.85
middling	38.08	1.20	14.14
feed	46.02	7.04	100.00
the second set of shaking tables			
concentrate	50.00	1.25	76.49
tailing	11.10	0.69	9.33
middling	29.93	0.40	14.19
feed	34.96	2.34	100.00
the third set of shaking tables			
concentrate	42.60	0.61	75.25
tailing	19.40	0.44	24.75
feed	32.87	1.05	100.00
the fourth set of shaking tables			
concentrate	49.00	0.17	45.73
tailing	26.30	0.36	54.27
feed	33.37	0.53	100.00
the fifth set of shaking table			
concentrate	53.00	0.14	56.49
tailing	20.80	0.27	43.51
feed	31.67	0.41	100.00
overflow	23.00	0.23	1.28
combined middling	35.80	1.60	12.05
combined concentrate	50.35	7.47	89.79
combined tailing	16.92	2.53	10.21
feed	41.90	10.00	100.00

Liberation characteristics of the ore was also determined by examining closely sized fractions of the feed to shaking tables under a binocular microscope. The results showed that the ratio of liberated chromite particles decreased sharply above 2 mm.

Below this limit, they were mostly liberated, the degree of liberation for the coarsest fraction in this range is 75%. Although this confirmed that the level of grinding achieved in the rod mill was satisfactory, it indicated that the use of 6mm aperture screen, as opposed to 2mm in the original design, prior to rod milling was creating a detrimental effect for the concentration circuit.

Although an overall concentrate grade of 50.35 % Cr_2O_3 with 89.79 % recovery gives impression that the plant is operating well, it seems that even better performance could be achieved by proper adjustment of the operating parameters of hydraulic classifier and shaking tables. However, as pointed out earlier, the additional improvements in terms of efficiency criteria and, particularly, plant economics could be achieved by making some changes in plant flowsheet. For this purpose, three modifications were found to be worth of further investigation. They were:

- Changing 6 mm screen aperture to its original size of 2 mm. In this case, the amount of oversize material passing through the screen would be avoided, however the amount going to the primary rod mill would be increased.
- The necessity of regrinding is questionable. Although some oversize materials were fed to the tabling circuit, they did not report to middlings which were mostly formed by liberated particles. Therefore, the circulation of middlings to the cone classifier without regrinding could be suggested.
- Since shaking tables treating the coarsest fraction have large ample capacity(8–9t/h), it seems that the capacity of the plant could increased by operating two rod mills, i.e. primary and regrinding mills in parallel.

The second modification could only be tested in the plant, but the other two could be investigated by computer simulation. Simulation would have an additional benefit of investigating the effect of variations in capacity.

MODELLING OF SCREEN AND ROD MILL

Simulation studies concerned with screening and grinding units. Although mathematical models for hydraulic classifier (Mackie et.al 1987) and shaking table [Manser et.al 1986, Razali et.al 1990, Tucker et.al 1991, Manser et.al 1991) were available in the literature, since pulp solids contents of the products and water flow rates could not be measured, the use of these models had not been possible. The objective was to obtain a product size distribution suitable for achieving better concentration performance. Therefore, only the mathematical models of screening and rod mill grinding were required. The model developed by Whiten (1972) (Eq. 2), was used for modelling of screen. The model included another parameter (k_3) to define the carryover of the fines with oversize material. The model parameters k_1 and f were considered constant within the range of feed flow rates to be investigated.

$$E(x)=\left(1-\left(\frac{h-x}{h+d}\right)^{2}\right)^{m} \quad (2)$$

E(x), the probability that a particle of size x which does not pass through the screen in m trials, where $m = k_1^2 \ell . f$.

x – particle size

h – screen aperture

d – diameter of screen wire

k_1 – efficiency parameter

ℓ – length of screen

f – load factor in screen model (1 for normal operation)

The efficiency parameter, k_1, was calculated by using a non–linear regression algorithm. Since there was significant differences in the partition curves obtained from the two sampling periods, screen aperture was also considered as a variable and its best fit value calculated separately for each set of data. The estimated values for screen aperture were 5.81 and 5.84 mm with k_1 values of 2.46 and 2.56 respectively for the first and second periods. These values verified the earlier views that nominal screen aperture was 6 mm in both periods and screening efficiency was satisfactory (on average k_1 should have a value around 2.6). The parameters of the second period were used in further simulation work. For the rod milling, matrix model given by Lynch (1977) was used.

$$p = \prod_{j=0}^{j=v} T_j \cdot \; f \quad (3)$$

where $T_j = (I-C)(B.S+I-S)[I-C(B.S+I-S)]^{-1}. \; f$

p: product vector; T_j: transition matrix; f: feed vector; v: number of breakage stages; I: unit matrix.

The model based on the selective breakage of the coarsest fraction. The mill is divided into imaginary compartments and the coarsest fraction entering a compartment will be completely broken down into the finer fractions. The model consists of three functions, namely classification, breakage and selection. Classification function (*C*) defines screening effect of the rods, breakage function (*B*) primary breakage distribution and selection function (*S*) the proportion of size fraction selected for breakage. For each compartment, the value of classification function for the coarsest size fraction was chosen as one. Breakage function is defined by Broadbent and Calcott equation (Lynch 1977) (Eq. 4):

$$B(x/y) = \frac{1 - e^{\left(\frac{-x}{y}\right)}}{1 - e^{-1}} \tag{4}$$

where y is the maximum particle size.

For the selection, a function defined by three parameters was used (Lynch 1994, Napier–Munn et al. 1996). This is illustrated in Figure 9. The number of coarsest fraction disappearing at the discharge of a mill is defined as the number of stages of breakage and this parameter is used in simulating the effect of feed rate on the product size distribution. The relationship is defined as mill constant and given below (Napier–Munn et al. 1996).

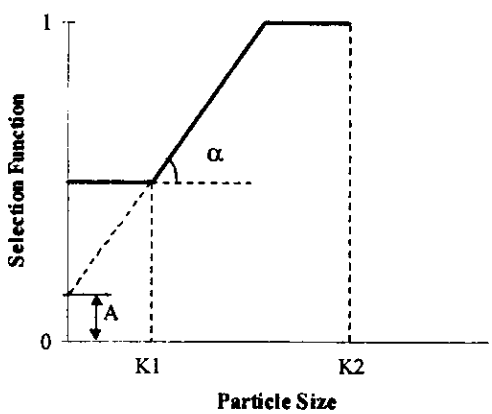


Figure 9. Schematic representation of selection function

$$(F.\nu)^{1.5}=MC \tag{5}$$

F – feed rate(t/h)
MC – mill constant
ν – stages of breakage

For the simulation studies, calculation of three parameters of selection, and mill constant were required. A non–linear regression algorithm was used for this purpose. Both sets of data provided similar parameters and satisfactory fit to the data, and those of the second were used further studies.

Since regrinding rod mill was not operating efficiently, the use of data obtained its feed and product size distributions for the calculation of model parameters would be misleading. Then, model parameters of primary mill scaled down the using relationships developed at Julius Kruttschnitt Mineral Research Center of Australia (Napier–Munn et. al 1996). This approach provided reasonable simulation results. The expressions used for mill scaling, Eq. 6 and Eq. 7, are given below.

For scaling number of breakage stages, when feed rate and feed size distribution were changed.

$$\nu_{SIM} = \left(\frac{MC_{FIT}}{Feedrate_{SIM}}\right)^{2/3} + ln\left(\frac{F90_{FIT}}{F90_{SIM}}\right) / ln\sqrt{2} \tag{6}$$

Subscripts SIM and FIT denotes parameter values for simulated and fitted conditions respectively. For scaling the mill constant, for a mill of different dimensions and operating conditions:

$$\frac{MC_{SIM}}{MC_{FIT}} = \left(\frac{D_{SIM}}{D_{FIT}}\right)^{2.5} \cdot \left(\frac{L_{SIM}}{L_{FIT}}\right) \cdot \left(\frac{1-LF_{SIM}}{1-LF_{FIT}}\right) \cdot \left(\frac{LF_{SIM}}{LF_{FIT}}\right) \cdot \left(\frac{fCs_{SIM}}{fCs_{FIT}}\right) \tag{7}$$

where LF=mill load fraction (volume of mill occupied by charge and media after grinding out), fCs = mill speed (as fraction of critical speed).

The models for both screen and rod mill provided very good fit for the existing data and this is illustrated in Figure 10. However, it should be noted that fitting of rod mill model parameters requires some trial and error for initial estimates of the parameter values.

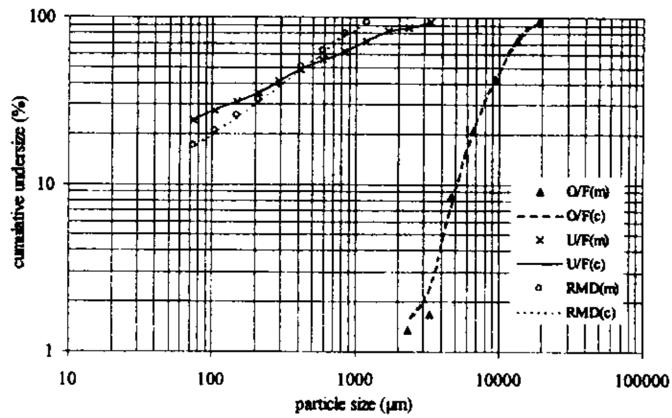


Figure 10. Model fits for screen and rod mill discharge (RMD); (m and c refers to measured and calculated, respectively)

SIMULATION STUDIES

In simulation studies, the following three points were investigated:

i. The effect of changing screen aperture from 6 mm to 2 mm on the size distribution of circuit product.
ii. The limiting capacity for the plant.
iii. Possibility of using the two rod mills in parallel to increase grinding capacity for utilizing full capacity in the tabling circuit.

Separate house made computer programs were written for the simulation of each type of circuit configuration

Since the liberation size of chromite was around 2 mm and a product low in slimes was desired, the amounts of +2 mm and -74μm sized material in the circuit product were selected as main evaluation criteria. The effect of feed rate on product size was investigated in a range between 8 and 16t/h.

As expected, the results indicated that changing screen to its original aperture, 2 mm, would be beneficial in terms of the proportion of -2mm material in the circuit product (Figure 11), since there would always be some oversize material in the circuit product when 6 mm screen aperture was used. Screen aperture did not have any significant effect on the amount of slimes produced (Figure 12). Decreasing screen aperture to 2 mm results in an increase in the rod mill feed rate. Therefore, the rod mill discharge rapidly becomes coarser with the increase in feed rate. Independent of screen aperture, the fraction of +2 mm sharply increased with the plant feed rate above 12 t/h. Since the fraction of slimes decreased with increased flow rate, it would be suggested that the plant should be operated with feed rate as high as possible.

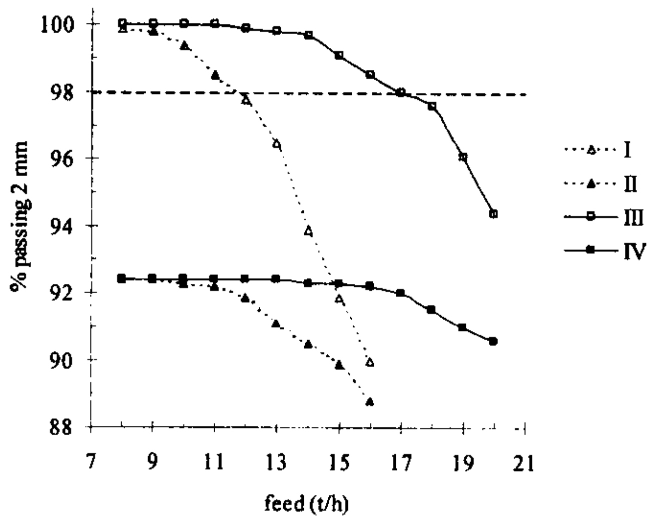


Figure 11. Variation of % passing 2 mm with the simulated plant feed rate(I and II for the primary rod mill with 2 and 6 mm screen apertures respectively;III and IV for the two mills in parallel with 2 and 6 mm screen apertures respectively)

Simulation results indicate that the optimum flow rate would be a good compromise between the two. Under these condition 98 % of the circuit product will be finer than 2 mm, when a screen with an aperture size of 2 mm is used. The same flow rate with a screening size of 6 mm would produce about 8 % oversized material. Such modification would also result in decreased middlings from tabling circuit. Before simulating the case with two mills in parallel, the ratio of rod mill circuit feed to be sent to the regrinding mill estimated by using the second period data so as to produce similar size distribution. It was found that the ratio should be 1 to 2.15.

Simulation studies were carried out with a feed rate ranging between 10–20 t/h. The results are summarized in Figure 11. It is shown that the capacity of the plant could be increased to 17 t/h. Above it, the proportion of oversize sharply increase. At this capacity, the amount of feed to the primary mill and regrinding mill would be 7.15 and 3.32 t/h, respectively. It should however be pointed out that running two mills in parallel increases the proportion of slimes and further deterioration in the performance of the hydraulic classifier is expected due to increased capacity (Wells

1991). Therefore, it would be recommended that they should only be used if the extra capacity created was fully utilized.

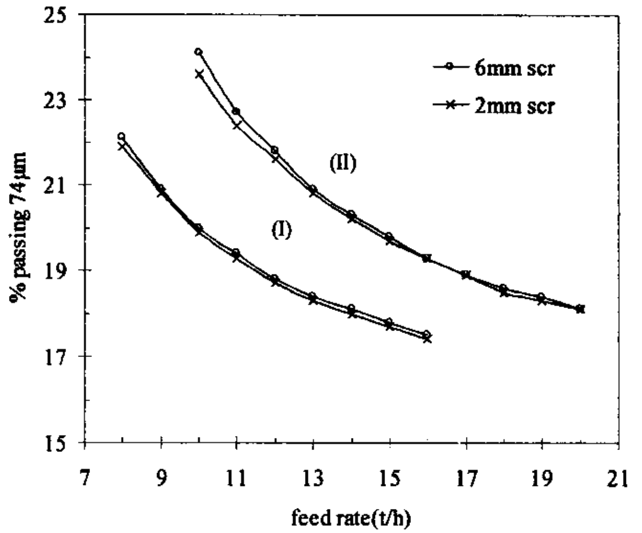


Figure 12. Variation of % passing 74 µm with the simulated plant feed rate
(I the primary mill only; II the two mills in parallel)

CONCLUSIONS

The performance evaluation study carried out at Sori Plant indicated that hydraulic classifier, shaking tables and regrinding rod mill were not operating properly, despite of the fact that a concentrate of 50.35 % Cr_2O_3 with recovery of 89.79 % was produced. It was also found that the full capacity of tabling circuit was not utilized.

Simulation studies indicated that the plant performance could be improved and the capacity could be increased by making some modification in the plant flowsheet only.

It is also shown that modern tools such as mathematical models and computer simulations could be used under the circumstances experienced in a plant with a very limited measuring and control devices. It appears that proper sampling is the key for success. It should however be coupled with detailed and accurate laboratory studies.

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W pracy przedstawiono wyniki badań dotyczące zakładu wzbogacania chromitu. Zakład nie jest wyposażony w aparaturę kontrolno–pomiarową. Dlatego ocena wzbogacania jest trudna i wymaga specjalnej metodologii. Badania rozpoczęto od opróbowania procesu, a tygodniowe pobieranie próbprzeprowadzono dwa razy. Po szczegółowej ocenie danych dotyczących procesu zaproponowano kilka modyfikacji, np.: zmiana rozmiaru sit z 6 do 2 mm, ograniczenie zdolności produkcyjnejoraz zastosowanie dwóch młynó prętowych ustawionych równolegle. Następnie zbadano wpływ tych zmian na pracę zakładu, dzięki symulacji komputerowej. Wyniki wzbogacania oraz przerób ulegną zwiększeniu po zastosowaniu proponowanych zmian.