Investigation into Jig Performance

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Mr. D. J. Batzer: First I would like to apologize for the omission from the paper of the important item mentioned by Mr. D. G. Armstrong, Mr. R. C. M. van der Spuy, and Mr. I. R. M. Chaston—namely, the average depth of sand on the jig bed. This was about 1 in., which is less than the critical depth mentioned in the paper by Lill and Smith (reference 1, p. 68, of my paper). The feed rate to the jig was deliberately kept light to avoid 'overload' effects, since it was not intended in this phase of the work to investigate the relationship between feed rate and jig recovery. As pointed out by Mr. P. M. Sheahan the light loading of the jig was probably one of the reasons for the generally good recoveries obtained. Tests so far carried out indicate that on a full-scale Ruoss jig the critical depth of sand on the first hutch varies from about 1½ to 3 in., depending on conditions.

Mr. J. H. Harris asked for details of tests on the method of treating the spigot product and tailing to estimate recovery. Since about 80 per cent of the cassiterite in the material used in the tests was between 52 and 100 mesh B.S.S., with very little below 150 mesh and none above 10 mesh, the cassiterite was in effect of short size range and therefore an overall recovery test was made and not one for each mesh fraction. A weighed amount of cassiterite (75 per cent Sn) of the same size distribution as that in the test material was mixed with about three times its weight of ilmenite, tourmaline, zircon and rutile, these being the most important heavy minerals present besides cassiterite, and some sand (−10 mesh B.S.S., see paragraph 5, page 62) cleaned by careful and repeated dulang washing. This made-up specimen was then washed in the dulang once and the resulting concentrate cleaned by the tin dresser in the same way as in the jig tests. Of 10 g of cassiterite (between 10 and 200 mesh B.S.S.) added to ⅓ ft³ of sand, 9·2 g was re-extracted as 75 per cent Sn concentrate, indicating a recovery of 92 per cent. If there were a dip in recovery of the 'middle size' ranges by both the jig and dulang then, as pointed out by Mr. Harris, the pattern of recovery shown in Table I (p. 67) would be expected. As a check, the tailings from the above dulang washing test were screened at 25 and 72 mesh B.S.S. Each fraction (i.e. 10/25, 25/72, and −72 mesh) was then rewashed in the dulang separately. Since most of the concentrate is between 52 and 100 mesh the middle-size range in the three new fractions should not coincide with that in the original unscreened material. The additional cassiterite recovered amounted to less than 1 per cent of that recovered in the first washing and it is therefore concluded that the losses which result from washing the −10-mesh sand, without

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further screening, are not large enough to affect the pattern of jig test results materially.

Mr. Chaston remarks that the dulang washing procedure must have taken a considerable time. The method described is, however, the routine one used by Anglo-Oriental (Malaya), Ltd., for washing all bore samples in the field, although in addition bore samples are screened at ¼-in., ⅛-in., 20, 40 and 60 mesh nominal before dulang washing, as this procedure has been found to enhance recovery slightly, as compared with washing of unscreened samples. I agree that Mr. Chaston may be correct in stating that the improvement in recovery with −10-mesh feed in the test results given in Table II (p. 68) may have been solely due to better dilation with that feed under those conditions. It should be pointed out, however, that the jigging conditions were deliberately chosen to give about 90 per cent recovery with the −¼-in feed to allow room for improvement.

I regret that the construction of the plunger was not clear in the paper and in reply to Mr. Armstrong's query confirm that the plunger of the jig used for the tests contained no valve. The smaller the jig the larger the plunger circumference to plunger area ratio, and the clearance between plunger and jig body can therefore be made correspondingly smaller. When the make-up water was turned on with the jig in operation, a head of water built up above the plunger to a height of some inches above the top of the jig bed. This head was responsible for the flow of water into the jig hutch past the plunger. It seems likely that make-up water flowed past the plunger throughout the cycle although I am unable to confirm this definitely or to determine any variations in this flow rate. When the jig is run without feed and the make-up water adjusted to balance the water flow through the spigot, after adjustment of the water level in the jig to just below the point of overflow, the movement of the water at the surface that Mr. Armstrong has calculated at 15·3 gal/ft²/min (p. 357), while the figures given in the paper refer to the much smaller make-up water current only. The total flow rate through the spigot was about 2·3 gal/min in all tests, being determined by size of the spigot hole and the head of water over it.

Mr. Sheahan's work with the stroboscope is interesting and should be useful, provided adjustment to give optimum conditions at the wall also results in optimum conditions in the jig bed generally, or provided the differences can be allowed for. When the details of the small Hartz jig used in the tests were under consideration, the installation of a glass side to the jig was contemplated but rejected, because the 'wall effect' zone, however narrow this may be, is all that can be observed through a glass side facing an opaque bed.

Mr. F. A. Williams asked about means of measuring water flow rates.

When the make-up water was turned on, a head of water built up above the plunger to a height of some inches above the jig bed. The plunger and jig body can therefore be made correspondingly smaller. The smaller the jig the larger the plunger circumference to plunger area ratio, and the clearance between plunger movement and it is apparently these latter currents due to the plunger movement and it is apparently these latter currents in amplitude between the water movement and the plunger movement at a movement but 180° out of phase with it. There is no measurable difference of the ragging is clearly seen to be a good reproduction of the plunger stroke of 0·25 in. and 130 strokes/min although presumably there must in fact be some 'slip'. The relatively small current due to the make-up water is superimposed on the very much larger pulsion and suction currents

This was done simply by running the feed water, make-up water, or spigot discharge separately into graduated containers for a measured period of time, then calculating the flow in gal/min. The feed and make-up water were supplied from a constant-head tank and were adjusted to the desired flow rate before each test.

I was particularly interested in Dr. D. J. Brown's contribution and would like to thank him for suggesting this approach, which should prove fruitful.

The main consideration behind the conception of the tests described was to further the quantitative description of jig operation. This is clearly no easy task as jigs have been in use, in much their present form, for several decades and so far adjustment of the operating variables has generally been by trial-and-error methods, which, while they enable an experienced operator to obtain an adequate performance, are insufficient to ensure that the best possible performance is being realized. The displacement as represented by the plunger motion in one direction only (since the movement of the plunger is the same in both directions but of opposite sign) and the intensity of stroke were therefore chosen as two of the parameters for plotting the data acquired in the tests, in order to make a quantitative assessment possible. This is admittedly a matter of expediency and it is hoped that other workers on the problems of jigging may be able to improve on the author's description.

Natural Leaching of Uranium Ores

1—Preliminary tests on Portuguese ores
R. P. MILLER, B.Sc., A.R.I.C., ELIZABETH NAPIER, B.Sc., and R. A. WELLS, B.Sc., F.R.I.C., MEMBER

2—A study of the experimental variables
A. AUDSLEY, Ph.D., F.R.I.C., and G. R. DABORN

3—Application to specific ores
A. AUDSLEY, Ph.D., F.R.I.C., and G. R. DABORN

Authors' reply to discussion on paper published in January, 1963, pp. 217-54*

Mr. Miller, Miss Napier, Mr. Wells, Dr. Audsley and Mr. Daborn: In reply to Mr. Cahalan's remarks, we have not attempted to use an external leaching circuit with uranium ores. We should like, however, to comment on the general principles involved in this process

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*pp. 507-17.

which employs 'the use and continual regeneration of a ferric sulphate sulphuric acid lixiviant' and incorporates iron oxidizing bacteria. The Kennecott patent states that the method can be used to recover the copper (presumably occurring in the form of sulphide) from pyritic copper ore and to extract the zinc from zinc sulphide ores, but when it is applied to the treatment of mixed oxide-sulphide copper ores the copper which is present as oxide is leached selectively, the unattacked copper sulphide remaining in the tailings. These claims, together with the results of our tests with uranium ores which contain tetravalent uranium (UO_2), suggest that the use of external oxidation alone is most suited to the extraction of metals from those compounds which require only a mildly oxidizing lixiviant, i.e. one in which the ratio of Fe^{3+}/Fe^{2+} in solution is quite low. For the efficient attack on those compounds whose breakdown can only be accomplished with lixiviants of high Fe^{3+}/Fe^{2+} ratio, then the presence of pyrite within the ore itself appears to be necessary, i.e. an internal as well as an external source of oxidation is required. The reason for this is discussed later.

Some quantitative idea of the relative ease with which zinc, copper or uranium can be extracted from their oxides or sulphides by ferric sulphate/sulphuric acid solutions can be obtained by calculating the free energy of the chemical reactions involved, the products of which are those which would be formed if oxidation were being promoted by Fe^{3+} ions in acid media. The values (ΔG) for some typical compounds are shown below, together with the calculated values of the oxidation potentials of the systems concerned.

Compound	Free energy, (ΔG), kcal	Potentials, V
Zinc sulphide $ZnS = Zn^{2+} + S + 2e^-$	$+8.0$	-0.17
Cuprous oxide $Cu_2O + 2H^+ = 2Cu^{2+} + H_2O$	$+10.3$	-0.22
Uranous oxide $UO_2 = UO_2^{2+} + 2e^-$	$+15.4$	-0.33
Cuprous sulphide $Cu_2S = 2Cu^{2+} + S + 4e^-$	52.0	0.57
Cupric sulphide $CuS = Cu^{2+} + S + 2e^-$	27.7	0.60

It will be seen that the potentials range from -0.17 V for zinc sulphide to -0.60 V for cupric sulphide, indicating that the energy required to promote attack increases down the series and that if oxidation is being promoted by Fe^{3+} ions the ratio of Fe^{3+}/Fe^{2+} required in solution will have to be correspondingly increased; it is significant that the potential required to decompose cupric sulphide approaches that of the ferrous/ferric couple (-0.77 V).

Rapid attack upon the compounds listed would be expected to take place only at negative voltages rather higher than the theoretical values given. Confirmation of this has been shown during the treatment of uranium ores (containing UO_2) by conventional leaching techniques when, in general, it was found that the lowest negative voltage at which uranium was satisfactorily extracted was of the order of -0.37 V (cf. the theoretical value of -0.33 V for $UO_2 = UO_2^{2+} + 2e$). Confirmation of the high voltages required for leaching copper from its sulphides was shown in bench tests in which ground chalcopyrite was agitated with acid oxidizing media for 24 h when it was found that dissolution of copper was very slight when the