A STUDY OF GRAVITY CONCENTRATION WITH EMPHASIS ON SURFACE PHENOMENA

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A Study of Gravity Concentration
with emphasis on Surface Phenomena

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Part I A Review of the Literature on Intermediate and Fine Particle Gravity Concentration

A  Principles of Sluicing
   (Accepted for publication in the International Journal of Mineral Processing, Elsevier, Amsterdam)

B  Principles of Spiral Concentration
   (Accepted for publication in the International Journal of Mineral Processing, Elsevier, Amsterdam)

C  Principles of Tabling
   (Accepted for publication in the International Journal of Mineral Processing, Elsevier, Amsterdam)

D  Recovery of Heavy Minerals from Slimes
   (Accepted for publication in the International Journal of Mineral Processing, Elsevier, Amsterdam)
Part 2 Surface Phenomena as applied to Slime Gravity Concentration

E Dynamic Flow Characteristics of Thin Films
(Accepted for publication in the Scandinavian Journal of Metallurgy)

F Roughness Profiles - A way to recover heavy minerals from slimes
(Submitted for publication in the Scandinavian Journal of Metallurgy)
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G Influence of Electrokinetic Environment in Slime Gravity Concentration
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A summary of this thesis is to form the major part of the paper "Progress in Gravity Concentration - theory and practice" to be presented at the "Arbiter Symposium on Advances in Mineral Processing" which will be held in conjunction with the 1986 annual meeting of the Society of Mining Engineers of AIME in New Orleans, U.S.A, in March, 1986. The authors are E. Forssberg, T. Nordquist and R. Sivamohan.
To Sumathy
CONTENTS

Study of Gravity Concentration ........................................... 1
   Introduction
   References
   Acknowledgements

Paper A: Principles of Sluicing ...................................... A1
Paper B: Principles of Spiral Concentration ..................... B1
Paper C: Principles of Tabling ....................................... C1
Paper D: Recovery of Heavy Minerals from Slimes............. D1
Paper E: Dynamic Flow Characteristics of Thin Films ....... E1
Paper F: Roughness Profiles - A way to recover heavy
   minerals from slimes .................................................. F1
Paper G: Influence of Electrokinetic Environment in
   Slime Gravity Concentration ....................................... G1

KEY-WORDS

Gravity Concentration, Sluicing, Spiral Concentration, Tabling, Heavy
minerals, Fine particles, Slimes, Viscosity, Roughness, Electrokinetic
environment, Dispersion, Aggregation
A Study of Gravity Concentration

INTRODUCTION

There has been no real awareness of the significance of gravity methods in dealing with slimes up to the present day. Any attempt to branch out in this direction may prove worthwhile and result in bringing out hidden potentialities of the methods. Although gravity methods have been successfully employed with heavy minerals of coarse and intermediate sizes there is a widely prevalent notion that it is not congenial for treating slimes. Some quarters even suggested abandoning gravity methods as a possible means of recovering heavy minerals from slimes (1, 2). But it should be noted that density differences yet exist between minerals in slimes. And, as can be seen from this thesis, it is more convenient to utilise this property than to selectively manipulate surface properties.

Available literature on slime gravity concentration is scarce and not coordinated properly. Therefore, they hardly provide any direction in which development can proceed. The main consideration in setting out to probe the problem in such a broad analysis is that the work should open up possibilities rather than develop a minuscule aspect. If such a line of development as the latter turns out to be futile it may perpetuate the prevalent myth regarding the scope of gravity methods.

Thus, there is an absolute necessity to critically study all the accessible literature to meet the questions arising from the need for development and the nature of the matter - whether gravity methods particular to intermediate particles such as sluicing and spiral concentration are applicable to slimes or the few methods exclusive to slime sized particles can be developed further. There is also the realisation that the exigencies of the problem may demand a departure from the above mentioned direction in the form of tapping hitherto unexploited resources of the methods. This may prove significant and highly valuable.

This thesis is an attempt to answer the questions posed in the above
discussion. The salient features of the main body of the thesis are delineated below.

In part 1 the literature is analysed. The areas covered are sluicing, spiral concentration, tabling and slime gravity concentration. The underlying mechanisms of the above unit processes, interrelationships among the involved variables, development and design of the relevant equipment and also their application are discussed. The behaviour of the particles in sluicing, spiral concentration and tabling are examined in detail on the basis of the fundamental knowledge available in mineral processing and fluid and particulate dynamics.

Analysis of the literature pertaining to theoretical aspects show that there is no universal theory available to describe or quantify any one gravity concentrator. However there are certain defined sorting mechanisms which can be related to the behaviour of individual concentrators (3,4,5,6,7,8,9,10,11,12,13).

In paper A it is implied that the mechanism of concentration in a sluice can be analysed by broadly categorizing the involved mechanisms into two separate, different actions; vertical stratification in the inhomogenous flowing suspension and the same in the moving dilated bed, beneath the former.

In paper B, it is implied that following Burch's (8) approach, along with computer simulations, should provide a wide scope in understanding and quantifying a spiral concentrator.

Paper C is on the principles of tabling. The thin-film flow, the asymmetric periodic motion of the deck and the action of the riffles are the significant characteristics of tabling. The mechanism of concentration of tabling was studied to a considerable extent by Taggart (5) and Gaudin (3). Such an analysis does not take into account the influence of the presence of particles in the fluid, the roughness of deck surface caused by recesses or the false bed formation of particles and the effects of the surrounding particles on the particle under consideration. Yet, despite this inadequacy, this is of fundamental value since it gives a mathematical picture of
the processes actually at work on the shaking table. It is implied in paper C that following the above way with the inclusion of the Bagnold theories as well as the theories on kinetics of the particle segregation should be an approach in describing and quantifying the segregation of the particles on a table.

In order to facilitate any such analysis a solution to the asymmetrical motion of the deck has to be obtained. At present the asymmetrical motion, because of its complexities, is approximated by a symmetrical motion (3, 11). A solution to a symmetrically reciprocating deck can also be found in paper C.

Paper D deals with the treatment of slime sized particles by gravity methods.

Firstly, in view of the widespread confusion in the mineral industry over the definition of the term "slime" and many other loosely used terms to indicate different size ranges of the particles, a classification is proposed with particular reference to gravity concentration which may suit the other areas as well.

The critical analysis of those few works (14, 15, 16, 17, 18, 19) that have been done on slime gravity concentration at a basic level, has in fact shown that much has to be done to improve the understanding and the effectiveness of slime gravity concentration. The influence of the apparent viscosity of pulps and the electrical double layer forces of concentrating surfaces were overestimated. Moreover the significance of the deck roughness was neglected.

The theoretical development of gravity concentration has not yet reached the stage at which the particle properties (ex: the particle size range for a given specific gravity and shape) necessary for the maximum performance of a given concentrator can be predicted. But the performance studies available on various gravity equipment can predict the necessary particle properties within the tested levels. An analysis of such performance studies on individual gravity equipment as well as circuits implies that the most suitable upper size limit for a slime circuit is 53-45 μm. Both sluice concentrators
and spiral concentrators are effective in recovering heavy minerals down to about 50 μm (ex: References 20, 21, 22, 23, 24)

Part 2 is an attempt to apply surface phenomena to slime gravity concentration and in fact it has proved to be wellworth the attempt.

The first example (paper E) refers to the dynamic flow characteristics of thin films. Viscosity is generally cited as a chief cause for any decrease in efficiency noticed in concentration processes, particularly with undeslimed feeds, despite the fact that there is little conclusive information available on the importance of viscosity. The body of literature that is available on viscosity as applied to gravity processes is therefore critically reviewed.

A mathematical analysis on the line of Gaudin's (3), of the dynamic flow characteristics of thin films is made as a function of distance from the surface of the deck, with the aid of experimental results. This convincingly shows that the viscosity effect due to the presence of ultrafine particles is negligible under the conditions prevailing in slime gravity processes.

The second example (paper F) refers to the influence of concentrating surfaces on slime gravity concentration. Surface roughness profiles of different concentrating surfaces are obtained using an equipment employing the "turn-table" technique. The roughness profiles obtained suggest the considerable influence it can have in slime treatment. Experiments were carried out to determine the effects of roughness profiles of varying dimensions on the treatment of slimes. These confirm the fact that concentrating surfaces with distinct roughness profiles can offer significantly better results than a strictly smooth surface, such as the stainless steel surface used. The influence of roughness is discussed. And concentrating surfaces with distinct roughness profile patterns are suggested in slime gravity concentration. Also the ability of the electrical double layer forces of a concentrating surface to have any significant influence on typical slime feeds is questioned.

The third example (paper G) refers to the attempt to improve the
effectiveness of a slime gravity process by the proper control of the electrokinetic environment. The need for such an improvement has been stressed by others as well. For example Burt and Stoelzle (25) firmly believed that this is one area giving scope for a major breakthrough in the increase of efficiency of gravity concentration and emphasized the need for further research work. Osborne (26) considered that a slime gravity concentrator (The Bartles Mozley separator) is highly suitable for the treatment of flocculated suspensions.

The present study (paper G) shows that the electrokinetic environment is an important factor in the treatment of natural slimes by gravity methods. Appropriate dispersion improves the performance by preventing heteroaggregation. This may be achieved in natural systems either by pH control or by the addition of a simple dispersing agent such as silicate. The addition of an anion specific to the heavy mineral to be concentrated should prevent the adsorption of silicate on to it. This will ensure a low electrokinetic potential of the heavy mineral. Dispersion alone may be sufficient at instances where there is only a little heavy mineral of value in the ultrafine range.

The addition of appropriate kinds and amounts of inorganic electrolytes to natural systems improves, but on a limited scale, the performance through selective aggregation. When considering the experimental conditions, stability diagrams, zeta potential studies, flotation test results (27) and flocculation test results (28) it appears that the inorganic polymolecular complexes play their roles in a manner similar to that of the organic polymolecular complexes. However it is readily conceded that further detailed basic studies on the specific roles of relevant inorganic ions and complexes are necessary for development.

The critical factor in the attempt to improve the performance of a slime gravity process by selective aggregation is the nature of the aggregates (whether it is compact or loose and open).
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My thanks are also due to those few persons who occasionally helped me to bring this thesis out in this form.
A Review of the Literature on Intermediate and Fine Particle Gravity Concentration

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Authors' note

Once again, after a lapse of about 50 years, there is widespread recognition of the importance of understanding and improving the treatment of particles by gravity concentration. An attempt is made here, therefore, to analyse the literature on intermediate and fine particle gravity concentration comprehensively and critically, in order to bring the current stage into perspective.

The areas covered are sluicing, spiral concentration, tabling and slime gravity concentration. The underlying mechanisms of the above unit processes, interrelationships among the involved variables, development and design of the relevant equipment and also their application, are discussed. The behaviour of the particles in sluicing, spiral concentration and tabling are examined in detail on the basis of the fundamental knowledge available in mineral processing and fluid and particulate dynamics.

Also, in view of the widespread confusion in the mineral industry over the definition of the term "slime" and many other loosely used terms to indicate different size ranges of the particles, a classification is proposed with particular reference to gravity concentration, which may suit the other areas as well.

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Principles of Sluicing

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Abstract

The theoretical background of the mechanism of concentration in the sluice is analysed in detail by broadly categorizing the involved mechanisms into two separate, different actions; vertical stratification in the inhomogeneous flowing suspension and the same in the moving dilated bed, beneath the former.

The significance of the many design and operational variables and their interrelationships are then examined together with the modelling of the cone variables. The cone's application is also discussed.

The design aspects of certain forms of sluices are analysed to show how theoretical and operational considerations should be knitted together to achieve an improved performance. Failure to obey such considerations is certain to make any sluice unpopular as is the case of Lamflo sluice.
INTRODUCTION

Sluicing is an operation where separation by gravity is attained by the settling of particles and the transportation of the non-cohesive ("loose") bed thus formed, in a slurry flowing through a trough which is essentially inclined and flat-bottomed.

Sluices of different forms have been used for the separations of denser minerals for centuries and it is perhaps the oldest technique of gravity concentration employed in mineral processing. The action of sluicing is observed in nature too - exposure of heavy minerals in mineralized areas after a heavy down-pour, heavy sands concentration on the beaches by tidal action and the removal of materials by river action, are the classical citations.

The earliest models of sluices were open channels cut along the ground or made of wood with rough textured floors or cross riffles fixed firmly to the floor. One type which has been widely used in the Oriental countries is the "Palong". It is about 300 m in length and 1.3 m in width and a typical installation uses several such launders in parallel. To the reader interested in knowing how the above and other earlier models are constructed and operated, the older text books and particularly Taggart's Handbook, which give excellent reviews, are cited. Though the earlier models have had extensive application, the gradual decline in the grade and the liberation sizes of the finely disseminated ores and also the need to treat low-density minerals by gravity methods, necessitated the development of the art of sluicing culminating in the fast pace witnessed over the last thirty years.

There are many different types of sluices now, from simple tray types to the finest form, the Reichert Cone Concentrator.

Fortunately, the interest shown in the development of the art of flowing film concentration has led to some theoretical enlightenment, though barely sufficient. The development of a universal theory which could explain the behaviour of a suspension through a sluice or any other gravity equipment is still not achieved. However, there are some
isolated defined sorting mechanisms which can be related to the behavior of individual devices (Mayer, 1964; Kelly and Spottiswood, 1982; Subasinghe, 1983). The other development on the theoretical side is the simulation models based on statistical analyses. Here, the equations can be purely empirical as seen in the works of Mineral Deposits Limited, Australia (Holland-Batt and Atfield, 1973; Holland-Batt and Terrill, 1975; Holland-Batt, 1978; Ferree and Terrill, 1978) or based on hydrodynamic relations with empirical constants as found in the work of Subasinghe (1983). The latter approach is preferable because of its physical significance.

There are considerable developments in the simulation of thickener-like gravity-separation vessels (Masliyah and Kwong, 1981), (here the theoretical model predicts the effects of the feed rate, pulp density and particle size distribution) and mechanical classifiers and hydrocyclones (Schubert and Neesse, 1973), (here the models predict the efficiency, cut size etc.).

The Russians have shown considerable interest in the kinetics of the gravity concentration process and the paper by Vinogradov et al. (1975) summarizes the work carried out in USSR on this aspect. In this work it is implied that second-order equations based on total energy balance can explain the segregation process in gravity concentration. This approach differs from the earlier first-order ones in that the accelerated nature of the segregation process is not neglected. More will be known of this work when translated versions are made available.

In the following sections, the theoretical background of the mechanism of concentration in the sluice is analyzed in detail and the significance of the many design and operational variables are examined together with the modelling of the cone variables. The cone's application is also discussed and the design aspects of certain widely differing modules of sluices are analyzed to show how theoretical and operational considerations should be knitted together to achieve an improved performance.
MECHANISMS OF CONCENTRATION IN A SLUICE CONCENTRATOR

A suspension of high pulp density is fed through a sluice at low flow velocity under conditions favouring the formation of a nearly non-cohesive bed of solids. The involved mechanisms which are relatively straightforward compared to other complex gravity processes of tabling or spiral concentration, can be broadly categorized into two separate different actions, vertical stratification in the inhomogenous flowing suspension and the same in the moving dilated bed, beneath the former.

Though the vertical stratification of the suspended particles can result in a certain degree of separation by specific gravity and size, it can be expected to be prominent in the immediate neighbourhood of the feed end only where the particles will settle down at a fast rate to form a loose bed. The degree of the effectiveness of separation in the suspension is governed mainly by the hindered settling nature of the particles (therefore pulp density) and the extent of the interstitial trickling at the end of the fall.

Bagnold (1946, 1954, 1956-1957) showed that there could be a dispersive force due to the particle movement even when they are not in actual physical contact, which he explained as due to the temporary displacements of particles in an overtaking layer giving rise to velocity fluctuations. The relationship found from the proposed model for the mean shear stress

\[ T = \eta (1+\lambda) \left(1+(f(\lambda))^2/2\right) \frac{du}{dy} \]

is:

\[ T \propto \lambda^{3/2} \eta \frac{du}{dy} \]  

(1)

where:

- \( \lambda \) is the linear grain concentration (=1/((C_{\text{max}}/C)^{1/3}-1))
- \( C \) is the volume concentration of grains
- \( du/dy \) is the (uniform) rate of shear of the grains
- \( \eta \) is the viscosity (Green et al., 1978)
- \( P \) is the Bagnold force (dispersive pressure).
Bagnold also explained how and why a dispersive pressure, $P$, should exist when particles form a loose bed in a flowing slurry. Under grain inertial conditions $T$ is found to vary as follows:

$$T \propto \rho(\lambda D)^2 (du/dy)^2$$  \hspace{1cm} (2)

where:

- $D$ is the diameter of the particle
- $\rho$ is the density of the particle and $T \propto P$.

Application of these relationships, though not rigorously due to the confined conditions under which Bagnold carried out the experiments to confirm the theory, to flowing film concentration, has been found greatly useful in accounting for certain phenomena. Bagnold himself successfully applied his theories to the behaviour of quartz sands in water and air (Bagnold, 1954).

The suspended nature of the particles in the flowing slurry is usually related to the turbulent eddies in fully or partially developed turbulent flows. If the vertical component of the mass rate of the displaced particles due to the eddy velocity is greater than the mass rate of the falling particles at the terminal velocity then the particles should remain in suspension. However, it is noticed that there is always a tendency for the particles to suppress the eddies and this can have considerable effect when the suspension is only partially turbulent (Abdinegoro and Partridge, 1979; Kelly and Spottiswood, 1982).

If a particle presents a face inclined at an angle to the direction of motion of the fluid it can be subject to an upward lift (aerofoil effect) and the effect can be expected to be prominent with flat particles. If the particle rotates in the fluid it will experience greater pressure at the lower edge due to the tendency to retard, because of the opposite directions of the movement of the fluid and the particle, than at the upper edge and this difference in pressure can exert a lifting force.

Robinow and Keller (1961) gave the magnitude of the lift force due to
spinning (with a rotating speed $\omega$) in a uniform unbounded fluid flow as:

$$L_\omega = \pi a^3 \rho_f (u_s - u) \omega$$  \hspace{1cm} (3)

In eqs. 3 and 4:
- $u_s$, $u$ are the particle and fluid velocity respectively
- $a$ is the radius of the particle
- $\eta$ is the viscosity
- $\nu$ is the kinematic viscosity
- $\rho_f$ is the fluid density
- $S = 2\omega$ for a freely rotating particle

These two kinds of lift forces will cease to operate if the above necessary conditions of an inclined face and spinning of the particles are absent.

Later in 1965, Saffman (1965) developed an expression for the lift force acting on a sphere, in a uniform, unbounded fluid flow with a linear velocity field, considering the shear-slip effect:

$$L_s = 6.46 \pi a^2 (S/\nu)^{1/2} (u_s - u)$$  \hspace{1cm} (4)

The above presentation of forces clearly show that the particles can remain in suspension even when eddies are absent and it may be said that different forces should operate in different parts under different conditions of the suspension. This also leads to the conclusion that irrespective of the nature of the slurry flow, there is always a high possibility of certain particles remaining in suspension leading to a heterogeneous suspended flow of stratified particles superimposed on a bed of particles, moving along the bottom of a trough.

In a sluice the converging side walls may lead to back-water rise which is prominent in the vicinity of the narrow, discharge end. Abdinerogo and Partridge (1979) saw experimentally that the velocity distribution of a flowing slurry through a pinched-sluice differs from that in parallel-sided channels in which the maximum flow-rate is found in the upper quarter of the stream. The maximum flow-rate was
found to be about half the depth in the pinched-sluice and, furthermore, it was found that the flow rates sharply decreased towards the free surface.

This difference is due to the exchange of velocity head for potential head as the stream thickens ("piles up"), causing turbulence. However, unfortunately, Abdinegoro and Partridge (1979) failed to notice the adverse effects that the greatly increased wall boundary layers can have, compared to the deck boundary layers, due to the very high depth/width ratio (=6) of the test sluice they used. Sluices found in the industry will normally have a depth/width ratio of about 0.3 only. Therefore, whether the same velocity profiles obtained by them would be found with the normal sluices, is also questionable.

The preferential lift of the coarse particles results in laminar flow (for example refer eqs. 3 and 4) while the turbulent flow is size-indiscriminative except when grain-inertial conditions prevail (refer eq. 2). It can also be shown that the shear process in a sluice is grain-inertial over a wide range of solids concentrations. However, in practice, it has been found that increased turbulent flow conditions in a sluice cause increased drops in the recovery of fine sizes (Abdinegoro and Partridge, 1979). Therefore, any desire to treat fine particles effectively by sluicing should also consider the suppression of the eddies. This should also explain why the Reichert cone concentrator is relatively more efficient in treating fine particles than the spiral concentrator of conventional design where a high degree of turbulence is prevalent.

The formulation of conservation equations for continuity and momentum of the flow of two-phase suspension remains an unsolved problem due to the complexities in the flow-field environment surrounding the particles. There are conservation equations, at present, based on the Boltzmann equation for the non-interacting particle suspensions which describe the transportation action of a two-phase suspension of sufficient dilution. However, obtaining conservation equations for non-dilute laminar and turbulent two-phase flows such as would prevail in a sluice, has not been achieved yet because of the difficulties in formulating governing conservation equations including the
interactions among particles and also the collisions between particles. In the case of turbulent flow, the particle interactions with the eddies in the surrounding fluid should also be taken into account.

Experimental studies on turbulent flows of dilute suspensions have in fact thrown some light onto some of the characteristics.

Two techniques that have been employed are the high-speed movie photography (Cumo et al., 1970, 1974) and the Laser Doppler Anemometry (Lee, 1982). While the photographic technique is capable of giving particle velocity and size number density but not fluid velocity, the Laser Doppler Anemometry technique is claimed to have the potential of measuring fluid velocity also. Worthiness of the applicability of these experimental techniques to denser and near or fully turbulent suspensions is highly questionable because of the loss of ability to discriminate, by the optical probing systems at higher particle concentration.

The second but more important effective separating action takes place in the transport of the bed which settles down near the feed point and forms a continuous bed with a velocity gradient in the vertical direction of the bed.

Halow (1973) in his attempt to study the mechanisms involved in initiating particle motion in a turbulent flow concluded that the force balances which include gravity, friction, lift and drag forces adequately explain the experimental observations of the fluid velocity required to initiate the motion of particles at rest. However, it should be noted that Halow confined his experiments to one single particle kept on the surface of the flow vessel which is in the viscous sublayer where turbulent velocity fluctuations are largely absent. Whether the same viscous sublayer environment can be expected to surround a particle on top of a loose moving bed with a constant tendency to dilate, must be studied. The drag and lift forces acting on such a particle can be expected to vary both in direction and magnitude by turbulent eddy velocities (Heywood, 1961; Fuchs, 1964; Concha and Almendra, 1979a, b, 1982). These forces can cause various
transitions in the type of motion of a particle: motion along and in contact with the surface of the bed (rolling or sliding) or motion along but not in contact with the surface (suspension) or no motion.

Naguib (Naguib, 1971; Naguib and Dyrenforth, 1972) attempted to apply Einstein's modified forms of equations of Manning and Duboy for flows in wide channels to obtain the weight rate of transport per unit width, which is an essential factor in deciding the capacity of the sluice, and also to predict the shear limits that could be permitted to ensure the removal of light material without being contaminated by the heavier product. Though this approach seems to be an impressive one, there are considerable areas which appear to have been taken for granted, demanding a detailed thorough revision of the approach.

The Bagnold forces and the theory of attaining minimal potential energy are the two fundamentally important deciding mechanisms in the vertical stratification of particles by size and specific gravity in a moving non-cohesive bed. The shear slip force may also be expected to operate here. However, a clear distinction between the kinds of forces that would operate at any particular environment of the bed may prove futile. Stratification, according to the potential theory tends to occur until the system attains the minimum potential energy state, providing the particles are free to move and yet, still maintain physical contact among the particles. Since the Mayer theory (Mayer, 1964) considers bulk densities, and not particle densities, particle size distribution and shape become significant. It can so happen that the heavier mineral with a lower bulk density than that of the lighter mineral can remain up in order to give a minimum potential state of energy. Furthermore, this theory also perhaps explains why the achievement of proper stratification is difficult when large amounts of middlings are present which can give the system a low total possible potential energy change (Kelly and Spottiswood, 1982).

**DESIGN AND OPERATIONAL VARIABLES AND THEIR INTERRELATIONSHIPS**

There is a considerable number of variables capable of affecting (or effecting) the performance of a sluice. The important variables are listed below:
(a) length of sluice prior to concentrate cut,
(b) width of sluice,
(c) degree of pinch,
(d) angle of slope and provision to vary
(e) arrangement of sluices to form a concentrate unit,
(f) feed rate,
(g) pulp density,
(h) tolerance to feed variations.

The degree of enrichment in a single stage greatly influences equipment selection for a given purpose. Improvement in the stage efficiency is achieved by better design and with optimum conditions of the operating variables towards which performance studies constituting diverse selections of test conditions play their role.

Abdinegoro and Partridge's study (1979) was a very useful one. The effects of different feed compositions at different rates and pulp densities on the stage efficiency of a pinched sluice were the objectives. It was shown that any size effect in vertical stratification could be almost be completely suppressed under hindered settling conditions. The fact that the detrimental effect of backwater rise at the vicinity of the discharge end also could be overcome by having a denser suspension, must be noted. Another significant effect that was shown is the ability to handle a feed of wide-size range with a denser pulp. Particles ranging from 79%, >180 μm to 33.6%, <90 μm were used in their experiments. For a perfect separation the concentration of light mineral at the lower depths should be as low as possible while that of heavy mineral should be as high as possible. Such a stratification was not seen for any size range or flow rates with low-density feeds. Furthermore, even at high concentrations, the high flow rates cause, due to turbulence, mixing - especially of the fine fraction denoting the preference for a low rate and coarse size fraction. It was also confirmed in their experiments by giving the best results at the lowest flow rate with the coarsest fraction.

The studies (Forssberg and Sandstroem, 1979a) made on the Reichert cone concentrator admirably confirmed the above-discussed inherent principles of sluicing. The best performance of the cone was seen at
high pulp densities only. Both grade and recovery were high. High percent solids in the pulp enhanced the hindered settling conditions which is the major deciding mechanism at the immediate neighbourhood of the feed end where the particles settle down at a faster rate. It is noteworthy that the effect of hindrance is felt more strongly on light particles than on heavy particles. Not surprisingly, the recovery in the finer fraction was substantially better at the high pulp density. As discussed in the section "Mechanisms of Concentration in a Sluice Concentrator" high percent solids in the pulp has suppressed the eddies in the pulp and made the flow near laminar which can cause only the light coarse particles to rise, unlike the turbulent flow which is size-indiscriminative. Higher feed rates caused a drop in the recovery due to the increase in the depth of the bed and also possibly due to the turbulent eddies. Which of these two causes would dominate would depend on the pulp density and the particle size.

Slope adjustment and slot width and (or) height adjustments are equipment based operating variables. A steeper slope and a narrower slot width will give low recovery but of high grade (Forssberg and Sandstroem, 1979a) and therefore it is best to have maximum slot widths in the rougher stage to achieve high recovery and to have progressively smaller widths in the cleaner and recleaner stages. A proper slope angle giving optimum conditions for each stage must be determined in conjunction with the maximum allowable slot widths in order to recover as high an amount as possible at the best grade by trying various slopes of the sluice. However, the slope is usually constant except with certain simple side-walled sluices.

Another equipment-based variable that was used in a sluice called Cudgen R.Z. multi-variable sluice unit (Pullar,1966), is the discharge end width. The width could be varied by shifting one of the side walls which is pivoted at the feed end. This design would have some adverse effects. The assymmetrical slopes of the side walls with respect to the longitudinal axis of the sluice could cause different flow patterns due to different wall effects at either side, leading to adverse transverse effects such as dislodging of the light materials to lower horizons of the moving bed. The plus point of the unit would
be the compact design.

Recovery of heavier fraction is in possible two ways in a sluice: pinching the slurry flowing from the discharge end at a suitable height by a splitter (Hobart, 1961) and drawing off a portion of the settled material moving in a bed on the trough by suitable slot arrangements. Examples for the first type are the old types, the Cannon circular concentrator, the Carpco fanning concentrator, etc. The undercut principle to draw off the concentrate has been used widely—the York star concentrator, Belmont multiple sluice concentrator, Cudgen M.V.S.U. and the Lamflo sluice. This second type can efficiently remove the heavy mineral portion of a bed of only a few particles thick when treating very low-grade ores such as the alluvial deposits of tin and gold.

The pinching assembly with the Reichert cone is actually a compound type. Variable inserts are available to give different annular widths. These annular rings are fitted to radial support arms which can be moved in ramped slots to be positioned at different elevations. Furthermore, with any one particular insert the effective gap varies because of the relative position of the curved downstream segment.

Another way of obtaining a concentrate is where the flowing stratified stream is allowed to impinge upon a flat plate positioned across the discharge end and split as concentrate and tailing streams. Though the manufacturers claim better performance with this separator than with the conventional types (Stratford, 1978), a thorough performance study on this otherwise apparently faulty unit to evaluate the influences of the adverse effects, (mainly the severe eddies caused by the impingement on the plate and the resultant kinetic energy loss) will reveal the worthiness of this kind of sluice.

A sluice gives only a very low degree of stratification in one single pass and therefore success of an operation by sluicing requires the inclusion of a sufficient number of cleaning, scavenging and roughing stages. The feed to each stage must also be directed by gravity wherever possible, otherwise these elaborate washes can turn out to be more expensive than the more efficient, but more costly processes such
as tabling. The first successful arrangement was found with the Belmont multiple sluice concentrator (Pullar, 1966) which consisted of 3 to 5 circular decks consisting of 16 sluices of about 1 m long, with 4 trays below directed radially outwards and downwards to retreat the product of each deck of sluices. The tailing from the deck was passed onto the next deck directly below by gravity. In addition, there were trays to re-retreat the products of trays directly above them.

DESIGN CONSIDERATIONS OF CERTAIN WIDELY DIFFERING MODULES OF SLUICES

The Cannon circular concentrator (Stewart, 1961)

This consists of 48 pinched sluices arranged in a circle with the discharge ends in the centre. Low flow velocities are obtained by passing the pulp into a feeding segment through feed pipes from a distributor, from which it is passed onto concentrating segments (Green et al., 1978). The discharged streams are cut by concentrate splitter rings positioned concentrically into three products. These splitter rings can be raised or lowered thus allowing for some control over the amount of concentrate removed. The slope is constant, just sufficient to enable the pulp to gravitate without banking.

The Carpco fanning concentrator (Stewart, 1961; Pullar, 1966)

Here the feed rises through the gradually widening central column until it is delivered into a pair of sluices set back to back. The discharge is cut by the splitter positioned like a fan. The idea of the fan is to spread the stream still further before they are cut by the splitters. Here the slope of the sluice can be adjusted by the turnscrews supporting the sluices from the feed column.

In both these modules the dilution water valves at the suction control the pulp.

The disadvantages of the fan concentrator are, the remixing of the stratified pulp when it collides with the fan plate which gives rise to a large recirculation of middling leading to high dilution water consumption; the absence of gravity flows and thereby pumping costs;
the tedious splitter-position adjustment on each sluice.

The Lamflo sluice

The Lamflo sluice is essentially a combination of three ordinary pinched sluices but with converging and diverging side walls for the first and third sluices.

The converging sides in fact cause certain effects that are counteractive. On the one hand, it exerts a velocity vector on the relatively slower particles at the sides, directed towards the central part of the sluice, but on the other hand, the velocity of the stream at the wall sides is further reduced, giving rise to the question "what is the resultant effect?". In addition, the directional changes of the stream cause a higher degree of turbulence which is claimed by the manufacturers as a contributory factor towards the unlocking of the locked light particles in the bed (Naguib, 1971; Naguib and Dyrenforth, 1972).

It is also noted by the manufacturers that care should be taken to ensure that the above turbulent eddies do not result in lifting the heavy particles as well. It is further noted that it could be ensured by considering the specific weights, the hydraulic size range of the particles, the abundance and shape and the extent of the heavy medium nature of the fluid. However, it appears that there is a serious handicap in trying to give this solution. Any consideration towards the particle size range, distribution and shape will require stringent classification, making the Lamflo sluice unpopular. Very high pulp density is needed for efficient separation in a sluice. If the medium is sized, a necessity above, the heavy medium nature of the fluid cannot be significant, considering the usual coarse size range of the feed for a sluice.

The diverging walls give rise to the reversal and direction of the velocity vectors. This can also result in liberating the locked particles and, in addition, dropping some of the heavy particles which still remain in suspension due to the loss in carrier velocity.
However, the important factor here is how far these desirable effects are reflected on the ultimate separation of heavier minerals from lighter minerals. Shallower depth of bed due to the increased width, it is said, leads to a higher proportion of the heavies being recovered at the horizontal cutters. Now, it is shown that the efficient separation of the particles by specific gravity and size, takes place in the non-cohesive bed by relating the behaviour to the defined sorting mechanisms, the attainment of minimum potential energy and the Bagnold forces mainly (see pages 3, 4 and 8). These theories require considerable dilation and reduction of interlocking and also high rates of shear, which make a non-cohesive bed of a considerable number of layers (high total possible potential energy change and high rates of shear) a prerequisite (Bagnold, 1954; Bagnold, 1956-1957; Mayer, 1964).

This rational line of justification perhaps convinces the reader that the curved converging and diverging walls can be detrimental. Guest (1980) also shares this negative view on the Lamflo sluice in his report on the survey of the literature on gravity separation, citing the workers.

The Reichert cone concentrator

The long search for a high-capacity concentrator with the ability to produce a product of suitable grade and tailing with a low amount of circulating load in a single unit, resulted in the development of the Reichert cone concentrator which is really a set of sluices arranged in a circle with the discharge ends pointing downwards and towards the axis of the circle and their side walls removed.

The unit has been well described by many (Ferree, 1972, 1973; Giffard, 1972; Graves, 1973; Terrill and Villar, 1975; Ferree and Terrill, 1978). A good description of the earliest model is given by Pullar (1966).

The main advantages of the cone over other types of sluices include the absence of side walls, high unit capacity with little floor space requirement, versatility: cones are used as a preconcentrator, as a
scavenger or as a complete plant with roughing, scavenging and cleaning stages (Buist et al., 1973; White, 1978; Luckett, 1978; Chong, 1978; Forssberg and Sandstroem, 1979a, b; Robinson, 1981, 1982; Robinson and Ferree, 1983), availability of a number of (thirty-three, Burt and Mills, 1982) separate configurations, ability to handle high-grade feeds with greater facility, ability to handle feeds of widely different grades due to the availability of a variety of combinations of slot setting (Holland-Batt, 1978) and also the ability to change the arrangement of a number of standard components - pinched sluices and cone elements - and finally the low operating costs due to reduced manpower requirement and power consumption, low maintenance and the robust design with long-wearing parts.

Graves (1972) first suggested the economical feasibility of scavenging the flotation tailing. One example is Renison Limited, Tasmania (Perkins, 1977; White, 1978), where cones are extensively used to scavenge the flotation tailing. Another area of cone usage is in the grinding circuits (Ferree and Terrill, 1978; Forssberg and Nordquist, 1984).

Frequent blocking of annular slots with oversize material and mill rubbish is a disadvantage. Other minus points are the largeness of the unit size and relatively high cost. These two disadvantages cannot be considered as disadvantages in the exact sense of the word, for plants with low capacities, the standard spiral circuits with almost the same efficiency in the same size range with less capital cost (however, complex pulp distribution system by pumping with a spiral circuit may be considered) along with a shaking table to take care of the coarse light particles, can be the alternative (Robinson and Ferree, 1983).

Cone feed with a wide size range can give a preconcentrate with some amount of both too coarse and too fine light particles because of concentrating characteristics of the cone. The more efficient way is to use the complimentary characteristics of a jig and a spiral concentrator (Robinson and Ferree, 1983)

Sensitivity to feed variations is another drawback.
MODELLING OF Cone VARIABLES

The continuous revision of the cone is summarized below so as to show how complicated the design has evolved into from a simple, rather straightforward state:

(a) The original cone consisted of one or two single cones with cylindrical movable cutters to pinch the slurry and the concentrates were cleaned on trays.

(b) Subsequent replacement of adjustable cutters by fixed gunmetal throat with two slots within a reasonable distance or one slot only.

(c) Development of double cone units with finger-splitters fitted at the outer periphery of the distributing cone to evenly split the feed between the two inverted cones which has given rise to a number of distinct separate cone configurations.

(d) Development of different types of slots and slot positioning arrangements - fixed inserts with double or single slots, variable inserts with nine elevations and the deflecting apparatus (Phillip, 1979) with means to divide two streams into four streams of different concentration and means to combine streams of intermediate concentration.

When considering the large number of variables of a cone concentrator it becomes apparent that the conventional types of investigations based on input-output calculations for graphical representation would be more than inadequate with regard to the desire for optimum performance of either a large plant or a confusing difficult circuit.

There is no exact mathematical theory based on exact analysis to quantify a flowing concentrator. Nevertheless, mathematical models can be developed by statistical investigation.

The work on such models for predictive purposes has been extensively carried out by the Mineral Deposits Limited, Australia (Holland-Batt and Atfield, 1973; Holland-Batt and Terrill, 1975; Holland-Batt, 1978; Ferree and Terrill, 1978). The statistical approach by computer simulation towards the effects of different operating variables on the effectiveness of separation can be broadly summarized as follows:

(1) Identification of dependent and independent variables. Usually the
dependent variables are recovery and grade or collection efficiency and selection efficiency.

(2) Design of an appropriate sequence of tests covering all the possible extremes of the conditions envisaged. Though a full factorial sequence is the ideal, usually the stages selected are partial factorial only because of the requirement of the large number of tests for the full factorial sequence which could be practically impossible. If necessary, in the case of large-plant simulation, more than one test series with different grades to get "response surfaces" for grade and recovery in the final concentrate as functions of feed grade and feed rate. A proper test rig with sampling facilities at different stages is needed for the test. A DSVSV test rig with a special concentric pipe assembly has been used by the MDL and the Mineral Processing Division, University of Luleå (Forssberg and Sandstroem, 1979a, b). A tray assembly can also be used.

(3) Construction of a regression model by fitting the test data. Usually a linear relationship between the variables involved, either first order or second order with or without extra compound variables of the general form $x^{P}y^{Q}$, where p and q can take either integer or fractional values, is assumed. The validity, the precision of fit, is tested by calculating the correlation coefficient and residual standard deviation of the regression equation.

(4) Evaluation of possible cone configurations for the necessary stages by optimizing each cone unit step by step.

Some possible disadvantages of this technique are, accumulation of errors from the numerical calculations on the data, errors due to operational defects and sampling, application of closed-circuit results to open circuits and not necessarily, scale effects.

On the standard of such models, it can be said that models which include only a few variables of the many possible variables should be used for identifying the significances of those variables only. For predictive purposes, extensive studies incorporating all the possible variables are necessary.
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Principles of Spiral Concentration

Accepted for publication in the
Abstract

The different stages of the mechanism of concentration in spiral concentrators are discussed. The significance of many design and operational variables and their interrelationships are examined. The various areas where the spiral concentrators are applicable are presented.
MECHANISMS OF CONCENTRATION IN A SPIRAL CONCENTRATOR

The spiral concentrator consists essentially of an oblique helical conduit of a number of turns.

As in any flowing film concentration process, vertical stratification of the flowing film down the spiral conduit can be related to the following defined sorting processes - the hindered settling, the interstitial trickling, the attainment of minimum potential energy and the Bagnold forces. The hindered settling conditions intensify the specific gravity effect in settling. While the dilation of the bed due to the forward and sideward travels of the bed of particles, favours the system's struggle to attain a minimal potential energy by forcing the heavies downward, providing the particle size distribution and also the shape of the heavy mineral particles are favourable, small particles in the system tend to trickle through the available interstices. The Bagnold dispersive force preferentially lifts the coarse, light particles into the high velocity layers of the flowing pulp which increases from zero at solid-liquid surface to maximum just at the free surface, providing appreciable turbulence is not created in the pulp which is achieved by keeping the pulp to spiral concentrator dense and steady.

The understanding of the lifting power of the Bagnold dispersive force may be obtained from a factor $N$, proposed by Bagnold (1954):

$$N = \frac{\lambda^{1/2} \rho D^2 \frac{du}{dy}}{\eta} \quad \text{for } \lambda < 14$$

where:
- $\eta$ is the viscosity of the liquid
- $\frac{du}{dy}$ is the rate of shear
- $\rho$ is the particle density
- $D$ is the particle diameter
- $\lambda$ is the linear concentration ($=1/(C_{max}/C)^{1/3} - 1$)
- $C$ is the volume concentration of solids
Depending on whether \( N > 450 \) or \( N < 40 \) "grain-inertial" or "viscous" conditions, as defined by Bagnold, exist.

In the "grain-inertial" region the Bagnold dispersive pressure \( P \) on a particle is:

\[
P = 0.04 \rho (\lambda D)^2 (du/dy)
\]

The effect of this dispersive pressure \( P \) is to counteract the apparent weight \( P_a \) of a particle:

\[
P_a = \frac{\pi D^3 (\rho - \rho_f)/6}{\pi D^2/4} = \frac{2}{3} (\rho - \rho_f)
\]

Therefore:

\[
P/P_a = \frac{kD(du/dy)^2}{1-(\rho_f/\rho)} \quad (k \text{ is a constant})
\]

If two particles of diameters \( d_1 \) and \( d_2 \) \((d_2 > d_1)\) are considered:

\[
(P/P_a)_{d_1} < (P/P_a)_{d_2}
\]

Therefore, \( d_2 \) will stay up in the fast-flowing layers of the film.

Similarly, if two particles of density \( \rho_h \) and \( \rho_1 \) \((\rho_h > \rho_1)\) are considered

\[
(P/P_a)_{\rho_h} < (P/P_a)_{\rho_1}
\]

Therefore, \( \rho_1 \) will stay up in the fast-flowing layers of the film.

Under the "viscous" conditions the dispersive pressure \( P \) on a particle is given as,

\[
P = 2.93 \lambda^{3/2} \eta \frac{du}{dy}
\]

Following the same line of argument found above, it can be seen that only small, light particles will stay up in the fast-flowing layers of the film in contrast to the Bagnold dispersive force's preferential lift of the coarse, light particles in the "grain-inertial" region.
Burch (1956) however, argued that it is unlikely that grain-inertial conditions would exist due to the shear produced by the flowing film only, when treating relatively fine particles, in troughs with rather flat profiles. A substantial pressure will be produced if an orbital or a rotational motion is given. Burch's shaken helicoid employed an oscillatory motion (Burch, 1957). Recently, the Chinese developed a revolving spiral concentrator (Robinson, 1983).

In addition to the above-mentioned defined sorting processes' effects in vertical stratification, there exists the influence of rising and falling currents near the inner radius and the outer radius respectively (explained below), unique to the spiral concentrator, which will move the smaller particles preferentially in contrast to the Bagnold force's preferential lift of the coarser particles.

The combined action of the above-mentioned sorting processes will cause the heavy particles to settle at the bottom with the lights above and thus the particles are caught in the different velocity layers travelling in a curved path. They will be acted upon by centrifugal forces of different magnitude which will tend to shift the particles towards the outer edge of the conduit. However, particles in the lower layers will not be able to migrate to the outer edge due to low centrifugal forces and the radial velocity of the lower part of the stream, which are associated with the inward slope of the conduit. The result is that the bottom layers will force the particles (heavy) towards the inner side of the spiral concentrator where openings may (or may not) be located, depending on the design.

Traditional spirals are meant to remove the concentrate at the first or second turn and to treat the middlings subsequently in the turns below, with concentrate ports at 180° intervals. Some of the ports are normally closed, to obtain a high ratio of concentration.

Positions of the concentrate ports vary according to the requirements. Closely located ports can be more selective in obtaining a concentrate when treating particles of near specific gravity or particles containing a high proportion of high specific gravity minerals. The modern spiral concentrator developed for feeds which contain only a
few percent of heavy minerals have no such concentrate ports. Wash water addition is also not preferred when treating fine particles particularly, to prevent any detrimental developments of turbulence (Wells and Elliot, 1982). The addition of wash water at the inner edge of the stream tends to force the fine particles back into the middle portion of the conduit where it is subject to further cycles of concentration. If added, care should be taken to ensure that the wash water flows steadily and is free of foreign matter.

As described above, while the particles in the lower layers of the stream are carried inwards due to radial inward velocity, there can occur an azimuthal separation of the particles based on the sizes. Since the coarse heavy particles in the inwardly moving layers are acted upon by large forward velocities they will go a greater azimuthal angle than the fine heavy particles.

The inward flow of the bottom layers and the outward flow of the upper layers are connected by rising currents on the inner radius and falling currents on the outer radius and the resulting helical boundary between the upward and downward moving layers will be determined by the concentrator design. The rising currents are significant in that they provide a unique phenomenon by lifting the small particles upwards which, if proportionated with the Bagnold effect, means particles of different sizes will be lifted to approximately the same extent which would alleviate any undesirable size separation.

Burch (1961-1962) proposed mathematical relationships to quantify the several flows involved in the sorting and the separating of particles in a pulp that flows down the conduit of a spiral concentrator. However, they may not be rigorously applicable, because they were based on quite a number of assumptions.

His relationships to obtain an approximation of the secondary circulation are as follows:
radial inward velocity $u$ at fractional height $yH$:

$$u = \frac{\sqrt{V_H^2 y(7y^5 - 42y^4 + 70y^3 - 70y + 32)}}{ry \cos \theta}$$

primary velocity $V$ at $yH$:

$$V = \frac{y(2-y)H^2 \rho g}{4\pi\gamma}$$

inflow rate per turn at radius $r$, within which it must rise:

$$1.376 \frac{P^2 H^2 g^2}{1680 \pi \gamma r^2}$$

rising velocity at $r$:

$$1.376 \frac{P^2 g d}{3360 \pi \gamma r}$$

height of the slope of the free surface $Z$:

$$\frac{2}{3} \frac{12/7}{2/7} \frac{3g \rho A}{r} = \frac{2}{14 \gamma}$$

height of the pulp layer $H$:

$$\frac{3/7}{4/7} \frac{A}{r}$$

Here $P$ is the pitch, $\theta$ is the angle of inward of the deck, $\gamma$ is the kinematic viscosity, $H$ is the height of pulp at radius $r$ and $A$ is a constant having the dimensions of length.

Additional basic work, along with computer simulations, on the footsteps of Burch should provide more valuable qualitative as well as quantitative information. This is an area which gives ample scope for a major breakthrough in spiral concentration.
DESIGN AND OPERATIONAL VARIABLES AND THEIR INTERRELATIONSHIPS

The pitch, profile and radius are the three design variables of a spiral concentrator and there is available a variety of models with varying dimensions enabling a correct choice from a wide spectrum for particular applications.

The velocity of the pulp, fed to the top of a spiral concentrator is determined by the pitch of the helix. Deep angles provide high capacities and high grades, but low recovery. They must perform well with the ores of very low grades. Shallow angles are suitable for operations involving small specific gravity differences and fine particle sizes. With the advent of moulded (in one piece), stretchable fibre-glass-reinforced spiral concentrators, tests became possible at different pitches. This led to present time's several pitch dimensions and also spirals with a varying pitch down the axis (Anonymous, 1979; Balderson, 1982).

Another advantage of the steep pitch is that it enables, with a reduced height of outer wall, a double-wound or a triple-wound unit on the same central column to be possible which saves space and cuts down on the launder requirements. A spiral having a continuously curved profile with a variable radius of curvature but without concentrate ports can also give a double-wound arrangement of troughs.

A properly designed trough profile prevents the pulp just running on. Profile patterns are many. A continuously curved profile as used in the conventional spiral concentrators is suitable for general applications as encountered in mineral sands treatment. With the intention of alleviating the problems of adjusting the concentrate splitters for a variable feed, a spiral concentrator with a few channels moulded side by side with fixed splitters, to direct the concentrate from the outer channel to inner channels successively, was developed (Anonymous, 1965). However it was found to suffer from the problem of inner channels becoming overloaded. A modified form of
this is the compound spiral concentrator with the inner section having less acute slope than the outer section with the point of intersection moving radially outwards from the centre column, from the top to the bottom of the trough. This is suitable for feeds containing a very low percent of heavy mineral since the design enables tailings and middlings to be treated on separate trough slopes. A flat-bottomed spiral concentrator with a reduced pitch should be able to perform well with very fine particles because of the low velocities that would prevail there. A continuously curved profile with a very long but variable radius of curvature is suitable for reducing ash contents in the washing of coal (Balderson, 1982).

Despite the work of Sukhonova et al. (1972) the radius of the spiral seems quite arbitrary except for the fact that the larger the diameter is the finer the material it will treat depending on the feed rate, trough profile and pitch. A spiral with an increasing radius should be able to treat particles with a very wide size range (2 mm -50 μm).

The nature of the mode of operation, whether it is roughing, cleaning or scavenging, governs the required number of turns (usually 3-10) for which a spiral is employed. The more difficult an operation is, the more the turns required. Burch (1961-1962) obtained the following relationship to calculate the number of turns essential for particles in the lowest layers to get from the edge to the concentrate region, for his shaken helicoid:

\[ \theta = \frac{9P(r_2 - r_1) \, dr}{4\pi r_1 r_2 \, dz} \]

where \( \theta \) is the azimuthal angle, \( dZ/dr \) is the deck surface inward slope (considered constant), \( P \) is the pitch and \( r_2 \) and \( r_1 \) are the outer and inner radii of the trough under consideration.

Variations in pulp characteristics - the feed grade, percent solids, flow rate and particle size distribution - affect the effectiveness, as in any concentration process, of a spiral concentrator leading to losses. However, while any limited long-term variations in the feed characteristics may be controlled by adjusting the concentrate
splitters and amount and direction of wash water, in the case of conventional spiral concentrators, any short term variations (within limits) are not too significant since a spiral circuit is always operated in multi-stages with middling recirculation (Alexis, 1978). An interesting study by Dallire et al. (1978) throws considerable light on a conventional spiral concentrator's (with ports and wash water) tolerance to feed variations.

Within the range of the tests maximum recovery was obtained at low flow rates and high percent solids. With increasing pulp feed rates, recovery was found to drop while the grade improved (Fig. 1). High centrifugal forces associated with the high pulp feed rate should have kept the middlings and fine particles away from the ports. High percent solids in the flow improved the grade by intensifying the hindered settling conditions (Fig. 1).

An intermediate but comparatively wide size range (< 20 mesh) was found to give good recovery (Fig. 2), because coarse heavy particles prevented the fine heavy particles being carried upward by the rising currents at the inner region of the spiral concentrator. Very fine particles which should find it difficult to settle through the fast flowing pulp would lead to very low recovery. Grade was found to improve, as would be expected, with a narrowing size range (Fig. 2).

Feed grade was found to be an important factor which attracts attention. Heavy loading required a careful control of feed grade, for surplus heavy mineral could not find its way to ports which decreased recovery (Fig. 3).

APPLICATION

Coal

Spiral concentrators have been extensively tested for the reduction of ash and sulphur from the fine coals. The best size range to obtain maximum ash reduction with minimum loss of coal, proved by a series of tests on coals with different washability characteristics (Geer et al., 1950), would be a classified feed in which the clean coal is
about minus 8-mesh and the accompanying impurity is about 28-100 mesh size. Otherwise, the coarse heavy particles settled farther in the stream would not reach the refuse ports. Heavy but very fine particles would also report to the clean coal side. The coal particles coarser than about 8-mesh in size can be easily treated by HMS and jigging. The most sensible way of treating such fine coals would be to pass the deslimed coal over a spiral to obtain a concentrate and then to classify and retreat the combined refuse-middling product on another spiral.

Another way of fine coal treatment is removing the (about) plus 6-mesh and minus 200-mesh particles before treating it on a spiral concentrator and then combining the plus 6-mesh product with the spiral concentrate. It is possible when the ash and sulphur are concentrated in the finer sizes and also the plus 6-mesh size contains an appreciable quantity of clean coal. In such an attempt on a washer waste 94% of the coal was recovered with an ash reduction to 5% from 24% (Browning, 1977). The recently developed portless, washwaterless Mark IX and Mark X spiral concentrators are claimed to perform well with coals of 6x200 mesh sizes at high capacities (Balderson, 1982; Mineral deposits, 1983).

The spiral concentrators are considered for reclamation of pyritic sulphur also, to eliminate stream pollution by acid water drainage and other environmental hazards (Browning, 1982).

Heavy minerals and sands

Spiral concentrators have been extensively used in the beach deposits such as those containing ilmenite, rutile, zircon and monozite where they have the role of producing a preconcentrate as well as a final saleable product. The low floor area requirement of the spiral concentrators make them a very good preconcentrator in offshore treatment. Other areas of application are barytes, chromite ores, iron ore fines, kyanite, Pb/Zn oxide and sulphide ores, tantalum/niobium ores, tin ores, tungsten ores and gold. Applications of the spiral concentrators including the modern spiral concentrators to heavy minerals and sands are sufficiently documented in the literature.
(Humphreys and Hubard, 1945; Carpenter, 1953; Robinson, 1973; Robinson and Ferree, 1983; Anonymous, 1981; Balderson, 1982; Wells and Elliot, 1982; Spiller, 1983).
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Recovery %

![Graph showing performance at different % solids and feed rates](image)

- 37.85 lpm
- 56.78 lpm
- 75.70 lpm
- 90.84 lpm

Fig. 1 Performance at different % solids and feed rates (after Dallire et al, 1978).
Fig. 2 Grade and recovery vs particle size at 75.7 lpm
(after Dallire et al, 1978).
Fig. 3 Grade and recovery vs feed grade at 75.7 lpm (after Dallire et al, 1978).
Principles of Tabling

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Abstract

The mechanism of concentration on the shaking table is discussed and a fresh attempt is made to obtain the theoretical model describing the velocity of a fluid in depth and time when in contact with a symmetrically reciprocating surface.

The significance of the many design and operational variables and their interrelationships are examined. The various areas where the shaking table plays its role are then considered.

The relevant features of micropanner, superpanner and the subsequent macropanner and pulsepanner are also treated since they all, like the shaking table, have the differential motion in common.
INTRODUCTION

The shaking table, which is relatively old and gradually unfolded from the earliest thin-film concentrators such as the early shaking vanners, the Evans multiple-deck, revolving round tables and the bumping tables, consists of a fully or partially riffled plain and near rectangular or rhombohedral surface shaken with a differential motion in the direction of riffles while the wash water is supplied down the slight slope across the riffles.

The first shaking table to use a differential motion mechanism instead of the bumping block was developed by Arthur Wilfley in the period of 1895 to 1896. The bumping table is obsolete now because of its mechanical limitations. Since Wilfley there had been many types of shaking tables but only a few continue to exist. Heavy mineral, heavy mineral sands and coal industries exclusively used shaking tables. They were used as primary concentrators as well. However, the gradual depletion of the grade and the increasing demand demanded changes in the form of high capacity, less costly and floor-saving equipment to handle the primary feed.

As a result, the spirals and later the sluices replaced the shaking tables as primary concentrators. Though the role of the shaking table is comparatively small now, its role is important because of its efficiency and reliability in giving a high-grade product. Good descriptions of the known types of shaking tables - Garfield, Butchart, Deister-Overstorm, Wilfley, Card and Plat-O - are found in the Handbook of Taggart. The main differences between these tables are drive mechanism, shape of deck and riffle pattern.

Recent models should include the multiple-deck tables and the shaking tables known as "slime-tables" such as Holman and Plat-O slime tables. Though the equipment of Bartles and Mozley fall under the group of "shaking concentrators" these are dealt with separately because of the many different basic principles and mechanisms and also of the unique recognition they possess in the mineral industry.

A considerable amount of literature is available on shaking tables.
However, most of them deal with performance studies or descriptions of the equipment only.

A few works concern themselves with the underlying principles of the shaking table.

Below, the mechanisms of concentration on the shaking table, its design and operating variables and also its application in the mineral and coal industries are considered. Also, relevant features of micropanner, superpanner and the subsequent macropanner and pulsepanner are treated here since they all have the differential motion in common.

MECHANISMS OF CONCENTRATION ON A SHAKING TABLE

The thin film concentration process which removes the particles of a bed from top to bottom progressively, the asymmetric periodic motion of the deck which stratifies the particles according to density, size and shape in the longitudinal direction of the deck and the action of the riffles to assist the stratification on the vertical plane by rendering the bottom portion of the thin-film flow more turbulent and also to aid the transport of the settled heavy particles towards the heavy product end are the significant characteristics of the shaking tables.

A thin-film flow is of a few particles thick and has low rates of shear. The thin-film flow was studied to a considerable extent by Taggart (1951) and Gaudin (1939), and Christ Mills (1978) has reviewed their work well. Here, only their conclusions are discussed.

It can be shown that the velocity $V_f$ at any depth, $y$, of a thin film flowing under laminar conditions is as follows:

$$V_f = (\sigma' g \sin \alpha) \frac{(2x - y) y}{2\eta}$$  \hspace{1cm} (1)

where: $\sigma'$ is the fluid density
$x$ is the film thickness
$\eta$ is the fluid dynamic viscosity
\( \alpha \) is the slope of the deck

Though the above equation and the following equations derived from it do not take into account the influence of the presence of particles in the liquid, the roughness of the deck surface caused by the recesses or the false-bed formation of particles and the effects of surrounding particles on the particle under consideration, this kind of analysis is of fundamental value since it gives a mathematical, though inadequate, picture of the processes actually at work on the shaking table. Another weak point in applying Gaudin's analysis rigorously to the shaking table is that Gaudin's analysis pertains only to plain surface whereas most concentration takes place on the riffled part of the deck. While this is significant in the concentration process across the riffles, this point, however, does not arise when stratification in the longitudinal direction is considered. Regarding the concentration process across the riffles the author's view is that thin-film concentration operates at the top layers only, for which the interference of the riffles may not be taken into account.

From eq. (1) above, the thin-film thickness for a given slope and rate of flow and vise-versa can also be obtained:

\[
\text{volume of the fluid } W = \sigma' g \sin \alpha \frac{x^3}{3\eta} \quad (2)
\]

\[
\text{the film thickness } x = 3 \sqrt[3]{\eta W / \sigma' g \sin \alpha} \quad (3)
\]

The rate of acceleration can be obtained for a particle resting on a smooth inclined surface, applying Newton's Second Law of Motion using a shape-factor coefficient \( k \):

\[
\frac{dv}{dt} = \frac{\sigma_s - \sigma'}{\sigma_s} g(\sin \alpha - \mu \cos \alpha) + \frac{9\eta k}{2\sigma_s r^2} + k \frac{\sigma' \sin \alpha g(x/r)}{2\sigma_s} - \frac{27k\sigma' g \sin \alpha}{8\sigma_s} \quad (4)
\]
where: \( \sigma_s \) is the particle density
\( \mu \) is the coefficient of friction
\( v \) is the particle velocity
\( r \) is the particle radius

It follows from the above that the minimum angle below which motion by sliding cannot occur is as follows:

\[
\text{cota(crit.)} = \left( \frac{-9k\sigma' x}{2(\sigma_s - \sigma')r} - \frac{27k \sigma'}{8(\sigma_s - \sigma')} \right) / \mu_s
\] (5)

This equation is useful in the sense it gives the critical angle for sliding motion for various minerals of various sizes.

\[
V_m(\text{sliding}) = \left( \frac{r^2}{\eta} g \sin\alpha \left( \frac{2(\sigma_s - \sigma')(1 - \mu_D \text{cota})}{9k} - \frac{3}{4} \sigma' \right) + \frac{rg\sigma' x}{\eta} \sin\alpha \right)
\] (6)

is the terminal velocity of a particle in motion, obtained from the equation (4) assuming that the \( \mu_D \), the dynamic coefficient of friction, remains constant.

When considering the motion by rolling, the shape plays the major role. Usually the triangular and square shaped particles will slide, the spherical or near spherical particles will role and the particles of intermediate shape will behave in an intermediate manner, under the existing conditions of shaking concentrators. For the rolling motion, the equation obtained is:

\[
V_m(\text{rolling}) = \left( \frac{2}{\eta} g \sin\alpha \left( \frac{2(\sigma_s - \sigma')}{9k} - \frac{3}{4} \sigma' \right) + \frac{rg\sigma' x}{\eta} \sin\alpha \right)
\] (7)

Equations (6) and (7) give a lot of information regarding the effects of slope, density, shape factor coefficient, friction coefficient and size on the particles' velocity. In fact, curves of relative particle velocity are computable for systems containing particles of different
specific gravities and sizes which should of assistance in determining the approximate velocities with which they travel down the slope under varying conditions. Such calculations can give the approximate slope and size range for the separation of minerals.

Next the asymmetric periodic motion of the deck and the resultant different velocities of the layers of fluid upon it shall be considered. Again, by applying Newton's Law for the motion of particles due to the asymmetrical motion of the deck the forces, the acceleration force $m(dv/dt)$, the frictional force and the fluid resistance to the motion, acting on them can be balanced for the limiting condition as follows:

$$\frac{dv}{dt} = \frac{(\sigma_s - \sigma')}{\sigma'} g u_s + \frac{F_D}{m}$$

(8)

where: $F_D$ is the fluid resistance

$m$ is the mass of the particles

For this equation to be solved, fluid resistance in terms of time and distance from the deck needs to be known. Fluid resistance could be obtained if the fluid velocity with respect to time and distance is known.

The general equation for the fluid velocity in terms of time and distance is:

$$\delta v/\delta t = (n/\sigma') (\delta^2 v/\delta x^2)$$

(9)

It is apparent that the above equation cannot be solved unless the equation of the asymmetric periodic motion is known. However, assuming a symmetrical periodic motion, an equation for the fluid velocity in terms of time and depth can be obtained. The final form is as follows:

$$v_x = V_o \exp\left(-\frac{y}{\sqrt{m\sigma'}/n}\right) \cos(2\pi ft - \frac{y}{\sqrt{m\sigma'}/n})$$

(10)

where:

$V_o$ is the velocity of the deck at time $t$ and frequency $f$

$v_x$ is the velocity of the fluid layer at height $y$
Now the proof of the above equation (10) is given:

\[
\frac{\partial v}{\partial t} = \frac{n}{\sigma'} \frac{\partial^2 v}{\partial x^2}
\]

\[= k \frac{\partial^2 v}{\partial x^2} \quad (n = k \text{ (say)})\]

Let us say:

\[v = v^0, v^*, t^*, x^* \quad \text{variables}\]
\[t = t^0, t^*\]
\[x = x^0, x^* \quad \text{constants}\]

from the above it follows:

\[\frac{\partial^2 v^*}{\partial \theta^2} = -\frac{\theta}{2} \frac{\partial v^*}{\partial \theta} \quad \text{where} \quad \theta = \frac{x^*}{\sqrt{t^*}} = \frac{x}{\sqrt{kt}}\]

\[
\int \frac{d(\partial v^*/\partial \theta)}{(\partial v^*/\partial \theta)} = \int -\frac{\theta}{2} \, d\theta
\]

\[\ln \left(\frac{\partial v^*}{\partial \theta}\right) = -\frac{\theta^2}{4} + \text{const.}\]

\[\frac{\partial v^*}{\partial \theta} = e^{-\theta^2/4} + \text{const.} = A' e^{-\theta^2/4}\]

\[
\int \frac{v/v^0}{dv^*} = \int_{\theta=0}^\theta A' e^{-\theta^2/4} \, d\theta
\]
when $x = 0$ and at any $t$, $\theta = 0$ and $v = v^0$ (a const. say)

i.e. $v^* = v/v_0 = v^o/v_0$

therefore

$$v = v_0 + 2A \int_0^{\theta/2} e^{-\theta^2/2} \, d(\theta/2)$$

which can be developed into

$$v = v_0 + A \text{erf}(\theta/2)$$

when $x \to \alpha$ at any $t$; $\theta = x/\sqrt{kt} + \alpha$ and $v \to 0$ at infinite depth.

Hence,

$$0 = v_0 + A \text{erf}(\alpha)$$

Since $\text{erf}(\alpha) = 1$, $A = -v_0$

that is

$$v = v_0 - v_0 \text{erf}(\theta/2)$$

$$= v_0[1 - \text{erf}(\theta/2)]$$

$$= v_0 \text{erfc}(\theta/2)$$

therefore

$$v = \frac{2v_0}{\sqrt{\pi}} \int_{x/2\sqrt{kt}}^{\alpha} e^{-u^2} \, du$$

Now having the same initial conditions but with the changed boundary conditions

$v = f(t)$ at $x = 0$; $t \geq 0$

Applying Duhamel's theorem, it can be developed to
\[ v(x, t) = \frac{2}{\sqrt{\pi}} \int_{x/2\sqrt{kt}}^{\alpha} f\left(t - \frac{x^2}{4k\mu^2}\right) e^{-\mu^2} d\mu \]

where \( \mu^2 = \frac{x^2}{4k(t-t')} \) and \( t' \) is a fixed parameter.

This is the solution which satisfies the differential equation and the initial and boundary conditions:

Now \( f(t) = V_0 \cos 2\pi ft \) at \( x = 0; t \geq 0 \)

\[ f\left(t - \frac{x^2}{4k\mu^2}\right) = V_0 \cos 2\pi f \left[ t - \frac{x^2}{4k\mu^2} \right] \]

then the solution is

\[ v(x, t) = \frac{2}{\sqrt{\pi}} \int_{x/2\sqrt{kt}}^{\alpha} V_0 \cos \left\{ 2\pi f \left( t - \frac{x^2}{4k\mu^2}\right) \right\} e^{-\mu^2} d\mu \]

\[ = V_0 e^{-x\sqrt{w/2k}} \cos (wt - x\sqrt{w/2k}) \]

\[ - \frac{2V_0}{\sqrt{\pi}} \int_{x/2\sqrt{kt}}^{\alpha} \cos \left( wt - \frac{wx^2}{4k\mu^2} \right) e^{-\mu^2} d\mu \]

neglecting the transient disturbance (the second term) caused by starting the oscillations of surface velocity at time \( t = 0; \) it dies away as \( t \) increases, leaving the first term which is a steady oscillation.

Therefore

\[ v(x, t) = V_0 e^{-x\sqrt{w/2k}} \cos (wt - x\sqrt{w/2k}) \]

\[ V_x = V_0 e^{-x\sqrt{w/2k}} \cos (wt - x\sqrt{w/2k}) \]

where \( w = 2\pi f \).
Therefore

\[ V_x = V_0 e^{-x\sqrt{\frac{\pi^2 \sigma^1}{\eta}}} \cos (2\pi ft - x\sqrt{\frac{\pi^2 \sigma^1}{\eta}}) \]

From the above equation it is found that the fluid velocity becomes negligible over a distance from the deck implying that the fluid resistance needs to be considered for fine particles only or, in other words, the fluid/m part of the equation (8) can be neglected for coarser particles and, furthermore, only fine particles are controlled by both size and density and the size has no effect in the case of coarse particles.

A solution could be obtained for the asymmetrical motion also if \( f(t) \) is known. However, in the absence of an analytical solution a numerical solution may be sought. This study, to obtain a solution for the asymmetrical motion, now in progress, will be discussed in a later report.

In general it is postulated (Gaudin, 1939; Mills, 1978) that the solution for the asymmetrical motion will be of the trigonometric type as in the case of the symmetrical motion and therefore it can be safely assumed that small and heavy particles will drift further than the coarse and light particles depending on the decreasing value of the fluid resistance.

The foregoing discussion is limited to a layer of one or two particle diameter only. In the event of a layer of several particle thickness, the particles must be continuously brought to the top where the thin film can effectively skim them.

Bagnold (1954, 1956-1957) argued that in grain transportation by fluid, there must be a dispersive grain pressure of such a magnitude that an appreciable part of the grains is in equilibrium between it and the force of gravity which otherwise look mysterious. Bagnold forces have been discussed in the previous papers of this review series showing how and why they should lift the coarse particles preferentially.

The riffles are employed to enable the deck to treat pulsating dilated
layer of several particle thick. Pulsating dilation of the bed between the riffles is the result of the combined effects of the asymmetrical motion of the deck, Bagnold dispersive forces and the turbulence in the down-slope part of the bottom layers caused by the obstruction of the riffles. The asymmetrical motion and the Bagnold dispersive forces are the driving forces for the trickling of the fine particles through the intersticies and the systems strive to attain the minimum potential energy. The turbulence causes a certain degree of hindered settling effect at the upper side of each riffle space and the cumulative effect of this, caused by the whole riffles, should be of sufficient magnitude. Kirchberg and Berger (1960) favoured a riffle pattern which would promote stratification by lifting the fine particles without creating excessive eddies.

Pneumatic tables are used under circumstances where water is scarce, soluble salts are present or wetting is not desired. The basic principles of dry tabling hardly differ from those of wet tabling. Here a throwing action is employed to move the particles stratified by the air blown through the porous riffled deck. The lower layers of particles are transported along the riffles by the reciprocating motion. A lengthy discussion on pneumatic tabling is avoided considering the usual coarse size range that is treated and also due to the very limited application. Taggart's Handbook and the papers by Hudson (1962) and Thomas (1978) are cited to those who are interested.

DESIGN AND OPERATIONAL VARIABLES AND THEIR INTERRELATIONSHIPS

The variables of the shaking table can be broadly split into two categories - the design variables and the operational variables. The design variables are:
(a) size and shape of the deck
(b) differential action (acceleration and retardation)
(c) riffle pattern
(d) surface material and its roughness.

The operating variables are:
(a) strokes per minute
(b) length of the stroke
(c) table tilt
(d) end elevation  
(e) pulp density  
(f) feed rate and uniformity  
(g) wash water  
(h) position of product splitters.

Concentrating decks are essentially a rectangular or rhombohedral shaped deck supported by suitable frame and bearings or suspended by cables. A table with a rhombohedrally shaped deck effectively uses deck area since the materials tend to distribute in a diagonal direction and also it can utilize more riffles which is a desirable characteristic for high-tonnage unclassified middling material to be separated.

Since the table capacity with respect to floor area is severely restricted double- and triple-deck tables are used. The advantages are quite a number: low floor space requirement; absence of robust supporting structure when they are suspended on wire ropes (however, there is an element of caution in suspending the decks because it seems that the efficiency of the shaking table is affected due to the non-horizontal oscillation of the deck and therefore the failure of the particles to travel in an efficient manner along the deck (Siirak et al., 1978; Guest, 1980); low cost of installation and decreased requirement of auxilliary items such as pipes, wiring etc.; ease of adjustment; less maintenance and low operating costs.

The modern shaking table surfaces are made of layers of fibre glass reinforced by polyester resin with a top surface of hard gel with the riffles incorporated as a part of the mould. It can also be of rubber or linoleum covered or wood such as marine plywood.

The differential action shown in the previous section causes lateral separation and also assists vertical separation in conjunction with the flow of fluid. The important parameters of differential action are the periods of acceleration and retardation, the speed and the length of the stroke. The deck has to come to a momentary stop at each end of its travel for the direction of movement to be reversed and also it reaches the forward end of its travel with greater speed than the
other to give momentum to the particles enabling them to travel further during the return stroke. Therefore, the rates of retardation and acceleration must be greater at that end than the other. Studies on the longitudinal motion are possible by experimental techniques. One such study was done by Muller and Pownall (1961-1962) with the intention of developing new drive mechanisms. Description of the experimental techniques employed are found in their paper. However, the Wilfley (1982) Ltd. has developed a simple device which is said to be capable of giving accurate displacement time diagrams and one of their modern tables is fitted with it. It is also possible perhaps to obtain the accurate motion pattern by the analysis of the mechanics of the head motion of the individual devices. In their study (1961-1962) they analyzed the differential motion behaviour of the Micropanner (Muller, 1958-1959), the Superpanner (Haultain, 1937) and the standard laboratory Wilfley Table. Displacement time diagrams were obtained at different speeds, stroke lengths and spring tension.

In the case of micropanner they found that the traces obtained for the forward motion conformed closely with the ideal longitudinal motion but the traces for the return stroke which is essentially spring controlled had higher acceleration which is undesirable since it could cause the heavy particles to get to the light particles side.

In the case of the superpanner it was noticed that the forward stroke of it began with a relatively large oscillation owing to the rebound of the deck from the bumping block. Also, it was noticed that the rate of acceleration was low during the forward stroke. The motion diagram study of the micropanner and superpanner led them to develop the macropanner and subsequently the pulsepanner, incorporating new drive mechanisms to eliminate the oscillations at the start of the forward stroke and also to give reduced acceleration at the end of the return stroke. In the pulsepanner, as the name itself implies, a pulsation is given to the bed of particles by passing water through a porous deck. The comparative tests reported by them gave improved results, particularly with the pulsepanner.

The encouraging results obtained led Muller and his co-workers (Muller et al., 1964) to develop a pilot-scale pulsepanner which is actually a
laboratory-scale shaking table fitted with a porous deck surface. This development is unique in the sense it incorporated entirely different sorting methods of jigging and thin-film concentration together. Both the jigs and the shaking concentrators allow the heavy particles to settle at the bottom but when sizes are considered the jig places the smallest particles on the top of the bed whereas the opposite is found with the shaking concentrators. The combined action can be expected to give a better performance, particularly with an un-sized feed. Also, the Pulsed Deck Gravity Concentrator is in possession of some extra variables in addition to the variables of a shaking table—phasing of the pulsed cycle relative to the deck motion, amplitude of the pulsed stroke and addition of make-up water to the porous deck. Again, positive results were obtained in the tests.

Muller et al. (1964) suggested the possibility of pulsed table flotation using the above unit. In fact table flotation is a long known concept. In the Holman-Michell Flotation Table (Pryor, 1974), air is blown into pipes which cross the table deck. One area where it was used was in the removal of sulphide from a rougher concentrate of cassiterite. Another process (Chapman and Littleforth, 1934; Moudgil and Barnett, 1979) was used for concentrating coarse phosphate rock using the usual Deister table. Here the pulp becomes aerated while being fed to the table.

A differential motion is obtained by the various designs of pitman and toggle made to act through an eccentric or a crank shaft, and by supporting the deck on a subframe or from hungers. The Concenco counter-rotating gear drive table headmotion is of a novel design which is claimed to be capable of giving perfectly uniform and exact differential motion at every stroke. An excellent description is given by Deurbrouck and Palowitch (1968).

The speed of the deck usually varies between 250-300 spm and stroke length can be as high as 18 mm. For washing small coals strokes up to 30 mm are used (Deurbrouck and Palowitch, 1968; Pryor, 1974). Usually an increase in the length of the stroke requires a decrease in the number of strokes and vice-versa to achieve optimum operating
conditions. A high amplitude is necessary when treating coarse particles to achieve complete dilation along with lower acceleration. Greater force and acceleration can be obtained by adjusting the tension on the spring or in the case of Concenco head motion, by having greater rotating weights. Greater rotating weights also give a high inertia to the deck which can have adverse effects such as complete collapse of stratification. Low amplitude and low frequency will give too low an inertia of the deck. For fine particles low stroke length and high speed are preferred.

Kirchberg and Berger (1960) did an extremely useful study on the effects of the longitudinal motion on table concentration. They probed the amplitude and frequency of the oscillation at which the loosening of the bed began, the amplitude of the longitudinal oscillation of particles at various depths in the bed and the phase differences of oscillation, the permeability of the loosened particles and other similar phenomena. Also, they studied the effects of riffle patterns on the formation of eddies.

The riffles of many designs can be used - the most common are the line of motion type but they can be slanted or curved too. The riffle pattern and height vary according to its duty - coal cleaning, scavenging or any other standard duties. The riffles have a maximum height at the feed end and generally die out at the discharge end or have a negligible thickness. In Deister tables the thin riffles are replaced by other larger riffles at varying intervals which provide pools to permit the settlement of fine grains. In addition, it can also prevent the heavy particles or middlings reporting to the light side.

On a partially riffled deck the discharge of the particles is as follows: coarse light particles placed on top will be the first to be acted upon by the thin film flow and discharged at the near side of the feed end. The small heavies will be discharged at the farthest end because they travel fast along the deck and moreover they are the last to be acted upon by the cross flowing film of water because of the tapering of the riffles. Intermediate particles will be discharged at intermediate distances.
Fully riffled decks which consume more wash water to cause the gangue particles to discharge more rapidly are used for roughing duties. On a fully riffled deck, coarse heavy particles are taken to the farthest followed by successively smaller heavies towards the near side.

The deck with riffles slanted towards the feed end are thought to be efficient because they force the pulp to climb slightly on its way across. However, there are adverse comments too about the use of special forms of riffling (Pullar, 1966; Pryor, 1974).

For very fine particles the decks with a series of planes rather than riffles are used to ensure a laminar film. The slime tables are used for a feed with a maximum of about 150 mesh.

In certain plants mill water is recirculated as wash water for the shaking table operation. It is generally discouraged fearing an increase in the apparent viscosity (Johnston, 1965; Michell, 1968) and also because of the coating of the deck with subtle layer of slippery slime. The recent work at the Division of Mineral Processing, Technical University of Luleå has shown that the fear of viscosity has no ground. In the light of this knowledge about the insignificance of the viscosity, it should be economical to recirculate the mill water as wash water. Washing the decks at regular intervals will alleviate the problems of slippery coating, if any. The works on the apparent viscosity study will be published soon.

The rising of the heavy discharge end (end elevation) is sometimes seen as necessary to obtain a better grade, particularly when the light particles are relatively big in size (Sala, 1975).

An increase in side slope is used to counter-balance the adverse effects of lack of wash water or a thick-bed feed. It appears only trial and error methods can give the correct end and side elevations. Wilfley and Holman tables are fitted with a tilting mechanism operated by means of a handwheel and the modern suspended tables are adjusted for side tilts and end tilts by altering the length of the cables. The Wilfley '76' concentrating tables incorporate an almost frictionless
spring-suspended system and a simple device to read the angle of tilt.

The feed rate depends on the requirements such as the degree of upgrading required, amount of tailing in the feed, size distribution and loss of fine heavy minerals to the tailing side.

A uniform supply of water can be ensured by so arranging the water pipes and valves that each table gets its flow of water, independent of the others. Blocking and abrasion of valves must be attended and also clean water and constant pressure must be ensured. Too much water will carry away the fine heavy minerals.

Changes in size distribution can occur for a particular feed, due to segregation, in an improperly designed bin if used ahead of the table. Usually mechanical feeders are preferred to avoid this. Hydraulic sizing of the table feed is suggested for a feed with low specific gravity difference. A multiple-deck continuous discharge hydraulic classifier must be appropriate for this purpose (Pryor, 1974; Tiernon, 1978). A cyclone, screen, screw or rake classifier should also meet this requirement.

The recent developments of the shaking table should include automatically controlled product splitters. One is the use of photoelectric cells as sensors to differentiate between two products of varying reflectivity (Welsh and Oeurbrouk, 1972) and the other is the use of a scanning unit - A Geiger Muller Counter in the concentration of radioactive ores (Nair et al., 1974). Improved efficiency has been reported by the use of automatic splitters.

APPLICATION OF THE SHAKING TABLE

Jigs, shaking tables, heavy media systems, water-only cyclones and froth flotation are the widely used methods in coal cleaning. Usual flowsheets of coal cleaning contain jigs as the primary cleaning units with secondary cleaning by heavy media systems. In almost all the coal cleaning plants world-wide, additional coal cleaning systems are also found to meet the different requirements such as low specific gravitites or fine particle sizes (Tiernon, 1973; Zimmerman, 1981,
It appears that jigs and shaking tables are very efficient in the specific gravity range 1.50/1.60 to 1.80/1.90. Efficient separation for specific gravities 1.40 or low heavy media equipment are required depending on the particle size.

The use of shaking tables in coal cleaning is much dependent on the nature of the presence of pyritic sulphur - whether it is free or associated, because the table is efficient in removing free pyritic sulphur only. Furthermore, the table can be used for the cleaning of oxidized coal with a maximum size of about 1 mm.

As known, the main reason for which the shaking table lost its popularity is the low throughput rates. Recently, KHD Humboldt Wedag AG, West Germany (1982) has developed a new shaking table system with a feed capacity of 350 tons per hour. Here 24 tables are set in motion by only six drives. This system saves a lot of space and is claimed by the manufacturer as capable of giving substantially improved efficiency.

The papers of Zimmerman (1950) and Copeland et al. (1960) give good reviews on the extensive use of shaking tables in fine coal cleaning in the 1940's and 1950's respectively.

Tabling efficiency is exceptionally high when the specific gravity difference between the valuable and gangue minerals is high (say about 3). Shaking tables can also perform well with systems of low specific gravity differences such as the separation of barite, iron sulphide, zircon or ilmenite from quartz, providing the feed is classified. At present shaking tables are widely used in the treatment of ores containing tin (Jones, 1967; Penhale and Hollick, 1968; Permpoon, 1969; Henley, 1972; Lloyd and Jackson, 1973; Osborne, 1973; Amsden, 1974; Eropkin et al., 1974; Kraft, 1978; Ottley, 1979; Anonymous, 1980), tungsten (Anonymous, 1963; Ottley, 1979; McNeill, 1982), barite, titanium, tantalum (Raicevic, 1968; Kraft, 1978; Ottley, 1979; Burt, 1979), zirconium and mica minerals. Other uses are in gold (Raicevic and Cabri, 1976; Nendick, 1983) and radioactive minerals. In a good recent paper, Bazzanella and Weyler (1980) discussed the basic design of a gravity plant with a detailed analysis on four currently
operating gravity plants.

Pneumatic tables are used for the removal of quartz and light heavies from zircon and also for monozite recovery from fine non-conductor minerals (Pullar, 1966). Coal cleaning and upgrading of asbestos are the other areas where pneumatic tables are applied.
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Recovery of heavy minerals from slimes

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Abstract

Development of various slime gravity equipment is traced and the merits and demerits of these are discussed and compared.

The significance of the design and operational variables and their interrelationships are examined. Emphasis is placed upon the necessity to selectively modify the particle surfaces in order to achieve selective aggregation in conjunction with selective dispersion, in slime gravity concentration. The significance of the deck roughness and the insignificance of the viscosity under the conditions which would normally prevail in slime gravity processes are noted.

The possibility of having slime gravity circuits for feeds with a top size of about 45-53 µm only is stressed and selected examples of plant flowsheets are given.
INTRODUCTION

Firstly, there is considerable confusion regarding the terms describing the particle size ranges in mineral processing. For example, Hukki's classification for size reduction differs from that of Somasundaran's (1980) for fine particle processing. However, Somasundaran's classification cannot be strictly applied to gravity concentration as seen clearly in his definition of the "fines".

Therefore, it has been decided to propose a classification to categorize the ranges of particle sizes with particular reference to gravity concentration, based on the behaviour of particles of different sizes in gravity processes:

- Super colloids: particles less than 0.2 \(\mu m\) in size
- Colloids: particles less than 1 \(\mu m\) in size
- Ultrafines: particles less than 5 \(\mu m\) in size
- Very fines: particles less than 20 \(\mu m\) in size
- Fines: particles less than 100 \(\mu m\) in size
- Intermediates: particles less than 500 \(\mu m\) in size
- Coarse: particles more than 500 \(\mu m\) in size

The above classification does not include the most widely used term "slime". This particular term has acquired a conventional usage among those involved in gravity concentration. It covers all the particles less than about 45-53 \(\mu m\) in size, conventionally. With the intention of respecting the conventional usage the term slime can continue to describe all the particles less than about 45-53 \(\mu m\) in size including very fines, ultrafines, colloids and supercolloids. However, immediately confusion arises regarding the conventional usage of the term "desliming" in gravity concentration and in other areas as well. Gradual replacement of this term by "ultrafine removal" and "colloidal removal" can bring this confusion to an end.

The current most pressing problem of the mineral industry is the difficulty of recovering valuable minerals from the slime. The various aspects of these problems have been discussed by many, notably Klassen and Mokrousov (1963), Collins and Read (1971) and Somasundaran (1983). The main causes are small mass, low momentum, colloidal coating,
heteroaggregation, high surface area, increased surface energy and also increased pulp viscosity.

Collins and Read (1971) adequately summarized the merits and demerits of the established processes' worthiness in dealing with slimes. They cited the main problems of extracting metal salts from very dilute solutions, low efficiency of ordinary froth flotation in the very fine size range, particularly with the oxide and silicate minerals, problems in recovering the flocs formed in wet magnetic methods against the inertia of the flow, limitations of the conventional gravity processes and also the inabilities of the various dry processes. They also discussed the then (1971) new processes - carrier flotation, the Mozley frame, selective flocculation etc. Since then there has been a lot of development in the above and also of the certain new processes with potential applications to the slime particularly very fine particle treatment. These have achieved varying degrees of success under certain conditions or circumstances. Somasundaran (1983) has discussed some of the successes of the present time and the associated difficulties.

The necessity to treat the slime efficiently was felt as early as late 19th century, by those involved in gravity concentration. Buddles, strakes and vanners were the earliest models. Subsequently, slime tables and much later the Sullivan deck were developed. Taggart's Handbook gives detailed descriptions of all the aspects of the round frames (circular buddies), strakes and vanners and also a brief description of the other types of stationary and rectangular buddies. Also, sufficient information is found in the old text books and other old literature. Here it will only be said that buddies, strakes and vanners, in all the available forms suffer from very low capacity per unit area, low-grade concentrates and the failure to treat very fine particles with appreciable efficiency. Also, the maintenance cost is comparatively high for the round frames, strakes and vanners. There are still a few scattered plants which make use of this old equipment because of the availability rather than for any other reason, except perhaps for the strakes in the recovery of precious metals.

After a lapse of about 50 years caused by the tremendous success of
flotation there has been a renewed interest in gravity concentration. The reasons are increasing reagent costs, environmental pollution and the limited success the flotation has had with oxide minerals.

The purpose of this paper is to deal with the development of the modern and relatively old gravity equipment for slime treatment, their principles, the many variables and their interrelationships and also their application. The theoretical work on the behaviour of particles in slime gravity processes is scarce and therefore it is excluded from discussion. Such an attempt to analyze the behaviour of particles in slime gravity processes, in terms of the established theories of mineral processing and fluid and particulate dynamics, now in progress, will be discussed in a later report.

DEVELOPMENT OF GRAVITY EQUIPMENT FOR SLIME RECOVERY

The slime gravity equipment considered here are the Denver-Buckman tilting frame, the shaken helicoid, the Bartles Mozley Separator (or Mozley's frame), the Bartles crossbelt concentrator and the latest unit Duplex concentrator. A brief account is given on the slime shaking table since many aspects of it have already been dealt with.

The slime table is essentially a smooth table with a series of pools aiming at a laminar film. The disadvantage here is the differential motion of the table. The rapid change of the direction of the deck which is very acute at the end of the forward stroke, can cause excessive turbulence which is detrimental to efficient recovery of the slime particularly very fine particles. A standard slime circuit will include the slime shaking table to upgrade the rougher concentrate. An upgrading of 8:1 has been achieved with the Ta rougher concentrate (all -45 μm), Tanco mill, Canada on a Holman slime table (Burt, 1979). Burt (1980) has expressed his preference for a Holman slime table to a Deister slime table. The low riffles found on the Deister table can be expected to seriously disturb the stratification when treating feeds containing high amounts of very fine particles. Much higher upgrading is possible on a slime table if the feed is of the correct nature.
The Denver-Buckman tilting frame is the developed commercial form of the Sullivan deck (Thunaes and Spedden, 1950). Like the Sullivan deck it is a stationary unit (Chaston, 1962). The decks are inclined at a comparatively steeper angle (about $10^\circ$). This is to achieve a high shear by having a big stream velocity. The high shear thus obtained dilates the bed sufficiently in order to bring the coarse, light particles into the fast-flowing layers of the film. However, this high slope results in the excessive removal of the very fine valuable particles as well, together with the lights.

For example, the fine-particle gravity circuit at Sullivan concentrator in Canada, which consists of Denver-Buckman tilting frames for roughing, cleaning and scavenging followed by Deister shaking tables to upgrade the concentrate, recovered cassiterite particles down to 40 $\mu$m only from the sulphide flotation tailing, though the cassiterite is fully liberated down to the 10 $\mu$m level and the pyrite in the tailing was floated prior to the gravity circuit. The recovery was less than 30% at a grade of 35% Sn (Furey, 1983).

The shaken helicoid was the innovative idea of Burch. Burch was one of the few to promptly perceive the value of the Bagnold theories on dispersive forces (refer the review papers Principles of sluicing and Principles of spiral concentration). Burch's shaken helicoid (Burch and Mozley, 1956) is, actually, a spiral concentrator but with a very low pitch and a much flatter trough profile and is shaken on an orbital plane by one or more out of balance masses. The shaken helicoid, unlike a spiral concentrator, can treat slime size particles effectively. The low pitch and the rather flat profiles of the helicoid troughs ensure a smooth pattern of flow with a low stream velocity. The shear produced by the orbital motion of the helicoid is expected to preferentially lift the coarse light particles, along with the lift of the small light particles, by rising secondary flows. The idea is to have particles of different sizes lifted to the same extent by correctly proportionating these effects, so that separation can be achieved based on density only. The main disadvantages of the helicoid are low throughputs and mechanical deficiencies.

Mozley, who introduced the resin bonded fibreglass trough surfaces to
gravity equipment by moulding the shaken helicoid out of fibreglass, incorporated the orbital shear principle of the helicoid and the many basic design principles of the Wilfley multiple-deck tilting slimer and the tilting frame into the practical form - the Mozley frame (Mozley, 1967). The commercial model is the Bartles-Mozley (BM) separator. The merits of this device are large total feed width, suspension system, mechanical simplicity, low floor space requirement, low tilt angle and automated operations. Here, the shear forces are generated on a plane parallel to the deck which avoids any splashing action as may be obtained on the shaken helicoid because of its non-tangential oscillation. Amplitude could be controlled by varying the eccentric weight, and the radius of its rotation.

Test results reported by the manufacturers (Burt and Ottley, 1973, 1974) include a recovery of 47% with an enrichment ratio of 6.4 on 45%, -10 μm cassiterite feed (0.36% Sn), a recovery of 70% with an enrichment ratio of 2.8 on a 60%, -45 μm scheelite(0.25% WO₃) etc. On another study made to verify the performance at different size ranges, good results have been obtained down to 10 μm (Table 1). As can be seen in the data given an enrichment ratio of more than about 3 is hardly achieved which makes it essentially a high-capacity preconcentrator. As on a slime table or Bartles crossbelt concentrator (see below) there are not any different zones for feeding, washing and cleaning, a design element of the utmost importance to obtain high grade concentrates.

The underlying principles of the Bartles crossbelt concentrator of superimposing the belt with an orbital motion was first thought of by Mozley. Mozley employed two masses with the intention of giving two different rates of shear at the tail and concentrate ends thereby simplifying the mechanics of the vanner shaking principles. The idea failed because of the low metallurgical performance compared to the slime table. Also no attempt was made to overcome the low throughputs - the inherent drawback of the conventional vanner (Burt, 1975a). Burt developed this idea into the present model known as Bartles (XB) crossbelt concentrator. On this the settled bed of particles is removed sideways after subjecting the bed to transverse wash prior to the discharge of the final concentrate. Ample description of the unit
is found in the literature (Bartles, 1975; Burt, 1975a,b, 1977a,b).

**TABLE 1**

Performance of BM separator on a tin tailing (after Burt and Ottley, 1973)

<table>
<thead>
<tr>
<th>Particle size (μm)</th>
<th>Recovery (%)</th>
<th>Enrichment ratio</th>
</tr>
</thead>
<tbody>
<tr>
<td>+53</td>
<td>87.4</td>
<td>1.48</td>
</tr>
<tr>
<td>-53 +20</td>
<td>89.6</td>
<td>3.86</td>
</tr>
<tr>
<td>-20 +10</td>
<td>54.3</td>
<td>2.13</td>
</tr>
<tr>
<td>-10 +5</td>
<td>18.7</td>
<td>1.71</td>
</tr>
<tr>
<td>-5</td>
<td>3.2</td>
<td>1.00</td>
</tr>
<tr>
<td>Head calc.</td>
<td>52.3</td>
<td>2.59</td>
</tr>
<tr>
<td>Actual</td>
<td>57.6</td>
<td>2.49</td>
</tr>
</tbody>
</table>

The main advantages of the unit are: the ability to get a third product—middling which is not obtained with the vanner since it is a two-product unit only; better performance than the Holman slime table (Table 2); efficiency in recovering very fine particles; higher capacity than the traditional vanners and the wider length of the concentrate band (about 1.6 m) than that of the slime and sand tables (about 0.2 m) which enables an efficient cut between concentrate and middling bands.

The Duplex concentrator (Mozley, 1980; GEC, 1981a,b; Burt and Mills, 1982) is of a kind of design, previously unknown. The laboratory version of this is the well-known Mozley separator (Anonymous, 1979). This device differs from the shaking table in that it employs, instead of an asymmetric periodic motion with a sharp return at the end of the forward stroke, a symmetrical motion which could be thought of as a nondiscontinuous motion. With the laboratory device the optimum conditions for banking the concentrate to the deck were examined in the early phase of the development. It was found that with a normal feed, a much larger amplitude and lower frequency are necessary. On a shaking table the amplitude and the frequency must be such that while assisting the banking of the heavy mineral the rapid longitudinal transportation of the minerals must also be promoted. Since the unit's aim is only to collect the heavy mineral and wash it
thoroughly, the best conditions for collection alone without any compromise can be employed. The collected products can be washed into different launders during wash cycle. A detailed description of this unit is found in the literature cited above.

TABLE 2

Comparison of results for slime table and Bartles XB concentrator (after Guest, 1980)

<table>
<thead>
<tr>
<th></th>
<th>Slime table</th>
<th>Bartles XB concentrator</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sn. % (concentrate)</td>
<td>13.4</td>
<td>31.8</td>
</tr>
<tr>
<td>Recovery, %</td>
<td>55.4</td>
<td>52.5</td>
</tr>
<tr>
<td>Enrichment ratio</td>
<td>38</td>
<td>88</td>
</tr>
<tr>
<td>Rec. of 5 (\mu)m frac., %</td>
<td>12</td>
<td>25</td>
</tr>
<tr>
<td>Rec. of 10 (\mu)m frac., %</td>
<td>70</td>
<td>60</td>
</tr>
<tr>
<td>Rec. of 20 (\mu)m frac., %</td>
<td>75</td>
<td>75</td>
</tr>
<tr>
<td>ER for 5 (\mu)m frac.</td>
<td>32</td>
<td>50</td>
</tr>
<tr>
<td>ER for 10 (\mu)m frac.</td>
<td>35</td>
<td>95</td>
</tr>
<tr>
<td>ER for 20 (\mu)m frac.</td>
<td>35</td>
<td>80</td>
</tr>
</tbody>
</table>

The comparative studies with the slime table (Holman) have shown marked improvements (GEC, 1981b). Because of the novel concentrate discharge design this has the ability of giving a very high grade in a single stage.

The rocking-shaking vanner developed in China is another slime gravity equipment. This is basically a vanner but with a belt of trough-like profile upon which two shaking motions are superimposed in the longitudinal and transverse directions. The transverse motion is more a simple harmonic motion and the longitudinal motion is that of the shaking table - asymmetric motion with a sharp return. This longitudinal motion in addition to dilating the bed should also urge the settled material forward. The intensity of the transverse motion which is determined by the frequency and angle of action, is limited and therefore this cannot be expected to have any adverse effects. They form only weak eddies. The longitudinal motion should disturb the stratified bed particularly of very fine size ranges leading to losses. Their performance study also showed a large drop in the recovery of very fine particles (Chin et al., 1979). Comparative studies with a flat surface vanner and a slime table have shown better results. But apart from the disadvantage of the inability to treat the
very fine particle effectively as does the slime table or Bartles XB concentrator or Duplex concentrator. It also has the serious drawback of having a very low capacity (30 kg/h).

Centrifugal methods of concentrating minerals have two major advantages - a very high g (centrifugal force) and an artificial shear on the particles in a continuous manner. The Chinese have used large capacity centrifugal machines for roughing operations of the slime (Chin et al., 1979). Ferrara (1960) developed a rotating tube for the centrifugal separation and obtained selective separation independent of the particle size. In addition he could mathematically predict the results. The Russians also have developed a centrifugal hydroseparator for slime treatment with facilities for continuous discharge (Revnivtsev et al., 1973). A recent device that has become popular for treating precious metals of fine size is the Knelson concentrator. Performance studies at the Colorado School of Mines have shown very impressive results and in fact it has outperformed Mark VII spiral concentrators and Deister slime shaking tables. However, application of this to the usual heavy minerals remains to be proved. Some serious problems will have to be solved if it is to be applied to the heavy minerals, among which are those of selectivity and obtaining the discharge of the heavy minerals. Furthermore, the inherent friability of certain heavy minerals discourages any utilization of very high centrifugal forces. However, as the Russians have shown in their interesting paper (Revnivtsev et al., 1973) by developing regression models for the main variables, proper separation may be achieved without excessively increasing the centrifugal forces but by adjusting the other design or operating variables.

Table 3 gives the recovery and enrichment ratio of many of the slime gravity equipment, obtained from the report figures to facilitate a comparison. It must be cautioned that only a general comparison is possible because of the different kinds of feed.

Table 4 gives the capacities/area of the floor space on the basis of the usual minerals, obtained from the reported figures.
TABLE 3

Comparison of the slime gravity equipment

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Feed (% Sn)</th>
<th>Concentrate (% Sn)</th>
<th>Recovery (%)</th>
<th>ER</th>
<th>Reference</th>
</tr>
</thead>
<tbody>
<tr>
<td>Vanner</td>
<td>0.58</td>
<td>4.95</td>
<td>19</td>
<td>8.53</td>
<td>a</td>
</tr>
<tr>
<td>Round frame</td>
<td>0.59</td>
<td>1.4</td>
<td>20</td>
<td>2.39</td>
<td>a</td>
</tr>
<tr>
<td>Helicoid</td>
<td>0.41</td>
<td>2.4</td>
<td>30</td>
<td>5.85</td>
<td>a</td>
</tr>
<tr>
<td>Tilting frame</td>
<td>0.81</td>
<td>4.01</td>
<td>40</td>
<td>4.95</td>
<td>b</td>
</tr>
<tr>
<td>BM separator</td>
<td>0.46</td>
<td>1.19</td>
<td>52.3</td>
<td>2.59</td>
<td>c</td>
</tr>
<tr>
<td>Slime table</td>
<td>0.35</td>
<td>13.37</td>
<td>55.4</td>
<td>38.2</td>
<td>d</td>
</tr>
<tr>
<td>Bartles XB conc.</td>
<td>0.36</td>
<td>31.76</td>
<td>52.5</td>
<td>88.2</td>
<td>d</td>
</tr>
<tr>
<td>Duplex conc.</td>
<td>0.343</td>
<td>10.51</td>
<td>51.25</td>
<td>30.64</td>
<td>e</td>
</tr>
</tbody>
</table>

References:
b - Chaston, 1962.
d - Burt, 1975b.
e - GEC, 1982.

TABLE 4

Capacities of the slime gravity equipment

<table>
<thead>
<tr>
<th>Concentrator</th>
<th>Capacity</th>
</tr>
</thead>
<tbody>
<tr>
<td>Shaken helicoid (single)</td>
<td>32-50 kg/hr/0.84 m²</td>
</tr>
<tr>
<td>Tilting frame</td>
<td>800-2000 kg/hr/3.35 m²</td>
</tr>
<tr>
<td>Slime table (single deck)</td>
<td>300 kg/hr/9.21 m²</td>
</tr>
<tr>
<td>BM separator</td>
<td>2000-4000 kg/hr/5.83 m²</td>
</tr>
<tr>
<td>Bartles XB conc.</td>
<td>500 kg/hr/9.27 m²</td>
</tr>
<tr>
<td>Duplex conc. (three tiers)</td>
<td>1800-2100 kg/hr/12 m²</td>
</tr>
</tbody>
</table>
DESIGN AND OPERATIONAL VARIABLES AND THEIR INTERRELATIONSHIPS

Variables of the slime gravity equipment are many and the performance of them generally varies considerably, according to the conditions of the operation. The common variables are:
(a) pulp density
(b) pulp flowrate
(c) particle size range and distribution
(d) cycle time (excluding the shaken helicoid)
(e) amount of wash water
(f) viscosity
(g) pH of the suspension
(h) cation concentration
(i) deck slope (excluding the shaken helicoid)
(j) eccentric weight and amplitude (excluding the Duplex concentrator)
(k) frequency of the oscillation (excluding the Duplex concentrator)

In addition to the above, the shaken helicoid includes the additional variables belt speed and the longitudinal slope (in addition to the side slope-transverse slope).

The Duplex concentrator since it employs a simple harmonic motion, includes the number of strokes and the amplitude of the stroke.

The helicoid though it is not in use anywhere has been included with the other proved units because it is considered by the author that the idea (or theory) proposed by Burch is valid and promising. It must be noted that general criticism is directed at the low capacity and mechanical deficiencies only, not at the theory. It must also be noted that in China a revolving spiral concentrator has been developed and it is reported that recoveries down to 20 μm have been obtained (Robinson, 1983).

Pulp density and pulp flowrate decide the capacity of a unit, depending on the required metallurgical performance. Also, pulp flowrate is affected by the deck slope and the viscosity. Figure 1 correlates the pulp density to recovery and enrichment ratio, obtained
by treating a wolframite fine feed containing 74% particles finer than 53 μm on a shaken helicoid (Douglas and Bailey, 1960-1961). The best results were obtained when the pulp density is between 5-10%. Any increase in the pulp density leads to a reduction in recovery and enrichment ratio, except for a particular point which may be considered as due to the experimental or assaying errors. It is known that free settling conditions must prevail for the fast settling of very fine particles, which ensures the absence of particle-particle interference. Mills (Mills, 1970; Mills and Burt, 1979) explained it by the wake velocities of the rising coarse particles due to the Bagnold forces. Viscosity can play a role here only when feeds containing substantial amounts of ultrafine particles are treated. Pulps of about 15% solids by weight hardly show any viscosity effects in the absence of sufficient amounts of ultrafine particles under the conditions prevailing in slime gravity processes. A feed with a pulp density of about 10-15% solids by weight should generally be the case for any slime gravity equipment. For instance with the BM separator, recovery was found to drop considerably (from 76% to 65%) when the percent solids in the feed was increased to 18% from about 8% (Burt and Ottley, 1973, 1974).

Pulp flowrates cannot be changed much with the shaken helicoid due to its mechanical design deficiencies. Douglas and Bailey (1960-1961) found a throughput of 8 l/min to be the most suitable. In the test with the BM separator higher flowrates gave a fall in recovery, and lower flowrates, as can be expected, did not have a considerable influence (Fig. 2). This phenomenon is common to all the film-type concentrators. The fast flowing pulp has the ability to remove the fine values. An optimum flow-rate can be expected under a given set of conditions. In the above case it was found around 5 l/sec.

However, the test done by Burt (1975b) to determine the effects of the pulp flowrate on Bartles XB concentrator indicated that the flowrate could be varied over a fairly wide range (+ or - 25% of the 0.20 l/sec/metre width) with hardly any loss in the effectiveness of concentration (Fig. 2). The only explanation that could account for this abnormal behaviour is as follows: the moving belt withdraws almost all the heavier fraction into the middling zone before the
momentum of the flowing pulp can take them over the lower side edge to the tailing launder. All the fine particles of the heavier fraction can be assumed to report to the middling launder. Since, on this unit, middlings are continuously recirculated to the middling container (feed-port) situated very close to the concentrating zone, chances are that the finer fraction will report more and more to the concentrate launder.

Though slime gravity equipment is generally suggested for the recovery of heavy minerals up to about 100 μm in size, a careful study of the many slime plants - the design and performance of the whole circuit and individual devices as well - clearly suggests that it is possible to have slime circuits for feeds with a top size of about 45-53 μm only. The development of modern spirals, efficient slime screens Derrick and CTS particularly, along with the proved modern slime equipment and also the improved plant design and automatic control, can make such a circuit possible. In an all gravity equipment, the coarser fraction can be treated either by modern spirals in conjunction with the shaking tables or by cone concentrators depending on several factors, such as capacity, size range and space. The availability of skilled plant operators will be an important factor in the above choice. It may also be possible to have additional units of slime gravity equipment to handle -74 μm to +53 μm or -20 μm ranges. The percent of the valuable minerals found in these ranges will decide the choice.

Cycle time includes the feed, drain and wash periods. Cycle time will vary according to the grade of the concentrate required and the percent of the heavy mineral in the feed. Excessive collection can increase the system's weight so as to cause a reduction in the amplitude of the orbital motion. It can cause a very uneven false bottom too.

The presence of electrolytes, pH and also viscosity were found to have certain effects on the performance of the slime gravity equipment. So far, only a few studies (Johnston, 1965; Michell and Cosio, 1965-1966; Michell, 1968; Rao and Sirois, 1974; Burt, 1978) have been made on these aspects. Using grade, recovery and efficiency of concentration
as tools the effectiveness of concentration at different pH and cation concentration were studied by Michell and his students (Johnston, 1965, Michell and Cosio, 1965-1966; Michell, 1968). Pulp viscosity and electrokinetic potential were also measured by them at different pH and cation concentration in order to facilitate any establishment of generalized empirical conclusions. However, their studies, to a greater extent, were made on synthetic samples. The main conclusions made were that the efficiency of concentration would be low at low pH due to high viscosity, caused by coagulation, in pulps containing clays and that the greater negative potential at high pH could cause greater mobility of all the particles keeping them in the upper part of the flowing film and thus reducing the efficiency of concentration.

Following Michell and his students, Rao and Sirois (1974) studied the effects of conditioning pH and the apparent iso-electric point (IEP) of the suspensions. He concluded that the greater the difference between the IEP of the bulk and the conditioning pH is, the better the efficiency of concentration.

A detailed comparative study of the above works emphasized the necessity of carrying out more detailed work on the effects of surface characteristics. Furthermore, the importance of thoroughly knowing the response of natural slimes containing heavy minerals to cation, pH and viscosity is also felt. A study on the above factors' influence on natural samples has already begun at the Mineral Processing Division of the Technical University of Luleå.

There have been a few attempts by Michell and others at Camborne School of Mines, Cornwall, to attribute the performance of the decks with different surface materials to the potentials of those surfaces (Michell and Cosio, 1965-1966; Michell, 1968; Michell and Osborne, 1975). Streaming potentials of several substances commonly used for concentrating surfaces, were measured. All the surfaces were found to possess negative potentials with the least value for wood. Burt (1978) did further studies to compare the performance of different kinds of decks at different pH and attempted to correlate the results to the different potentials of those decks. Effects of surface roughness which can have considerable roles on typical slime circuit
feeds containing particles with a top size of about 45 μm, by partly or fully shielding the very fine particles, were not taken into account. This effect could be visualised most readily if one would consider the operating conditions of Burt's (1978) work. If the feed had an average size of about 20 μm (deslimed feed) there would be about 8 layers of particles only, which emphasize the significance of the bottom layers. Also, the nature of the profile patterns of the surfaces, for example sharp tips of the recesses, which can retard the flow of the suspension, should have been considered.

The best way of effectively manipulating the particle-surface characteristics would be to approach this problem as it would be approached in selective coagulation and flocculation techniques. Detailed electrokinetic property studies of the major and valuable mineral constituents of the ore both in distilled water and pulp liquids should be made to facilitate the proper selection of a suitable electrokinetic environment (appropriate pH and correct kinds and amounts of the ions added). The proper selection of the electrokinetic environment may permit the preferential aggregation of the valuable mineral with a slight negative potential while rendering the other constituent minerals stably anionic. Tests over a wide range of the selected electrokinetic environment enable a proper analysis of the influence of added foreign ions. In the case of polyvalent cations such as Al\(^{3+}\) or Fe\(^{3+}\) the production of polymolecular complexes becomes an important factor. Such species could show favourable electrostatic interactions. Inorganic polymolecular complexes can exert their influence on bridging mechanisms in a manner similar to that of the organic polyelectrolytes such as polyacrylamide (LaMer, 1967). Moreover the nature and concentration of anions decide the behaviour of metal hydroxides, depending on the basicity of anions, the coordinative binding affinity of the anion for a particular cation and the resistance of bound anion to displacement by added hydroxyl ion (Marion and Thomas, 1946). Such interactions could take place at mineral surfaces and thereby lead to preferential aggregation by inorganic polymolecular complexes. For example, surface chromate ions of chromite may be expected to preferentially interact with the polymolecular complexes of Al ions or else chromate loaded Al ion complexes may be expected to get preferentially adsorbed on chromite.
The magnitude of the shear caused by the deck motion is determined by the eccentric weight, its speed of rotation and its offset. The studies made on the BM separator (Fig. 3) and the Bartles XB concentrator (Fig. 4) showed steep increases in the enrichment ratio at an increasing rate of shear as against marginal drops in the recovery.

If one assumes that practically all the particles in a feed of coarse lights and fine heavies settle on the deck of a slime concentrator (for example, BM separator, Burt, 1980), the rate of stratification will be slow in the absence of sufficient loosening of the bed. The rate can be promoted by increasing the shear, as evident in Bagnold's analyses and it can also be shown that the loosening occurs due to the lift of the coarse lights in preference to the fine heavies, even if the environment in the concentrator, as some quarters maintain, is viscous. However, for the latter to be valid, the particle size distribution should be within limits. The interstices of this expanded bed pave the way for the heavies to trickle through and also the dilation enables some heavies to move down to lower the potential energy.

The effects of shear is also dependent on the velocity of the pulp. Maximum shear is exerted on the bed when the pulp velocity is roughly equal and opposite to the rate of shear under optimum conditions. Under opposing and equal conditions of the orbital shear and pulp velocity, particles tend to be arrested and fall vertically. This fall will preferentially be the fines. The tests by Douglas and Bailey (1960-1961) with the helicoid gave the optimum conditions at orbital rates which coincided with the mean velocity of the flow. Certain studies by Burt and Stoelzle (1977) also gave results in confirmation with the above postulates.

It was said above that the mass of the orbital weight and the radius of rotation decide the amplitude. In addition, the location of the mass will also have its influence by deciding the effective amplitude at various parts of the concentrating surface. In Bartles XB concentrator, the mass is located in the region where middlings are
directed from the feed zone for subsequent cleaning, thereby obtaining an increasing amplitude of shear. The BM separator which has no such different zones does not require a difference in the shear rate at any parts. The tests with the BM separator (Burt and Ottley, 1973; 1974) and Bartles XB concentrator (Burt, 1975b) further revealed that the same recovery-enrichment ratio could be maintained by correspondingly altering the speed of rotation when changing the mass, within limits. Therefore, the manufacturers decided to keep the mass and location of the above units, decided by prior knowledge, fixed for the sake of simplicity. However, provision may be made to vary the radius of rotation of the mass which may allow some control of the amplitude if desired.

Deck slope is an important factor in determining the capacity and grade since it determines the forward velocities of the stream and of the particles of different sizes and specific gravities, for a given throughput. On a unit like BM separator which has no provision for middling recirculation or collection, recovery can be expected to drop drastically with the increasing slope. However, on a unit like Bartles XB concentrator with washing and cleaning zones and with provisions for recirculation, a marked increase in the enrichment ratio should result with the increasing slope. Generally, a slope which can give the desired maximum flow-rate without much turbulence or removal of fine heavy particles is considered a good slope.

The belt speed can have only a marginal effect on the performance, within limits. It determines the residence time of the particles, settled on the belt.

The amount of wash water is another parameter. There may be achieved high grades without loss of recovery by the use of correct amounts of wash water.

The studies on the effects of variables on the Duplex concentrator are largely limited to the laboratory version of it. Some studies (GEC, 1981b) have been made in Cornwall with a full size, single tier Duplex concentrator. However, the feed studied contained only about 15% of the particles less than 20 µm which is not a "fine feed", in the
strictest sense of the term, such as a BM separator would normally treat. For example, at South Crofty concentrator, only slime tables are used to treat the deslimed fine cassiterite. Coincidentally, the feed for the above studies (GEC, 1981b) was also from a slime table. Conversely, at Hydraulic tin a number of BM separators took not-deslimed waste dump material (Osborne, 1973). Also, the extensive tests done at the manufacturer's laboratory to prove the applicability of the Duplex concentrator to feeds of different minerals, grades and sizes (GEC, 1982) qualify it more as a cleaner than scavenger. Further work is required on materials with known particle size and mineral distributions to obtain a full understanding of the various design and operational parameters involved, particularly the effect of particle size on recovery resulting from the high feed density (about 33% solids by weight) suggested by the manufacturers.

APPLICATION

As seen in this series of review papers, there has been ample development in the art of intermediate and fine particle gravity concentration within the last two or three decades. In addition, there are modern jigs with high capacities for efficient separation in the coarser size ranges. Even centrifugal jigs, receiving the feed within a rotating cylindrical jig bed to separate the heavier fraction from the lighter fraction have been developed. One such jig is patented by Campbell (1981).

In the light of all these advances in gravity concentration technology, along with the parallel advances in instrumentation, classification, screening and pumping, it becomes clear it is possible to have efficient, high capacity gravity plants capable of handling feeds of very wide size ranges (3 mm-10 μm and down).

There are, at present, gravity circuits to scavenge the tailings from other processes (example: flotation) and also to work the old tailing dumps with the intention of producing a material with a grade of the run-of-mine, in addition to their traditional functions of treating beach sands, iron and heavy minerals, coals etc.
The slime gravity circuits are at present found in many plants treating Sn and W. Other areas where such circuits can be potentially utilized are Ta, Cr, Au, Pt, Pb and beach sands of fine sizes. The modern units of such circuits are, as seen in this chapter, capable of recovering heavy particles as fine as 5 μm. Two of the plants with slime gravity circuits are Geevor Tin Mines Limited (Fig. 5) and Berni Lake tantalum Mining Co. (Fig. 6). The treatment methods found in the above flow-sheets and also a few other slime gravity circuits are sufficiently discussed in the literature (Burt and Ottley, 1973; Osborne, 1973; Burt, 1975a; Anonymous, 1977; Burt, 1979; McNeil, 1982).

Slime recovery by gravity methods in conjunction with or without other methods, high-intensity magnetic separation, flotation etc, is a promising route enabling flexible circuits and economical treatment of very low grade feeds. It is worth probing, always, all the possibilities of employing such a circuit successfully, wherever possible.
REFERENCES


Burt, R.O., 1980. Slime recovery by gravity concentration - a viable alternative? In: P.Somasundaran (Editor), Proc. of Fine Particle


GEC, 1981b. Experience with GEC Duplex mechanical concentrator in Cornwall. GEC Mechanical Handling Ltd., U.K.

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Michell, F.B. and Cosio, H.J., 1965-1966. The gravity concentration of
fine cassiterite with special reference to effect of eletrokinetic properties. Camborne School of Mines, U.K.


Fig. 1 Performance at different pulp density
Fig. 2  Effect of flowrate on recovery and enrichment ratio
(after Burt and Ottley, 1974; Burt, 1975b).
Fig. 3 Effect of orbital speed on recovery and enrichment ratio for BM separator (after Burt and Ottley, 1974).
Fig. 4  Effect of orbital speed on recovery and enrichment ratio for Bartles XB concentrator. (after Burt, 1975b).
Geevor Tin Mines - Slime Plant
Flowsheet (1)

Fig. 5
TANCO MILL - CANADA
Flowsheet (2)

1500-ton fine ore bins

Derrick Screen

B.M.MULTI DECK

Concentrate

Fig. 6
Dynamic flow characteristics of thin films

Accepted for publication in the Scandinavian Journal of Metallurgy.
Abstract

Viscosity is generally cited as a chief cause for any decrease in efficiency noticed in concentration processes, particularly with undeslimed feeds, despite the fact that there is little conclusive information available on the importance of viscosity. The body of literature that is available on viscosity, as applied to gravity processes, is therefore critically reviewed here.

The experiments described concern the dynamic flow characteristics of natural slimes.

A slime containing a very high amount of ultrafine particles—exclusively clays (about 55% -5μm), is studied in detail at different slime concentrations and pH. The dynamic flow curves obtained show a considerable range of behaviour. Except for some weakly developed tendency to gel at low pH at low rates of shear, which is shown to be insignificant in slime gravity concentration, they behave almost as Newtonian fluids with extremely low apparent viscosities at typical slime circuit concentrations. At intermediate concentrations, suspensions show a strong gelling behaviour and the pH effects also become pronounced. Nevertheless, at rates of shear as those on the concentrating equipment, pH influence becomes less significant and apparent viscosity drops greatly.

The presence of a very high amount of non-clay ultrafine particles also makes a suspension gel. However, the viscous behaviour is shown to be insignificant under the conditions which would prevail in slime gravity processes.

Finally, a mathematical analysis of the dynamic flow characteristics of the thin films is made as a function of distance from the surface of the deck, with the aid of the experimental results, which convincingly shows that the viscosity effect due to the presence of ultrafine particles is negligible under the conditions prevailing in slime gravity processes.
INTRODUCTION

"Little conclusive information is available on the importance of viscosity" - so write Kelly and Spottiswood, while discussing gravity concentration, in "Introduction to Mineral Processing" - one of the latest text books (1982, Wiley - Interscience) to be published on Mineral Processing.

Nevertheless, there are quite a few articles (5,6,7,9,10,11,16,17) citing viscosity as the deleterious element for the "observed" poor effectiveness of the concentration processes, particularly with undeslimed feeds. How could they all have correlated viscosity to the "observed" drops in effectiveness in the presence of only little conclusive information? What made them choose viscosity as the cause despite the presence of other possible causes - reduced settling rates due to crowding, wake velocities of rising coarse particles and heteroaggregation.

Perhaps the answer lies in those few works on viscosity as applied to gravity processes and therefore it becomes important that one reviews them thoroughly before embarking on any further work in order to bring the correct picture into light.

Schack, Dean and Molloy (4) were the first to recognise the non-availability of any substantial knowledge of apparent viscosities of mineral suspensions encountered in mineral processing. Five different common gangue minerals - quartz, feldspar, calcite, gypsum and muscovite and also an amorphous material - ground pyrex glass, representing different cleavability, shape, crystallinity, hardness, friability and solubility were studied. The objective was to find the relationship among pulp density, particle size, crystallinity, cleavage and viscosity. However, their study was exclusively on particles coarser than 5 μm in size. Despite this, their conclusions are of great importance. Apparent viscosity induced by the above six substances is small (3-10 cp) below a pulp density of 35% solids by weight. Size, angularity or crystallinity has only a small effect up to a pulp density of 30% solids. The general concept that the fineness of particles is a major cause of viscosity was shown to be incorrect,
at least within the experiments carried out. For example, the quartz fraction with an average size of 4 μm in spite of its dilatant behaviour (1,3) is highly fluid at 85% solids by weight than the 20 μm fraction at 75% solids. They also showed that a heterogenous mixture of sizes had a viscosity, intermediate between maximum and minimum values for the individual fractions composing the mixture. In the initial phase of their study it was also found that aging and any treatment with the dispersants of the materials they studied had no effect on the apparent viscosity. The above conclusions led them to suggest that any observed viscous nature of finely ground dilute ore pulps must be attributed to the presence of colloidal and thixotrophic minerals such as clays and that all future works should be directed towards the study of such minerals.

As a consequence, perhaps, Michell and his students (5,6,7,9) embarked on a fairly large study in the mid 60's on the effects of slime on gravity concentration with particular reference to cassiterite slimes. Michell and Cosio (7) wrote that the the presence of colloidal and near colloidal particles would greatly increase the apparent viscosity and thereby influence the penetration of the larger grains. In a set of experiments he (Cosio) carried out with -15 μm Geevor slime, it was observed that the apparent viscosity rose from 14.5 cp to 103 cp when the percent solids by weight was increased from 5 to 20. Johnston (6) also attempted to study the detrimental effects of suspended clay on recovery and carried out experiments with different kinds of clay materials at different pulp densities and conditions. Johnston as well as Cosio, used a superpanner to carry out their experiments in order to correlate the viscosity to the efficiency of concentration.

Using a feed of -16 +8 μm ferrosilicon with -75 μm quartz Johnston produced a model interrelating apparent viscosity, pH and efficiency of concentration (Fig. 1) which is more or less widely published (6,9,16,17).

Cosio also from his experiments with a synthetic feed of -12 +5 μm cassiterite and -32 μm quartz gave a correlation between apparent viscosity and efficiency of concentration, showing that the presence of near colloidal to colloidal particles markedly reduces the
Michell (9) summarizing his students' work, concluded that the efficiency of concentration would be low at low pH values due to high viscosity caused by coagulation. In addition, Burt (16) too concluded, citing Michell and his students, that the efficiency would be poor at low pH values due to high apparent viscosity.

Nevertheless, a careful consideration of the works of Michell and students and also of Burt, suggested otherwise. Viscosities were measured at a low shear rate (not prevailing in any non-stationary gravity concentrators as reported in the original report by Johnston (6) himself). The marked effect, the presence of relatively coarser particles can have by disrupting the gelling slurry structure by their passage through, was neglected. The pH effect on apparent viscosity cannot be as significant as shown in the above works in typical pulps containing less than 5% solids by volume only. This is so even in the presence of substantial amount of ultrafine particles, under the conditions that would prevail in slime gravity processes. It can be shown, assuming an average particle size of 4.4 μm and a tetrahedral packing, that the centre to centre distance between particles in a 6% by volume clay suspension which could very well be taken as an example for a worse slime feed in practice, is 5.8 μm. This distance is sufficiently outside the field due to the charge on the particle surfaces.

Moreover, the influence of other causes - reduced settling rates due to crowding, wake velocities of the rising coarse particles and heteroaggregation - was not taken into account. When a 'mine slime' is added to a feed instead of water, the restraints of crowded conditions become effective and these include friction, localised turbulence and drag. Ferrosilicon was found to have a positive potential up to a pH of 9.5 (9), which can lead to its heteroaggregation with gangue minerals up to about this pH.

Surpassing all the above possible causes there was a grave fault, unfortunately, in the way Johnston carried out the above experiments on the superpanner. While mentioning the importance of keeping the
conditions on the superpanner as constant as possible, he (6) argued that the slope of the superpanner must be gradually given a tilt from horizontal in order to keep the fluidity of the viscous wash water. The maximum slope he reached was 1 in 5 (11.30°). To quote his own words - "The slope of the superpanner was initially horizontal and a tilt was gradually given to the bed reaching a maximum at the end of a test. The slope used was one which gave visually the best concentration, with a viscous wash water the slope had to be increased to keep fluidity, but never exceeded 1 in 5."

Thereby, he assumed, without a proper base, that the pH influence on apparent viscosity would be significant at the shear as that which would prevail on the superpanner also. In the superpanner experiments, with the cam rotated at over 250 rpm (or 500 reversals of direction every minute) and with a relatively thin film the shear should have been much higher. The way Cosio carried out his experiments is not known. However, it may be safely assumed that he followed the same pattern as did Johnston for there is extensive reference to Johnston's work in his report.

The adverse effects of the slope on very fine particles are clearly demonstrated in several recent papers (10,11,14,15). In any slime gravity equipment the slope hardly exceeds 2°.

Finally, in reiterating his students' findings, Michell (9) wrote that the relationship between viscosity and solids concentration as found by his students, was in good agreement with that found by Norton, Johnson and Lawrence (3) and obeyed their proposed relationship:

\[ n_s = n_l (1-C) + K_1 C + K_2 C^n \], where

\( K_1, K_2 \) and \( n \) are constants

\( C \) is the volume concentration

\( n_s \) and \( n_l \) are the suspension and liquid viscosities respectively.

Though the above equation is based on hydrodynamic relations, the constants are entirely empirical and are applicable to the shear rates used in the study only (maximum of 30.1/s which is probably very low
because of the large annulus -5.3 mm- of the viscosimeter used). The same relationship need not hold at shear rates such as those that would prevail in slime gravity equipment.

Apart from the above cited works, there has hardly been any work on the apparent viscosity of dilute mineral suspensions. It also appears that the above works have greatly shaped the thinking of those who are involved in gravity concentration. Therefore, it is decided that it is appropriate to study the degree of influence of apparent viscosity of slime suspensions in detail since it can bring forth a new trend of thinking.

**SELECTION OF VISCOSIMETER**

The versatile Bohlin viscosimeter (18) was used for the dynamic flow characteristic studies. The reasons are:

a. the annulus of the CC2 cylinder and bob system of the Bohlin viscosimeter is only 1.5 mm in width, equivalent to that of the flowing films generally encountered in slime gravity equipment (15,16);
b. it operates in the range of shear rates prevailing in slime gravity equipment;
c. the flow pattern in the viscosimeter is laminar in the range of shear rates at which the flow pattern in practice is also essentially laminar.

In this rotational viscosimetry system, the shear stress of the fluid rotated at constant angular velocity is imparted to a suspended bob which transmits the torque to a torsion element. This torque is interpreted by a linear variable differential transformer (LVDT) through an arm attached to the torsion element. Viscosity is a function of the change in the generated voltage.

The instrument is easy to operate since it is computer controlled entirely and the flow behaviour is recorded on a printer with the sample detail, the temperature, the measuring geometry etc. The measuring program is controlled by the alphanumeric keyboard. The whole program can be followed on a display by the operator. Using a regression system a mathematical model of a linear form is also
possible with a plot on the printer.

The Bohlin viscosimeter gave reproducible results on the standard oil with deviations up to about 7%, over a wide range of shear rates. With the thixotropic mineral suspensions containing high amounts of ultrafine particles the reproducibility was within the range of 5-20%. The smaller the viscosity measured, the higher the deviation. With a thixotropic mineral suspension such as the slime from Geevor plant, Cornwall settling of the relatively small amounts of coarser particles is absent at low rates of shear, due to the prevailing high apparent viscosities.

At intermediate and high rates of shear, rotation prevented the settling greatly since the particles had the tendency to settle along a helical path. The slope of this path is determined by the effective particle weight, rate of rotation and the prevailing apparent viscosity. Also, at the intermediate and high rates of shear ring vortices can form. These vortices, which may spin alternately in both directions, could prevent sedimentation without causing the total flow in the suspension to become turbulent.

Moreover, any small amounts of settling within permissible limits such as in the case of Geevor slime, could not have changed the average torque transmitted to the relatively tall bob of the CC2 system of the Bohlin viscosimeter.

TEST MATERIALS AND PREPARATION OF SAMPLES

Schack et al. (4) used model minerals and an amorphous glass material in their attempt to estimate the influence of viscosity in mineral processing. Johnston (6) also made his studies to a greater extent on synthetic samples. Also, it is realised that the results of any dynamic flow characteristic studies made on appropriate natural materials would be of greater interest to mineral industries, than any attempts made to ascertain the influence of extraneous factors with the results obtained by analysing synthetic samples. Furthermore, Johnston concluded that whether the suspension is a naturally occurring slime or not makes little difference to flow behaviour.
Therefore, it is decided to base the study on natural materials and to improve and analyse the study according to the above two works if necessary.

Not deslimed (i.e. in the presence of ultrafine particles) natural samples with a top size of 38 µm were chosen from the slime circuits and tailings of various plants in Europe - The Geevor slime plant, Cornwall (U.K.), Kemi (Finland) chromite slime circuit and Yxsjöberg (Sweden) scheelite tailings. The samples selected are representative of different varieties of minerals and have different properties such as surface characterisitics, solubility, friability, thixotrophy etc.

The particle size distributions of the test materials studied are given in Table I and Table II. The approximate mineral distributions are given in Table III, Table IV and Table V.

Table I. Particle size distribution(by micro-mesh sieves).

<table>
<thead>
<tr>
<th>Sieve range µm</th>
<th>%-weight</th>
<th>Geevor</th>
<th>Kemi (I)</th>
</tr>
</thead>
<tbody>
<tr>
<td>-38</td>
<td>11.10</td>
<td>14.82</td>
<td></td>
</tr>
<tr>
<td>-20</td>
<td>16.48</td>
<td>11.31</td>
<td></td>
</tr>
<tr>
<td>-10</td>
<td>12.38</td>
<td>6.28</td>
<td></td>
</tr>
<tr>
<td>-5</td>
<td>60.04</td>
<td>67.59</td>
<td></td>
</tr>
</tbody>
</table>

Table II. Particle size distribution(by micro-mesh sieves).

<table>
<thead>
<tr>
<th>Sieve range µm</th>
<th>%-weight</th>
<th>Yxsjöberg</th>
<th>Kemi (II)</th>
</tr>
</thead>
<tbody>
<tr>
<td>-38</td>
<td>46.01</td>
<td>47.38</td>
<td></td>
</tr>
<tr>
<td>-20</td>
<td>27.16</td>
<td>29.89</td>
<td></td>
</tr>
<tr>
<td>-5</td>
<td>26.83</td>
<td>22.73</td>
<td></td>
</tr>
</tbody>
</table>
Table III. Geevor slime tailing Analysis
(-38 μm - sieve diameter)

<table>
<thead>
<tr>
<th>Oxide</th>
<th>Weight %</th>
<th>Mineral</th>
<th>Weight %</th>
</tr>
</thead>
<tbody>
<tr>
<td>SiO₂</td>
<td>57.5</td>
<td>Al₂ SiO₅ (OH)₄ (kaolinite)</td>
<td>29.7</td>
</tr>
<tr>
<td>Al₂O₃</td>
<td>19.3</td>
<td>SiO₂ (silica)</td>
<td>27.8</td>
</tr>
<tr>
<td>Fe₂O₃</td>
<td>12.8</td>
<td>H₂ KAl₂ (SiO₃)₄ (mica)</td>
<td>19.6</td>
</tr>
<tr>
<td>K₂O</td>
<td>4.6</td>
<td>Fe₂O₃ (silica)</td>
<td>12.8</td>
</tr>
<tr>
<td>CaO</td>
<td>1.5</td>
<td>1/2 (CaO₄) (MgO)₄ (bentonite)</td>
<td>10.2</td>
</tr>
<tr>
<td>TiO₂</td>
<td>0.6</td>
<td>SnO₂</td>
<td>0.704</td>
</tr>
<tr>
<td>Sn</td>
<td>0.55</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cu</td>
<td>0.164</td>
<td></td>
<td></td>
</tr>
<tr>
<td>W</td>
<td>0.016</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
Table IV. Yxsjöberg (Sweden) old scheelite tailing analysis (-38 μm - sieve diameter)

<table>
<thead>
<tr>
<th>Oxide</th>
<th>Weight %</th>
<th>Mineral</th>
<th>Weight %</th>
</tr>
</thead>
<tbody>
<tr>
<td>SiO₂</td>
<td>32.85</td>
<td>Skarn minerals</td>
<td>37</td>
</tr>
<tr>
<td>CaO</td>
<td>23.50</td>
<td>SiO₂</td>
<td>32.85</td>
</tr>
<tr>
<td>Fe₂O₃</td>
<td>19.71</td>
<td>CaF₂</td>
<td>21</td>
</tr>
<tr>
<td>Al₂O₃</td>
<td>6.20</td>
<td>CaCO₃</td>
<td>5</td>
</tr>
<tr>
<td>S</td>
<td>3.30</td>
<td>CaWO₄</td>
<td>1.005</td>
</tr>
<tr>
<td>MgO</td>
<td>2.19</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Na₂O</td>
<td>1.42</td>
<td></td>
<td></td>
</tr>
<tr>
<td>K₂O</td>
<td>1.14</td>
<td></td>
<td></td>
</tr>
<tr>
<td>WO₃</td>
<td>.81</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
Table V. Kemi slime (II) Analysis
(-38 μm - sieve diameter)

<table>
<thead>
<tr>
<th>Oxide</th>
<th>Weight %</th>
<th>Mineral</th>
<th>Weight %</th>
</tr>
</thead>
<tbody>
<tr>
<td>SiO₂</td>
<td>37</td>
<td>Mg₂SiO₃(OH)₃</td>
<td>32</td>
</tr>
<tr>
<td>MgO</td>
<td>30</td>
<td>MgSiO₃(OH)</td>
<td>29.5</td>
</tr>
<tr>
<td>CrO₂</td>
<td>16.08</td>
<td>Mg₂SiO₃(OH)</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>(serpentine)</td>
<td></td>
</tr>
<tr>
<td>FeO₂</td>
<td>10</td>
<td>Mg₂SiO₃(OH)</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>(serpentine)</td>
<td></td>
</tr>
<tr>
<td>Al₂O₃</td>
<td>6</td>
<td>(Fe,Mg)(Cr,Al)O₂</td>
<td>38.5</td>
</tr>
<tr>
<td></td>
<td></td>
<td>+ (Mg,Fe)((Cr)Al)O₂</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>(chromite and Cr bearing chlorite)</td>
<td></td>
</tr>
</tbody>
</table>

The Geevor slime was studied in detail, at different slime concentrations and pH in view of the presence of large amounts of ultrafine particles—exclusively clays. Also, the Kemi 'slime (I)' was studied in detail since it also contained a very high amount of ultrafine particles and is a representative of materials containing ultrafine particles other than clay.

In addition, flow properties of the montmorillonite group of clay, bentonite, were also considered because of the clay's water swelling property and its presence in the Geevor slime (about 10%). Bentonite is (Ca,Mg)₂SiO₃(Al,Fe)₂O₃ with trivalent Al replacing tetravalent Si and divalent Mg replacing trivalent Al occasionally in the tetrahedral sheet and octahedral sheet of the 2:1 layers of the prototype mineral respectively. This is possible owing to the small sizes of the replacing atoms which result in an excess of negative charge. In the presence of water these clays permit penetration of water molecules between the layers, leading to swelling. This could be seen by the increase in basal spacing of the clays which can lead to an increase
of the order of 2 times in the volume of dry clay (12).

The Geevor slime was gently dispersed in a disk mill and stored. Samples for viscosity studies were first screened through a 38μm sieve in batches to remove the oversize and organic materials and the slurry containing the undersize was dried at about 85-95°C, without decantation. The dried samples were gently dispersed in a pestle and mortar with a little water and kept in a closed cabinet at room temperature and allowed to dry thoroughly. The required amounts were weighed and made into suspensions of desired pulp densities. This is achieved by adding deionized water at room temperature and dispersing first in an ultrasonic bath and then by stirring for predetermined periods at required pH levels. The pH was adjusted by adding either HCl or NaOH. The pulp densities studied were 15%, 17.5%, 25%, 27.5% and 40% solids.

Kemi slimes and Yxsjöberg scheelite tailing were also screened at 38 μm and the undersize portions were dried and stored. The procedure for the Geevor slime was adopted here also to prepare the suspensions of required pulp densities, except the volume of the suspension and method of stirring.

It was found during preliminary studies that the Kemi 'slime (II)' and Yxsjöberg tailing tended to settle down very quickly. Nearly three-quarters of both the materials were coarser than 5 μm. The apparent viscosities of these materials observed during preliminary studies were comparable with that of Schack et al. (4) (Fig. 2) and Green, Lamb and Taylor (13) (Fig. 3). Therefore, it was decided to avoid low concentrations where settling was severe. It was possible with Bohlin's viscosimeter to have a very high rate of shear (1160/s (1054 rpm)) where complete turbulent conditions are prevalent in order to bring the particles in suspension by turbulent eddies before a set of measurements. Also, the time of duration of measurements was kept short (3 sec). The apparent viscosities of Kemi 'slime (II)' and Yxsjöberg tailing at intermediate and high concentrations were measured at three rates of shear only (the cycle used is 29.1, 99.5, 340, 1160, 340, 99.5 and 29.1/s).
Standard samples of bentonite in distilled water at different pulp densities were available at the Division of Soil Mechanics of the above University.

The CC2 cylinder and bob system was used throughout the experiments except when bentonite was studied. The distance of the bob from the bottom of the cup was kept constant (Fig. 4).

The programming gives uniform acceleration of the rotating cylinder for the full range of shear rates (0.7/s to 1160/s). Measurements were made with different starting shear rates and different time intervals.

The shear history was kept constant as much as possible before measurements since difficulties were encountered with the Geevor and Kemi slimes because of their thixotropic behaviour, where rheological properties of the system depend on the previous shear history. Shear in pumping or cycloning reduces the apparent viscosity and the yield stress greatly and since the formation of gelling slurry structure takes time, the slime remains disrupted with broken linkages, when fed onto a slime gravity concentrator.

Further, it was seen during preliminary experiments that the influence of the shear history becomes less significant with increasing shear rates. Likewise, conditioning time was also found to cause only slight deviations in the apparent viscosity.

EXPERIMENTS

The results of the studies on the dynamic flow characteristics of the Geevor slime at pulp densities 15%, 17.5%, 25%, 27.5% and 40% solids by weight are summarized in Figs. 5 and 6. The experimental temperature was 17°C and each reading was taken four times and averaged. The shear rate was first increased to 628/s from 29/s and then brought back to 29/s. The averaged readings of the 'down' part of the above measuring cycle are only reported in Figs. 5 and 6. This is analogous with the actual situation in practice where the shear in pumping or cycloning reduces the apparent viscosity and the yield stress greatly and the slime remains sheared when fed onto a slime
gravity concentrator. Further, the slime takes time to develop its former gelling slurry structure.

Pulp densities between 15 and 25% solids by weight and shear rates between 29/s and 340/s are of interest to slime gravity concentration. Results show a considerable range of behaviour. Except for some weakly developed non-Newtonian behaviour at low pH with low rates of shear which can be shown insignificant in slime gravity concentration, they behave almost as a Newtonian fluid at typical slime circuit concentrations (15% to 17.5% solids by weight) with extremely low apparent viscosities. The pH dependence is within about 1 cp only at these pulp densities under the conditions prevailing in slime gravity processes. At intermediate concentrations (25-27.5% solids by weight) suspensions show strong non-Newtonian behaviour. Also the pH effects become pronounced. Nevertheless, at rates of shear as those on the slime gravity equipment the pH influence becomes less significant and the apparent viscosity drops greatly. At high concentrations (40% solids by weight) both pH influence and apparent viscosity are high, at all shear rates.

The experiments also demonstrate the significant effect that the pH can have when the pulp is completely at rest or at extremely low rates of shear which is of theoretical interest only as far as the slime gravity methods are concerned. Such conditions, as can be shown, are never prevalent because of the additional shear velocities due to the flowing thin films and the non-stationary concentrating surfaces.

When the pulp pH was high, and the densities were low and intermediate a reduction in apparent viscosity, though very marginal, was obtained at decreasing rates of shear. Any fear that such settling might have led to erroneous interpretations is unjustifiable because even at an extremely low rate of shear a 10% kaolinite (75% <1 μm, 100% <10μm) gave a viscosity of only about 1.6 cp at high pH values. Also, a ball clay suspension of 10% solids (71% <5 μm, 84% <10 μm) gave a viscosity of about 1.5 cp only at the pH of 10 (6). More evidence is found in the work by Norton, Johnson and Lawrence (3) who showed that any considerable increase in the viscosity of deflocculated suspensions is possible only when centre-to-centre distance approaches the particle
diameter. The cause is the mechanical interference of the revolving or sliding particles which begins to gain importance as the distance between particles approaches the particle diameter.

Fig. 7 demonstrates the effects that the pH and pulp density can have on flow properties at individual rates of shear. Here, the apparent viscosity in centipoise is plotted against the percentage of solids by weight. The decreasing degree of influence of the pH on apparent viscosity at increasing rates of shear clearly demonstrates the limited influence of the pH on apparent viscosity under the shear conditions prevailing in slime gravity processes. Note that the maximum viscosity obtained at typical slime circuit feed densities, pH and rates of shear is less than 3 cp only. The rapid increase in apparent viscosity is obtained around 25-27.5% solids by weight.

The non-Newtonian behaviour of bentonite (Fig. 8) explicitly demonstrates the difference the shear can make. A near-sol of 9% bentonite (85% -2µm) drops from nearly 2000 cp to 300 cp (a drop of 85%) when the rate of shear is increased to 184/s from 21.4/s. A sol containing only 5% bentonite falls to 60 cp from 350 cp - a fall of 83%. A suspension of 40% Geevor slime by weight contains approximately 4% bentonite, yet the apparent viscosity is only about 20 cp at a shear rate of 184/s. This can well be explained by the distribution of particle sizes at which bentonite and other clay particles exist in the Geevor slime. The presence of coarser particles also has a marked effect by disrupting the gelling slurry structure by their passage through.

The results of the studies on Kemi 'slime (I)' at pulp densities 15, 17.5, 25, 27.5 and 40% solids by weight are summarized in Figs. 9-11. The experimental conditions were the same as that for the Geevor slime. It is well known that serpentine is an easily soluble mineral component of chromium ore and therefore the acid consumption was very high. This could result in high amounts of Ca, Mg, Al and SiO₂. The interactions among these dissolved specie and the resultant polymeric aggregates could influence the apparent viscosity of the suspensions. Therefore, the experimental results under acidic conditions are not reported because of the dissimilar conditions between acidic and
alkaline regions.

Despite the presence of nearly 67.6% of -5 \( \mu \)m particles, the apparent viscosity was hardly more than 2.5 cp at the standard slime circuit pulp density of 15% solids by weight. Therefore, it should be possible to treat the chromite slimes without desliming at the standard pulp density, in the absence of the fear of increased apparent viscosity. Also it is possible to increase the pulp solids to 17.5% solids by weight by a simple least dispersion (Fig. 9). This is a way to recover the valuable heavy minerals that are being lost unnecessarily due to desliming because of the unfounded fear of increased apparent viscosities. The small amount of extra gravity equipment needed to treat the undeslimed Kemi chromite which is of a comparatively high grade, at standard slime circuit pulp densities, should easily be repaid by extra overall recovery.

The behaviour of the Kemi 'slime (I)' is similar to that of Geevor slime. At low concentrations (15-17.5% solids by weight) it possesses very low apparent viscosities. At 40% solids by weight both pH influence and apparent viscosity are high. Intermediate suspensions behave in an intermediate way.

The Geevor slime because of its very high clay content, and also the Kemi 'slime (I)' are two of the worst slimes but yet with economical significance. Therefore, the conclusions based on the study of these two natural slimes should be sufficient to the mineral industry. The main conclusion that arises is that the natural slimes containing very high amounts of ultrafine particles show only a very limited viscous behaviour and therefore it should be possible to tolerate a substantial amount of such particles in the slime circuit feeds at standard pulp densities. Also, the use of dirty hutch water (which can have only about 5% by weight of clayey matter under very bad conditions (5)) in jigging cannot be significant enough to affect the efficiency of the process by appreciably retarding the penetration rates of the particles.

Apparent viscosities of Kemi 'slime (II)' and Yxsjöberg tailing hardly exceeded 2 cp at the intermediate pulp density, at alkaline pH.
Because of the very high acid consumption measurements were not done in an acidic environment.

At the high solids content (40% solids by weight) the Kemi 'slime (II)' was found to peptise with increasing alkalinity but the Yxsjöberg tailing failed to show any detectable tendency to peptise. The apparent viscosity dropped from a maximum of 4.82 cp at an alkaline natural pH to a maximum of 2.51 cp when the pH was increased to more than 10, at the measured shear rates of 29.1, 99.5, and 340/s. The apparent viscosity values obtained for Kemi and Yxsjöberg slimes fitted well into the model of Schack et al. for apparent viscosities (Fig. 2).

DISCUSSION

For a particle kept on a smooth inclined surface over which a thin film of fluid flows, the following relationship (2.14.20) can be developed for the minimum angle below which motion by sliding cannot occur, by balancing the forces acting on it:

\[
\cot \theta_{crit.} = \frac{(9/2)k \left( \frac{d'}{d_s - d'} - \frac{x}{r} \right) + (27/8)k \left( \frac{d'}{d_s - d'} \right) + 1}{\mu_s}
\]

where
- \( d' \) is the fluid density
- \( d_s \) is the particle density
- \( x \) is the film thickness
- \( r \) is the particle radius
- \( k \) is a shape factor
- \( \mu_s \) is the coefficient of static friction.

From the above it can be seen that the critical angle increases with:

1. decreasing film thickness and hence rate of flow
2. increasing particle size and specific gravity.

Though the above relationship does not directly depend on the
apparent viscosity of a fluid the thin film thickness $x$ depends on the apparent viscosity:

$$x = 3 \sqrt[3]{\frac{3 \eta W}{d' g \sin \alpha}},$$

where

- $\eta$ is the fluid viscosity
- $W$ is the rate of flow.

Any increase in the apparent viscosity requires a corresponding increase in the slope to keep the film thickness constant. This is best explained by the following numerical example:

When $\eta = .015$ units and $\alpha = 1^\circ$ and therefore when $\eta = .05$ units $\alpha = $ becomes equal to $3.34^\circ$.

The critical angles for high density particles (ex. cassiterite) of 5-20 um in size, fall between $1^\circ$ to $2^\circ$ under the conditions prevailing in slime gravity concentration. This could also be seen by a numerical example:

Assume $\eta_s = 1/2$, $d' = 1$, $k = 4/3$, $d_s = 7$ and $x/2r = 50$, therefore $\cot \alpha_{\text{crit.}} = 150.75$, $\alpha_{\text{crit.}} = 0.38^\circ$.

Allowing for the interference of surrounding particles and bed or concentrating surface roughness it may be safely assumed that the critical angle is around $1-2^\circ$ or less, a conclusion well-augmented in practice. An increase beyond this slope will cause the heavy particles to slide. Therefore any experiments carried out on both sides of the critical angle, like the attempts by Johnston and probably Cosio as well, to establish a correlation between viscosity and efficiency of concentration are not valid. They are carried out under totally different conditions. Under such circumstances the adverse effects of the critical angle must be taken into account. Any models based on such experiments are not acceptable since the conditions are not kept constant.

Johnston carried out his experiments by having successively increasing slopes with decreasing pH and the argument was that with a viscous wash water the slope had to be increased to keep fluidity. The maximum
slopes Johnston reaches was $1$ in $5$, which is far in excess of the critical angles obtained above.

In order to strengthen the above deduction, a set of simple experiments were carried out on a superpanner. One gram of cassiterite particles between $5$ and $20$ $\mu m$ in size was treated on the superpanner at constant conditions for two minutes each with $17$ ml of water at slopes of $1.59^\circ$, $3.18^\circ$ and $4.76^\circ$. The cassiterite particles left up to the distance of $10$ cm from the point of entry were collected. However, at the slope of $4.76^\circ$ no material was left (Fig. 12).

The rates of shear in the CC2 system of the Bohlin viscosimeter are not essentially constant but vary within narrow limits because of the relatively large diameters of the cylinders and small annulus. The velocity of the different layers of the thin film on a symmetrically reciprocating smooth deck decreases with the distance across the film from the deck rather rapidly. However, it must be stressed that the velocity-depth relationship greatly depends on the equation describing the motion of the deck which is

$$f(t) = V_0 \cos 2\pi ft \quad \text{at } t \geq 0,$$

for the simple harmonic motion. The velocity-depth relation was derived mathematically in the author's review work on tabling (20) which clearly shows the dependence of the velocity of different layers on the equation describing the motion of the deck.

For example, if the B-M concentrator is considered the relationship describing the oscillating motion of this concentrator has to be different from that of the simple harmonic motion for it cannot be a trigonometric function depending on time. Here the shear stress produced due to the oscillating motion is continuous and this is the very idea of having an oscillating motion instead of a reciprocating motion where the shear produced is discontinuous.

The next consideration is the velocity distribution of a thin film of fluid flowing down a smooth inclined surface which is parabolic under laminar flow conditions, increasing from zero at the deck to a maximum
at the top:

\[ V_y = \frac{(d' \cdot g \sin \alpha) (2x - y) y}{2\eta} \]

Under typical operating conditions of the B-M concentrator, where \( \alpha = 1.75^\circ \), \( d' = 1 \), \( x = 1 \) mm and \( \eta = .015 \) poise, \( V_x \) becomes equal to 9.99 cm/sec (Fig. 13) which corresponds to an angular velocity of 63.6 rpm of the CC2 system of the Bohlin viscosimeter. Note from Fig. 13 that the speed of the layers down to nearly half the depth is about the top layer speed. An increase of viscosity to 0.03 poise will reduce the value of \( V_x \) to 5 cm/sec corresponding to an angular velocity drop to only 31.8 rpm of the CC2 system.

As stated above, when a thixotrophic suspension is fed onto the deck of a slime gravity equipment, it is at a highly sheared state and possesses very low apparent viscosity. Since the suspensions take time to develop their gelling slurry structure, they will be able to develop the laminar flow with layer velocities determined by the above initial very low apparent viscosity. These initial flow velocities of the top layers will tend to keep the apparent viscosities low, at corresponding layers by keeping the particle structure disrupted.

Moreover, the motion of the deck will always have the tendency to counteract any increase in the apparent viscosities at the top layers due to the gelling slurry structure by correspondingly increasing the fluid velocities at these layers (Table VI). For example, at a viscosity of 3 cp the topmost layer of 1 mm thick film have \( V_y/V_0 \) ratio of 0.18 at a deck speed of 300 spm, which should be sufficient to give a very low apparent viscosity of 2 cp or less at typical slime circuit conditions (pH and pulp density).

With a deslimed feed such as Burt (16) used in his attempt to estimate the influence of different concentrating surfaces at different pH the concentration profile over the depth can lead to a large reduction at the upper layers due to the decreased concentration of solids. The reduction in apparent viscosity due to settling solids was noticed with the Geevor slime also at low concentrations. A mathematical deduction is possible employing a diffusion approach to quantify the
concentration profile over the depth for a turbulent flow (19). Such an approach may be possible for a laminar flow as well.

Burt argued in the above particular work that the increased viscosity at low pH was the cause for the drop in efficiency. However, it is found in my experiments with Geevor slime, which contains more than half the weight of clays, that the pH dependence was within about 1 cp only even at 15% solids by weight. The experiments with the Kemi and yxsjöberg slimes also showed that the apparent viscosity was hardly above 2 cp even at 25% solids by weight. Burt used a feed of 10% solids by weight only in his experiments. He neglected the possibility of heteroaggregation at low pH values which was really the cause for the observed reduction in efficiency. Mills (15) also discussed the insignificance of viscosity.

The disadvantage of the shaking table is the differential motion of the deck which is very acute at the end of the forward stroke and can cause a certain degree of turbulence which is detrimental to the efficient recovery of slimes, particularly of very fine particles. A thin film on a deck with riffles instead of a series of planes will experience more turbulence due to hindrance to flow by these riffles. Under such circumstances a considerable portion of the shear is caused by the turbulent momentum transfer also. Flow velocities of the upper layers can be considerably higher due to the thicker films (upto 1.5 - 2 mm) used usually in tabling (Fig. 13).

From the equation obtained for the velocity-depth relationship of the simple harmonic motion of a deck, certain conclusions are made.

\[ V_y = V_0 e^{-y \frac{\pi f d'}{n}} \cos \left( 2\pi f t - y \frac{\pi f d'}{n} \right) \]

If an analysis is made numerically:

assume \( f = 300 \) spm, \( d' = 1 \) and \( n = .01 \)
then $\sqrt{\frac{\pi d'}{n}} = 39.63$ and

$$V_y = V_o e^{-39.63 \ y \ \cos(2\pi ft - 39.63 \ y)}$$

when $n = 0.02$

$$\sqrt{\frac{\pi d'}{n}} = 28.02$$

when $n = 0.5$ and $d' = 1.6$

$$\sqrt{\frac{\pi d'}{n}} = 7.09$$

From these the ratios of $V_y/V_o$ can be calculated for various increasing distances from the deck (Table VI).

Table VI. $V_y/V_o$ ratio as a function of depth (when $f = 300$ rpm)

<table>
<thead>
<tr>
<th>Distance from deck</th>
<th>$V_y/V_o$ for $n = 0.01$</th>
<th>$V_y/V_o$ for $n = 0.02$</th>
<th>$V_y/V_o$ for $n = 0.5$</th>
</tr>
</thead>
<tbody>
<tr>
<td>1 mm</td>
<td>0.019</td>
<td>0.06</td>
<td>0.59</td>
</tr>
<tr>
<td>750 µm</td>
<td>0.05</td>
<td>0.12</td>
<td>0.70</td>
</tr>
<tr>
<td>500 µm</td>
<td>0.14</td>
<td>0.25</td>
<td>0.87</td>
</tr>
<tr>
<td>200 µm</td>
<td>0.45</td>
<td>0.57</td>
<td>0.97</td>
</tr>
<tr>
<td>50 µm</td>
<td>0.82</td>
<td>0.87</td>
<td>0.99</td>
</tr>
<tr>
<td>10 µm</td>
<td>0.96</td>
<td>0.97</td>
<td>0.99</td>
</tr>
</tbody>
</table>

For a liquid of viscosity of 0.02 poise, layers up to 500 µm are sheared considerably. It is extremely important to remember that there is a very high pulp density near the deck due to the settling of solids. Under steady conditions particles can be expected to occupy the bottom portion up to a height of about 200 µm at solids concentration as high as 35% by volume (60% solids by weight) which can give rise to a viscosity of about 20-100 cp. Under such conditions it can be concluded
a. a layer of less densified suspension upto a height of about 50 μm or more on top of the settled particles will move with the deck;  
b. layers of suspension upto about 500 μm will experience a greater part of the shear produced by the reciprocating shaking table.  
c. layers above that (750 μm) will be sufficiently sheared due to their own flow velocities. From Fig. 13 it can be seen that an increase in the thickness of the film results in an increase in velocity.  

On a Duplex concentrator the shear rates of the deck are between 90-110 rpm only. And also as in any thin film concentrators the flowing film velocities can be 10 cm/sec or even more, down to about half the depth from the top. Therefore the apparent viscosities cannot be more than about 2 cp over a great distance from the top as well as from the bottom.  

This could leave a small layer in the intermediate area of the flowing film with increased apparent viscosities. But in the relationship between velocity and depth of the simple harmonic motion:

\[
V_y = V_0 e^{-y \sqrt{\frac{\pi f d'}{n}}} \cos \left(2\pi f t - y \sqrt{\frac{\pi f d'}{n}}\right)
\]

\[
\sqrt{\frac{\pi f d'}{n}} \text{ becomes equal to } 15.35
\]

when \( f = 90 \text{ rpm} \) \( d' = 1 \) \( n = 0.02 \).  

It may be noted from Table VII that the \( V_y/V_0 \) ratio is about 0.46 at 500 μm in contrast to 0.25 obtained for a shaking table.
Table VII. $V_y/V_o$ ratio as a function of depth (when $f = 90$ rpm)

<table>
<thead>
<tr>
<th>Distance from deck</th>
<th>Relative velocity $V_y/V_o$</th>
</tr>
</thead>
<tbody>
<tr>
<td>1 mm</td>
<td>0.32</td>
</tr>
<tr>
<td>750 μm</td>
<td>0.46</td>
</tr>
<tr>
<td>500 μm</td>
<td>0.74</td>
</tr>
<tr>
<td>200 μm</td>
<td>0.93</td>
</tr>
<tr>
<td>50 μm</td>
<td>0.98</td>
</tr>
<tr>
<td>10 μm</td>
<td></td>
</tr>
</tbody>
</table>

With the cleaner circuit equipment such as Bartles XB concentrator the problem of the presence of ultrafines does not arise.

Finally, it is interesting to note that Batzer (8) as early as 1965 found that the presence of slime in the feed and make-up water had only a small adverse effect upon jig recovery unto a slime (-300 mesh) content of about 20%.

CONCLUSIONS

1. With increasing viscosity the critical angle for a particular mineral decreases and therefore the slope has to be increased to maintain the critical angle constant. However, any increase in slope beyond the critical angle for the mineral under consideration leads to totally different operating conditions and hence any model obtained under such differing conditions is incorrect.

2. At low pH it is heteroaggregation that causes the efficiency drop with deslimed feeds and not the apparent viscosity.

3. Shear on any slime gravity equipment can be estimated by selecting
a suitable viscosimeter operating in that range of shear rates.

4. Any viscous nature of dilute suspensions of slimes is due to the presence of extremely high amounts ultrafine to colloidal particles such as clays only, under the conditions prevailing in slime gravity processes.

5. The effect of apparent viscosity is minimal and hardly dependent on the pH under the conditions prevailing in slime gravity processes, even with a presence of about 55% -5 μm clay particles at a concentration of 17.5% solids by weight.

6. Except for some weakly developed tendency to gel at low pH with low rates of shear which is insignificant in slime gravity processes, they behave almost as Newtonian fluids with an extremely low apparent viscosity at typical slime circuit concentrations (15- 17.5% solids by weight).

7. At intermediate concentrations suspensions show strong gelling behaviour and also, the pH effects become pronounced. Nevertheless at shear rates as those on the concentrating equipment pH influence becomes less significant and the apparent viscosity drops greatly.

8. The presence of a very high amount of non-clay ultrafine to colloidal particles also makes a suspension gel - however, the viscous behaviour is negligible under the conditions such as that would prevail in slime gravity processes.

9. Ultrafine removal is necessary only when extremely high amounts of such particles are present.

10. However, any removal of particles coarser than about 2-3 μm prior to concentration by slime gravity equipment, is superfluous and will certainly result in unnecessary losses of valuable minerals. Short diameter high pressure hydrocyclones for such removal can be detrimental, particularly with friable minerals and therefore thickeners are recommended.
ACKNOWLEDGEMENT

The authors wish to thank Ms. Kerstin Pousette of the Division of Geoteknik, Technical University of Luleå, for the use of and help with Bohlin's viscosimeter and also for providing the results which appear in Fig. 8.
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Fig. 1  Relationship between pH, apparent viscosity and concentration efficiency of ferrosilicon from quartz in a 10% ball clay suspension (after Johnston (6))
Fig. 2 Apparent viscosity vs. pulp solids (weight percent) for mineral composites and apparent viscosity vs. particle size of quartz at various pulp solids (weight percent) (after Schack, Dean and Molloy (4)).
Fig. 3 Apparent viscosity vs. pulp solids (volume percent) at various particle sizes (after Green, Lamb and Taylor (13))
Fig. 4 The CC2 cylinder and bob system

\[ H = 37.6 \text{ mm} \]
\[ D_i = 27.0 \text{ mm} \]
\[ D_y = 30.0 \text{ mm} \]
\[ \alpha = 10^\circ \]
Fig. 5A Apparent viscosity vs. shear rate for Geevor slime at various pulp solids (weight percent)
Fig. 5B Apparent viscosity vs. shear rate for Geevor slime at various pulp solids (weight percent)
Fig. 5C Apparent viscosity vs. shear rate for Geevor slime at various pulp solids (weight percent)
Fig. 6A Apparent viscosity vs. shear rate for Geevor slime at various pulp solids (weight percent)
Fig. 6B Apparent viscosity vs. shear rate for Geevor slime at various pulp solids (weight percent)
Fig. 6C Apparent viscosity vs. shear rate for Geevor slime at various pulp solids (weight percent)
Shear rate 340/s

- O pH 3-4
- X pH natural
- △ pH 10-11

Fig. 7A Apparent viscosity vs. pulp solids (weight percent) for Geevor slime at various pH
Fig. 7B Apparent viscosity vs. pulp solids (weight percent) for Geevor slime at various pH
Fig. 7C Apparent viscosity vs. pulp solids (weight percent) for Geevor slime at various pH
Fig. 7D Apparent viscosity vs. pulp solids (weight percent) for Geevor slime at various pH
Fig. 8 Shear thinning of bentonite (85% -2 μm)
Fig. 9A Apparent viscosity vs. shear rate for Kemi slime at various pulp solids (weight percent)
pH 10-11

Fig. 9B Apparent viscosity vs. shear rate for Kemi slime at various pulp solids (weight percent)
Fig. 10A Apparent viscosity vs. shear rate for Kemi slime at various pulp solids (weight percent)
Fig. 10B Apparent viscosity vs. shear rate for Kemi slime at various pulp solids (weight percent)
Fig. 11A Apparent viscosity vs. pulp solids (weight percent) for Kemi slime at various pH

Shear rate 340/s

X = pH natural
Δ = pH 10–11
Fig. 11B Apparent viscosity vs. pulp solids (weight percent) for Kemi slime at various pH
Fig. 11C Apparent viscosity vs. pulp solids (weight percent) for Kemi slime at various pH
Fig. 11D Apparent viscosity vs. pulp solids (weight percent) for Kemi slime at various pH
Fig. 12 Influence of slope
Fig. 13 Velocity-depth relationship of thin film (in films of 2.0, 1.5 and 1.0 mm deep at a viscosity of 1.5 cp and for a slope 1.75°)
Roughness profiles - a way to recover heavy minerals from slimes

Submitted for publication in the Scandinavian Journal of Metallurgy

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Abstract

Various concentrating surfaces are used in slime gravity processes. However, a proper basis for the selection has not yet been laid down. There have been a few attempts to attribute the performance of different types of surfaces to the electrokinetic potentials of those surfaces. Effects of the surface roughness which can have considerable roles on typical slime circuit feeds were neglected.

Surface roughness profiles of different concentrating surfaces are obtained using an equipment employing the "turn-table" technique. The surface roughness profiles obtained suggest the considerable influence it can have in slime treatment. Falls and dips as deep as 4-5 µm with mouths as wide as 25-40 µm are quite common on fibreglass surface whereas on stainless steel it is only .25-.50 µm deep. Wood has the deepest and the widest patterns of profiles.

Experiments carried out, to determine the effect of roughness profiles of varying dimensions, on synthetic and natural slimes confirm that concentrating surfaces with distinct roughness profiles can offer significantly better results than a strictly smooth surface, such as the stainless steel used.

The influence of roughness is discussed and profile patterns based on that obtained for the wood are suggested. Also, the ability of the electrical double layer forces of a concentrating surface to have any significant influence on typical slime feeds is questioned.
INTRODUCTION

"The problem is analytically complex and beyond the scope of this text; it is nevertheless a problem that might well be explored further if a full insight is desired into mechanisms of flowing film concentration." - so wrote Gaudin (1) in "Principles of Mineral Dressing" - one of the first text books to appear (1939) on the science of Mineral Processing - on the effect of deck roughness.

There were a few attempts to make use of the roughness of concentrating surfaces. At Anaconda Copper Co. cement-concrete decks, trowelled to a finish approximately that of the canvas, were tested against the performance of planed wood, linoleum and medium weight canvas. Rough-finish and smooth-finish cement-deck revolving tables were also compared at Anaconda. It was found that when using rough-surfaced cement and canvas-covered decks, solids built up as much as 3/8" (9.4 mm) and the rough deck gave better protection for sulphides (2). One way of preparing a rough cement surface was to dress a 3" (76.2 mm) to 6" (152.4 mm) thick concrete with a 0.75" (19.05 mm) layer of 1:1 cement mortar carefully trowelled, which when nearly dry, was dusted with dry cement and scored lightly transversely with a wire comb with the teeth at 1/8" (3.18 mm) intervals (2).

Fluted- and corrugated-glass surfaces were tested against wood on revolving round tables in the Cornish Tin Mines. The best results were obtained with about 16 flutes to an inch (25.4 mm), 1/32 to 1/16 in. (705 μm to 1.6 mm) deep, with crests either sharp or rounded (2).

The nature of the particle size distribution of the feed tested and the concentrate obtained was unknown in the above attempts. However, it is known that, although the revolving tables gave satisfactory
results with slime size particles, they failed to recover particles finer than $-20 \mu m$ (1). Also, the profile patterns tried were relatively coarser. Coarser profile patterns can act as minute riffles, to particles coarser than $-20 \mu m$, in which such particles settle and are protected from the sweep of the current. The particles finer than $-20 \mu m$ are likely to be swept upward into momentary suspension and be caused to move downslope by the strong eddies formed in such relatively deep depressions. This is precisely why the Holman table, which has an essentially smooth surface with pool riffles, is considered to be better in treating slimes than the Deister table, which consists of low concentrating riffles in between pool riffles (3).

However, successfully employing finer profiles to treat finer particles was neglected. Shallower profile patterns could have been tried to recover very fine particles, in the absence of the fear of causing strong eddies. It has always been found that wood gave the best results from the nominally smooth surfaces employed, to recover the heavies from slimes. The fact that the naturally occurring finer profiles on wood could have caused the improvement in the effectiveness of the concentrating processes in the extreme cases was not identified.

This summarises the stages of the development of the deck roughness. We should note more than 40 years has passed since Gaudin (1) made his comment on deck roughness, without any significant breakthrough.

As a result, perhaps it was noted in one of the latest text books on Mineral Processing as follows - "The roughness of the sloping surface is too complex to analyse".
INFLUENCE OF THE CONCENTRATING SURFACE ON SLIME GRAVITY CONCENTRATION

When we consider very fine particles, the roughness of the surface becomes more important than the coefficient of friction, in the case of commonly employed concentrating surfaces. If the roughness is of a magnitude considerably less than the particle size, it affects the coefficient of friction. Otherwise the shielding effect of the roughness profiles becomes more significant. Also, when the rolling motion of a particle is considered, the coefficient of friction becomes very small and can be neglected (4).

In the mid-1960's at the Camborne School of Mines, U.K., Michell and his students attempted to correlate the effectiveness of concentration from different kinds of concentrating surfaces with their electrokinetic potentials (5,6,7,8). The streaming potentials of a number of concentrating surfaces commonly employed in gravity concentration were measured using an apparatus based on that devised by Street and Stewart (9). The surfaces were found to have a negative potential over a wide range of pH values (Fig. 1). The above measurements and some other previous observations (6) led them to conclude that the effect of the electrokinetic potential of the concentrating surface is important. Since P.V.C. and linoleum have been found to possess much greater negative values than rubber and wood, the former two surfaces should have an adverse influence on the retention of mineral particles. They further stated that it might explain why wood was preferred by operators for extreme operations (6).

Later, in the late-1970's, Burt (10) compared the performance of different types of surfaces at different pH values and attempted to attribute the results to the electrokinetic potentials of those surfaces (Fig. 2). The test material and the test equipment used were a deslimed Cornish(U.K.) cassiterite ore and a pilot plant Bartles-Mozley separator.
The effect of surface roughness, which can have a considerable influence on typical slime feeds consisting of particles with a top size of about 45-53 μm by partly or fully shielding the very fine particles, was not taken into account. This effect could be visualised most readily if one would consider the operating conditions of Burt's (10) work. If the feed had an average size of about 20 μm (deslimed feed), there would only be about 8 layers of particles which emphasise the significance of the bottom layers (we should remember that there is a washing period in the operating cycle of the devices like Duplex concentrator and B-M concentrator, and the cycles can be effectively modified to treat the very fine particles fully exploiting the roughness of the concentrating surface). Also, the nature of the profile patterns of the surfaces should have been considered. For example, sharp tips of the recesses can retard the flow of the fluid.

Moreover it is appropriate to note on the distance upto which the field due to the charge on the concentrating surface would be significant and the net weight of the particles that the repulsive forces could counteract. Table 1 is sufficient to give a clear idea of the distance upto which the field is significant. The surface forces can act only at a few, very small, points of contact. Von Buzagh's (11,12,13,14) extensive experiments on the adhesion of small particles to plates of the same material, in water and electrolyte solutions, clearly show the e.d.l. forces' magnitude. It was found that when electrical double layer forces were sufficiently reduced, the shorter ranged London-van der Waals forces were strong enough to sustain the net weight of a 12 μm quartz grain. Conversely, at pH-10 and low ionic strength the repulsive forces could counteract the weight of such particles. At low pH levels and/or at intermediate to high ionic strengths it was not possible to keep such a particle in suspension.
Table 1: Approximate thickness of the electrical double layer as a function of electrolyte concentration at a constant surface potential

<table>
<thead>
<tr>
<th>Concentration</th>
<th>1/K (nm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Distilled water</td>
<td>-900</td>
</tr>
<tr>
<td>10^{-4} M NaCl</td>
<td>31</td>
</tr>
<tr>
<td>10^{-4} M MgSO₄</td>
<td>15</td>
</tr>
<tr>
<td>Sea water</td>
<td>-0.5</td>
</tr>
</tbody>
</table>

K - the Debye-Hückel reciprocal length parameter which for aqueous electrolyte solutions at 25°C is given by:

$$K = 3.29 \times 10^9 \left( \text{Summation of } C_i Z_i^2 \right)^{1/2} \text{ m}^{-1}$$

$C_i$ and $Z_i$ are the molar concentration and valency of ion "i" respectively.

$1/K$ - thickness of double layer

Therefore, once the first layers of the particles have settled on the concentrating surface, which would naturally be coarser in size and heavier in density and thus would not be influentially repelled by e.d.l forces of the concentrating surfaces, the subsequent settling of the finer particles on top would not experience the repulsive forces of the concentrating surface. This also partly shows the folly of attempting to treat feeds of very fine particles only. However, the already settled particles, if they possess a high negative potential will retard the subsequent settling of the fine particles. This will lead to a drop in the efficiency of concentration.
Profile patterns of small dimensions, say, as deep as 100 μm, with gradually widening mouths can promote the effectiveness of a slime gravity process. Such depressions will enable the deck to treat a larger number of pulsating dilated layers of one particle thickness than a strictly smooth surface. Pulsating dilation of the bed of particles in and above a depression is the result of the combined effects of the motion of the deck, Bagnold dispersive forces, and the eddies created within the depression, particularly at the down slope of the depression. Laminar flow with low velocities can be expected to persist over a large part of such shallow and wide profiles. Under such circumstances, eddies produced are weaker and lack the momentum to carry away the very fine heavy particles particularly when they are caught amongst the coarser particles. The motion of the deck and the Bagnold dispersive forces are the driving forces for the trickling of the fine particles through the intersticies and the tendency for the system to attain the minimum potential energy.

On the basis of the above line of argument, it is postulated that different concentrating surfaces with nominally smooth planes should have profiles of varying dimensions and patterns, and that this should account for the different effects of the various deck surfaces noticed in the past.

**APPROACH**

The profile patterns of different existing concentrating surfaces were measured using an equipment which employs the "turn-table" technique. The specific type of equipment used is known as Perth-O-Graph (Fig. 3). Appropriate details on this equipment can be found in the work of Berglund (15). Assessment of surface texture is a well developed field. A detailed account on many modern methods of assessing the surface texture is given by Whitehouse (16).
Roughness profiles of fibreglass, stainless steel and wood concentrating surfaces were obtained. The wood was marine plywood (10). Measurements were made on the plywood with and without coatings of varnish. Representative measurements are found in figures 4-8.

Experiments were carried out on a Mozley laboratory separator (17,18) to determine the effect of the above used concentrating surfaces which possess different profiles of roughness. The separator was originally fitted with a flat stainless steel surface. The fibreglass surface studied was especially made and supplied by the manufacturers of the above laboratory separator. The wood concentrating surfaces were obtained by lining the stainless steel surface with marine plywood coated with and without varnish.

The synthetic test materials used in the experiments were a high grade quartz (95.6% SiO₂) and a high grade hematite (68.9% Fe) slimes, available in the Division of Mineral Processing of the above university. The zeta potentials of these two slimes are presented in figures 9 and 10. The experimental methods used in the zeta potentials studies are described elsewhere (18). Gravity experiments were carried out with single and mixed mineral slimes made of the above two materials.

The natural test material used was a sample of Yxsjöberg (Sweden) old scheelite tailing. In addition to scheelite, the other relatively heavy minerals present in the tailing were garnet-pyroxene skarn and pyrohotite (19).

The particle size distributions of the above slimes are given in
Table 2. The operating conditions for the experiments are given in Table 3.

Table 2: Particle size distributions of the slimes

<p>| Size range | % Cumulative weight (Fine) |</p>
<table>
<thead>
<tr>
<th></th>
<th>Quartz (by sedigraph)</th>
<th>Hematite (by microsieves)</th>
<th>Yxsjöberg old tail (by microsieves)</th>
</tr>
</thead>
<tbody>
<tr>
<td>-38</td>
<td>96</td>
<td>99.99</td>
<td>100</td>
</tr>
<tr>
<td>-30</td>
<td>92</td>
<td>76.26</td>
<td></td>
</tr>
<tr>
<td>-20</td>
<td>75</td>
<td>55.06</td>
<td>53.99</td>
</tr>
<tr>
<td>-10</td>
<td>22</td>
<td>44.53</td>
<td></td>
</tr>
<tr>
<td>-5</td>
<td>10</td>
<td>39.60</td>
<td>26.83</td>
</tr>
</tbody>
</table>
Table 3: Operating conditions

<table>
<thead>
<tr>
<th></th>
<th>Set 1</th>
<th>Set 2</th>
<th>Set 3</th>
<th>Set 4</th>
</tr>
</thead>
<tbody>
<tr>
<td>material</td>
<td>quartz</td>
<td>hematite</td>
<td>mixed</td>
<td>schee. tail</td>
</tr>
<tr>
<td>weight of feed(g)</td>
<td>90</td>
<td>20</td>
<td>90 quartz</td>
<td>100</td>
</tr>
<tr>
<td>pulp density (%solids w/w)</td>
<td>50</td>
<td>20</td>
<td>50</td>
<td>50</td>
</tr>
<tr>
<td>pH</td>
<td>8.6±0.1(nat.)</td>
<td>8.8±0.1(nat.)</td>
<td>8.6±0.1(nat.)</td>
<td>10.5-11.0*</td>
</tr>
<tr>
<td>flowrate (l/m)</td>
<td>2.5, 3.0, 4.0</td>
<td>2.5, 3.0, 4.0</td>
<td>2.5 for s.steel</td>
<td>2.5 for f.glass</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>3.0 for wood 1</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>4.0 for wood 2</td>
</tr>
</tbody>
</table>

* also in the presence of silicate surfaces: stainless steel, fibreglass, wood with one coating of varnish (wood 1), wood with no coating of varnish (wood 2)

slope: 1.75°
strokes: 3" (76.2 mm)
speed: 90 rpm
con. time: 30 mts.
oper. time: 3 mts.
temperature: 20± 2°C

After each experiment the material on the concentrating surface was divided into three parts and collected separately. The high grade concentrate (i.e. fraction nearest the feed end) is designated 'conc 1' (0-43 cm from the feed end), the next fraction 'conc 2' (43-68.5 cm from the feed end) and the last fraction 'conc 3' (68.5 cm-end of the surface). The material that left the surface during the whole
operation is designated 'tail'.

RESULTS AND DISCUSSION

The surface roughness profiles obtained for various concentrating surfaces suggest the considerable influence it can have on the treatment of slimes by gravity methods. Falls and dips as deep as 4-5 μm, with mouths as wide as 25-40 μm are quite common on fibreglass. However, on the stainless steel tray it is only about .25-.50 μm deep (Figures 4 & 5). Wood has the deepest and widest patterns of profiles (Figures 6, 7 and 8). Wood always gives the best performance with slime feeds. For example, in the experiments by Burt (10), wood gave the best recovery, followed by fibreglass and stainless steel (Fig. 2). Stainless steel gave the least recovery, because this tray failed to shield the particles, even partly. Wood shielded the particles partly and also fully. It also promoted concentration and thereby increased the efficiency (10).

More evidence is seen in the results of the current experiments. Tables 4 and 5 which contain the results of the experiments with single slimes, clearly demonstrate the advantages of employing concentrating surfaces with distinct roughness profiles.

For example, during the experiments with the quartz material which assays only 95.6% SiO₂, it was seen through the microscope that the material left on the 'cone 1' region of the surface was predominantly brown to black in colour, presumably iron minerals and/or parts of the grinding media. As high as 400 times more recovery (Table 4) is possible in the 'cone 1' region when a prominently rougher surface is used instead of the stainless steel surface.
Table 4: Roughness profiles studies

Material: A high grade quartz (95.6% SiO₂)
Weight of the feed: 90 grams

<table>
<thead>
<tr>
<th>Cone</th>
<th>s.steel</th>
<th>f.glass</th>
<th>wood 1</th>
<th>wood 2</th>
</tr>
</thead>
<tbody>
<tr>
<td>2.5 (l/m)</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Conc 1</td>
<td>0.001</td>
<td>0.06</td>
<td>0.25</td>
<td>0.45</td>
</tr>
<tr>
<td>Conc 2</td>
<td>0.17</td>
<td>0.20</td>
<td>2.09</td>
<td>1.78</td>
</tr>
<tr>
<td>Conc 3</td>
<td>2.36</td>
<td>2.78</td>
<td>7.28</td>
<td>7.97</td>
</tr>
<tr>
<td>3.0 (l/m)</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Conc 1</td>
<td>0.001</td>
<td>0.05</td>
<td>0.12</td>
<td>0.38</td>
</tr>
<tr>
<td>Conc 2</td>
<td>0.07</td>
<td>0.31</td>
<td>1.06</td>
<td>0.99</td>
</tr>
<tr>
<td>Conc 3</td>
<td>1.16</td>
<td>3.08</td>
<td>4.06</td>
<td>7.29</td>
</tr>
<tr>
<td>4.0 (l/m)</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Conc 1</td>
<td>0.001</td>
<td>0.04</td>
<td>0.05</td>
<td>0.17</td>
</tr>
<tr>
<td>Conc 2</td>
<td>0.07</td>
<td>0.19</td>
<td>0.95</td>
<td>0.90</td>
</tr>
<tr>
<td>Conc 3</td>
<td>0.71</td>
<td>1.38</td>
<td>2.37</td>
<td>3.58</td>
</tr>
</tbody>
</table>
Table 5: Roughness profiles studies

Material: A high grade hematite (68.9% Fe)
Weight of the feed: 20 grams

<table>
<thead>
<tr>
<th></th>
<th>s.steel</th>
<th>f.glass</th>
<th>wood 1</th>
<th>wood 2</th>
</tr>
</thead>
<tbody>
<tr>
<td>2.5 (1/m)</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Conc 1</td>
<td>15.00</td>
<td>40.05</td>
<td>40.55</td>
<td>39.70</td>
</tr>
<tr>
<td></td>
<td>(3.00)</td>
<td>(8.01)</td>
<td>(8.11)</td>
<td>(7.94)</td>
</tr>
<tr>
<td>Conc 2</td>
<td>45.50</td>
<td>28.05</td>
<td>32.05</td>
<td>36.90</td>
</tr>
<tr>
<td></td>
<td>(9.10)</td>
<td>(5.61)</td>
<td>(6.41)</td>
<td>(7.38)</td>
</tr>
<tr>
<td>Conc 3</td>
<td>13.95</td>
<td>8.90</td>
<td>4.10</td>
<td>1.60</td>
</tr>
<tr>
<td></td>
<td>(2.79)</td>
<td>(1.78)</td>
<td>(0.82)</td>
<td>(0.32)</td>
</tr>
<tr>
<td>3.0 (1/m)</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Conc 1</td>
<td>0.60</td>
<td>27.55</td>
<td>28.80</td>
<td>29.55</td>
</tr>
<tr>
<td></td>
<td>(0.12)</td>
<td>(5.51)</td>
<td>(5.76)</td>
<td>(5.91)</td>
</tr>
<tr>
<td>Conc 2</td>
<td>46.05</td>
<td>33.35</td>
<td>40.95</td>
<td>43.30</td>
</tr>
<tr>
<td></td>
<td>(9.21)</td>
<td>(6.67)</td>
<td>(8.19)</td>
<td>(8.66)</td>
</tr>
<tr>
<td>Conc 3</td>
<td>27.50</td>
<td>13.90</td>
<td>4.85</td>
<td>6.40</td>
</tr>
<tr>
<td></td>
<td>(5.50)</td>
<td>(2.78)</td>
<td>(0.97)</td>
<td>(1.28)</td>
</tr>
<tr>
<td>4.0 (1/m)</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Conc 1</td>
<td>0.20</td>
<td>7.15</td>
<td>8.10</td>
<td>23.10</td>
</tr>
<tr>
<td></td>
<td>(0.04)</td>
<td>(1.43)</td>
<td>(1.62)</td>
<td>(4.62)</td>
</tr>
<tr>
<td>Conc 2</td>
<td>14.50</td>
<td>45.15</td>
<td>50.75</td>
<td>47.65</td>
</tr>
<tr>
<td></td>
<td>(2.90)</td>
<td>(9.03)</td>
<td>(10.15)</td>
<td>(9.53)</td>
</tr>
<tr>
<td>Conc 3</td>
<td>53.40</td>
<td>22.80</td>
<td>12.90</td>
<td>3.00</td>
</tr>
<tr>
<td></td>
<td>(10.68)</td>
<td>(4.56)</td>
<td>(2.58)</td>
<td>(0.60)</td>
</tr>
</tbody>
</table>
The experimental results on the synthetic slime and Yxsjöberg old tailing which exemplify the significances of employing roughness profiles are tabulated in tables 6 and 7 respectively. The material reported to 'conc 3' region is not considered significant because:

a. Material that reached the 'conc 3' region under laboratory conditions can possibly be lost during plant conditions (Figure 2; for more details refer Burt's (10) work);
b. In gravity equipment like slime shaking tables the concentrates are discharged in a direction perpendicular to that of the flow of the pulp. It becomes clear, therefore, that the heavy mineral to be recovered is kept as high as possible up the slope and is protected from the sweep of the flowing pulp.

Table 6: Role of concentrating surfaces, possessing different profiles of roughness, on the effectiveness of a slime gravity process (synthetic slime).

<table>
<thead>
<tr>
<th></th>
<th>s.steel</th>
<th>f.glass</th>
<th>wood 1</th>
<th>wood 2</th>
</tr>
</thead>
<tbody>
<tr>
<td>Conc 1 region</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Recovery (%)</td>
<td>0.83</td>
<td>5.64</td>
<td>17.16</td>
<td>20.43</td>
</tr>
<tr>
<td>Grade (%) Fe</td>
<td>69.0</td>
<td>69.3</td>
<td>67.4</td>
<td>68.4</td>
</tr>
<tr>
<td>Efficiency (%)</td>
<td>0.73</td>
<td>5.00</td>
<td>14.90</td>
<td>18.13</td>
</tr>
<tr>
<td>Conc 1 and Conc 2 regions</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Recovery (%)</td>
<td>42.14</td>
<td>47.13</td>
<td>64.75</td>
<td>66.31</td>
</tr>
<tr>
<td>Grade (%) Fe</td>
<td>69.0</td>
<td>69.3</td>
<td>65.1</td>
<td>65.9</td>
</tr>
<tr>
<td>Efficiency (%)</td>
<td>38.73</td>
<td>43.97</td>
<td>57.04</td>
<td>59.35</td>
</tr>
</tbody>
</table>
Table 7: Role of concentrating surfaces, possessing different profiles of roughness, on the effectiveness of a slime gravity process (Yxsjöberg old tailing).

<table>
<thead>
<tr>
<th></th>
<th>s.steel</th>
<th>f.glass</th>
<th>wood 1</th>
<th>wood 2</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Conc 1 region</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Recovery (%)</td>
<td>0.00</td>
<td>5.71</td>
<td>32.91</td>
<td>29.68</td>
</tr>
<tr>
<td>Grade (%) Fe</td>
<td>-</td>
<td>40.0</td>
<td>29.60</td>
<td>15.20</td>
</tr>
<tr>
<td>Efficiency (%)</td>
<td>-</td>
<td>2.75</td>
<td>11.65</td>
<td>5.14</td>
</tr>
<tr>
<td><strong>Conc 1 and Con2 regions</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Recovery (%)</td>
<td>43.57</td>
<td>43.90</td>
<td>65.03</td>
<td>58.28</td>
</tr>
<tr>
<td>Grade (%) Fe</td>
<td>20.60</td>
<td>25.61</td>
<td>8.18</td>
<td>10.28</td>
</tr>
<tr>
<td>Efficiency (%)</td>
<td>10.55</td>
<td>13.39</td>
<td>5.76</td>
<td>6.66</td>
</tr>
</tbody>
</table>

Depending on the nature of the feed, particularly on the associated other heavy mineral contents, a drop in grade can result from roughness (example: refer table 7). This drop is, however, tolerable in the roughing stages if very high recoveries down to about 5 \( \mu m \) can be achieved. It can be seen from tables 5 and 6 that 100 percent recovery down to 5 \( \mu m \) is possible in the 'conc 1' and 'conc 2' regions alone. Such a rougher concentrate can be efficiently upgraded on a cleaner equipment like Bartles XB concentrator (20).
CONCLUSIONS

i. Of the essentially smooth concentrating surfaces measured, wood has the deepest and widest patterns of profiles, followed by fibreglass and stainless steel.

ii. The question arises whether e.d.l. forces of a concentrating surface do really have any significant influence on typical slime feeds.

iii. The effectiveness of concentration increases with the increasing roughness profile of an essentially smooth concentrating surface.

iv. The roughness profiles obtained explain why wood was always preferred by operators for extreme operations.

v. It is suggested that concentrating surfaces with cut profile patterns should be used in appropriate slime gravity processes instead of the existing fibreglass surfaces.

ACKNOWLEDGEMENT

The project was financed by grants from the Swedish Board for Technical Development and the Swedish Mineral Processing Research Foundation.
REFERENCES


18. R.Sivamohan and E.Forssberg, Influence of electrokinetic environment in slime gravity concentration, Accepted for publication in Scandinavian Journal of Metallurgy.

Fig. 1 Streaming potentials of various concentrating surfaces.
(After Michell and Osborne, 1975; Ref. 7)
Fig. 2 Effect of pH and concentrating surface on recovery of cassiterite. (After Burt, 1978; Ref. 10)
Fig. 3  The equipment used for roughness profile measurement.
Fig. 4 Roughness profile of stainless steel.
Fig. 5  Roughness profile of fibreglass.
Fig. 6  Roughness profile of wood with no coating of varnish.
Fig. 7 Roughness profile of wood with one coating of varnish.
Fig. 8  Roughness profile of wood with two coatings of varnish.
Zeta potentials of quartz (95.6% SiO₂) in distilled water (HCl and NaOH were used as pH regulators).
Fig. 10  Zeta potentials of hematite (68.9% Fe) in distilled water (HCl and NaOH were used as pH regulators).
Influence of Electrokinetic Environment in Slime Gravity Concentration

Accepted for publication in the Scandinavian Journal of Metallurgy.
Abstract

Manipulation of particle-surface characteristics can be a way to improve the effectiveness of slime gravity processes. The possibilities for such an improvement when treating natural slimes are probed. The approach adopted was to attempt selective aggregation of the heavy mineral, by the addition of a multivalent inorganic salt, in conjunction with controlled dispersion of the constituent minerals.

In the first part, the relevant principles of selective aggregation by the addition of inorganic salts are presented. The suitability of slime gravity methods to separate the heavy mineral aggregates from the constituent minerals is discussed.

In the second part, the initial experimental studies carried out on natural slimes in accordance with the principles set out in the first part are presented and discussed. Cassiterite, chromite and scheelite slimes representing different varieties of minerals and having widely different properties such as surface characteristics, solubility, friability and thixotrophy, were chosen as examples. Despite the complexities present in these systems, it has been possible to show that the electrokinetic environment is an important factor in deciding the effectiveness of a slime gravity process.
INTRODUCTION

Recovery of valuable minerals from slimes is an increasingly pressing problem for the mineral industry. As the particles get finer in size the influence of surface related causes, to name a few, surface area, surface energy, colloidal coatings, heteroaggregation and pulp viscosity, become greater. Treatment of slimes by gravity methods is one of the areas where the effects of the surface related causes are felt (1,2,3,4,5). A review of these works suggested the necessity of carrying out more detailed work on surface phenomena (6). This present work is a step in that direction.

Part 1 - BASIC PRINCIPLES OF SELECTIVE AGGREGATION BY THE ADDITION OF INORGANIC SALTS

Homocoagulation need not necessarily occur simply due to the reduction of zeta potential to near zero. From figure 1-A, it can be seen that the repulsive force due to the surface potential is long ranged (up to a distance of about, say, 1 μm). It drops more gently as the inverse square of the distance. The attractive London-van der Waals force is short-ranged. Any attempt to bring the zeta potential (the potential at the slip plane of the double layer) to zero at low and intermediate concentration of indifferent electrolytes will lead to an energy barrier (Figs. 1-A and 1-B). This could be detrimental to the selective coagulation of the particles of a particular mineral from a polydispersed and polymineralised pulp of slimes, in the light of certain theories relating the energy barrier to the adverse effects noticeable in selective coagulation (8,9,10).

The height of the energy barrier, for the collision of two particles of radii $a_1$ and $a_2$, is proportional to the size factor, $f$, where $f=\frac{a_1 a_2}{a_1 + a_2}$, and therefore the barrier is high for any collisions involving large particles. The rate of coagulation falls off very steeply with the increase in the height of the energy barrier. The consequence of the existence of such an energy barrier will be the reduction in the probability of adhesion. However it must be noted that, for large particles, the probability of adhesion is also dependent on the kinetic energy of those particles (8).
Rapid coagulation (adhesion at every collision) is ensured when the energy barrier is removed. A high indifferent electrolytic concentration achieves this (Fig. 1-C). But the use of multivalent ions which show a certain chemical affinity for the mineral to be coagulated possesses the following advantages: 1) because of its specificity it adsorbs directly into the Stern layer (10-20 Å from the surface) and thereby forces the potential to drop to near zero at a distance within the range of London-van der Waals forces 2) because of its multivalency and specificity the concentration needed to achieve coagulation is low. It appears that there is disagreement over the way in which the multivalent ions show their specificity towards a mineral. Jan Leja (11), in his book "Surface Chemistry of Froth Flotation", writes, "The coagulation of any colloidal particle is affected by the hydroxylated complex specie and not the free uncomplexed multivalent ions. Matijevic (1967) emphasized that non-hydroxylated species are generally, unable to reverse the charge....". Fuerstenau and Raghavan (12) write, citing one of Fuerstenau's previous articles (13), "Certain inorganic ions exhibit surface activity: Ba$^{++}$ on quartz, Ba$^{++}$ and SO$_4^{--}$ on alumina, Ca$^{++}$ and SO$_4^{--}$ on rutile and hydrolyzed metal cations on a wide variety of solids".

According to Somasundaran (8), "Multivalent ions also indeed reduce interfacial potential and cause aggregation, but in this case it is necessary to exercise careful control of the addition since they can specifically adsorb to reverse the zeta potential and then even increase it to very high values at large additions. With certain multivalent ions, the effect has also been attributed to their increased adsorption when they are present in hydrolysed form...". James and Healy (14) have considered that the adsorption of hydrolysed species would not be through the hydroxy group by hydrogen bonding, but through the metal ion itself. It is, therefore, unclear whether a multivalent ion can show specificity on its own and if so what the limits are or whether it has to be hydrolysed to show specificity.

There are more instances where inorganic ions have shown a certain specific affinity. For example F$^{-}$ on alumina, F$^{-}$ on hematite (15) and
(CrO₄)²⁻ on chromite (16) have shown such behaviour. Sobieraj and Laskowski (17) found Al ions to have the greatest influence of the various trivalent species that could be present in the pulps of aluminium chromites. Such distinct effects of Al ions on chromite were obtained in an earlier work too (18). Whether this was partly due to the specificity of the naked Al ion or otherwise has to be carefully analysed. The points worthy of being considered are, amongst others, that in normal spinels trivalent ions are in octahedral sites, whereas the divalent ions are co-ordinated tetrahedrally with oxygen and that some chromites have a high content of Al and other chromites don't. For example, the chromites from Kemi (Finland) deposit belong to the group of iron-rich aluminium chromites (19).

Ions which show surface activity in addition to electrostatic attraction may be identified by zeta potential studies because of their ability to reverse the charge of a mineral surface on adding reagents of various concentrations under conditions of constant potential determining ions. Or, more easily, by the shift in the point of zero charge to another value (iep) in the presence of a specifically adsorbing ion.

In the case of polyvalent cations the production of "metal hydroxides" becomes a very important factor. Such species could show favourable electrostatic interactions. Inorganic polymolecular complexes can exert their influence on bridging mechanisms in a manner similar to that of the organic polyelectrolytes such as polyacrylamide (20). Moreover, the nature and concentration of anions decide the behaviour of "metal hydroxides", depending on the basicity of anions, the co-ordinative binding affinity of the anion for a particular cation and the resistance of bound anions to displacement by added hydroxyl ion (21). Such interactions could take place at mineral surfaces, between the surface anions and the inorganic polymolecular complexes, which will lead to preferential aggregation. For example, surface chromate ions of chromite may be expected to preferentially interact with the polymolecular complexes of Al ions or else chromate loaded Al ion complexes may be expected to preferentially adsorb on chromite. In the presence of chromate ion the pH for maximum floc formation of alum (Al₂(SO₄)₃·18H₂O) is reported to shift over a wide
range (20). In the experiments carried out by Sobieraj and Laskowski (17) with an Albanian chromite, the presence of polymolecular Fe(OH)$_3$ complexes did not have any adverse effects on the flotation behaviour until a very high concentration was reached (Fig. 2-A). But the Al(OH)$_3$ complexes did have a marked effect (Fig. 2-A and Fig. 2-B).

SUITABILITY OF SLIME GRAVITY PROCESSES TO TREAT THE HEAVY MINERAL AGGREGATES, OBTAINED BY THE ADDITION OF INORGANIC SALTS

i. Slime gravity equipment can effectively concentrate the heavy minerals (minerals of Cr, Sn, Ta, W etc.) of very fine sizes (10 μm and down). Likewise, the slime gravity equipment may be able to treat the aggregates of the above heavy minerals as well. However, experience has shown that the flocs obtained by the addition of a high molecular weight anionic polymer have the tendency to be washed down to the middling side (Fig. 3). Therefore the polymer length and the nature of the flocs, whether it is loose and open or compact become important factors. Usually flocs obtained by the addition of polymers are loose and open (8).

ii. Problems of weak aggregates should not arise here because the slime gravity equipment have a mode of operation which eliminates turbulent conditions capable of deaggregating any aggregates, particularly coagulums.

iii. If the coagulums are small in size they will possess rather slow settling rates, a disadvantage that has been cited widely (10,22,23,24). This disadvantage is almost overcome when such coagulums are allowed to settle in thin films of 1-2 mm thick, found on the slime gravity equipment. For example, a coagulum of 12 μm in size since it is compact, unlike a floc which is loose and open, will take about 3.2 seconds only to fall through a depth of 1 mm (on the basis of the Stokes Law). However, this problem does not arise when ultrafine particles are aggregated with relatively coarser particles. Such a coagulation process may avoid any probable adverse effects occurring due to secondary minima phenomena.
iv. In addition to the precipitation of inorganic polymolecular complexes on the surface of the mineral, bulk precipitation is also possible. The entrapping of gangue minerals in the bulk precipitates attempts to mar the effectiveness of a concentrating process. Stirring during conditioning helps to release such entrapped particles. It is more important to get rid of the entrapped particles during settling - when aggregates are being separated from the dispersed particles. Slime gravity equipment should be able to meet this requirement due to their ability to work to release such entrapped particles, caused by the motion of the concentrating surface they incorporate.

v. Ability to gently feed the conditioned pulp onto any slime gravity equipment by gravity and thereby preventing the pumping of the pulp.

A HYPOTHETICAL PROCEDURE TO ACHIEVE SELECTIVE AGGREGATION BY THE ADDITION OF INORGANIC SALTS, IN NATURAL SYSTEMS.

i. Survey of the relevant literature in order to select a suitable multivalent ion capable of showing affinity for the heavy mineral to be aggregated and also to find details on the dispersants and modifiers to be added to control the electrokinetic environment.

ii. Zeta potential studies in distilled water and pulp liquids.

iii. Stable but not thorough dispersion of the constituent gangue minerals. Over-stabilization can impair the subsequent step of selective aggregation of the valuable mineral. A potential less than 10-15 mV is ideal for the mineral to be aggregated (8).

A dispersant such as sodium silicate which does not show any chemical affinity towards the heavy mineral to be aggregated, is suitable for this purpose.

Under suitable circumstances, it is preferrable to achieve the dispersion of gangue minerals by the addition of proper
dispersants which can specifically adsorb. For example, fluorite may be selectively dispersed by the addition of \( \text{CO}_3^{--} \) ions (25) and the addition of \( \text{F}^- \) ions along with the \( \text{CO}_3^{--} \) ions will control the solubility of fluorite.

iv. Addition of a modifying agent which incorporates a negative ion that is specific to the heavy mineral that is to be selectively aggregated. This will expel any dispersants that may have been adsorbed onto it. For example \( \text{Na}_2\text{CrO}_4 \) in the case of chromite (16) and \( \text{Na}_2\text{S} \) in the case of galena (26) may be used.

This step is critical and care must be taken not to exceed the amount that is just sufficient to expel the dispersant.

v. Addition of a multivalent aggregant that is specific to the heavy mineral that is to be aggregated. This step too is critical since it can reverse the zeta potential and then even increase it to very high values.

vi. Conditioning for a fixed period of time at a pH where the mineral to be aggregated is expected to have its iep in the pulp liquid.

vii. Separation in a suitable slime gravity equipment.

Part 2 - EXPERIMENTAL STUDIES OF THE NATURAL SLIMES

Test materials. "Not deslimed" (i.e. in the presence of ultrafine particles) natural samples with a top size of 38 to 75 \( \mu \)m were chosen from the slime circuits and tailings of various plants in Europe - Yxsjöberg (Sweden) old scheelite tailings, Kemi (Finland) chromite slime circuit and the Geevor slime plant, Cornwall (U.K.). The samples selected are representative of different varieties of minerals and have widely different properties such as surface characteristics, solubility, friability, thixotrophy etc.

The particle size distribution of the materials studied is given in table I. The approximate mineral distributions are given in tables II, III and IV.
Table I. Particle size distribution (by micro-mesh sieves).

<table>
<thead>
<tr>
<th>Sieve range</th>
<th>Yxsjöberg (-53μm frac.)</th>
<th>Yxsjöberg (-38μm frac.)</th>
<th>Kemi (material 1)</th>
<th>Kemi (material 2) (-38μm frac.)</th>
<th>Geevor</th>
</tr>
</thead>
<tbody>
<tr>
<td>-75 +53</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>7.88</td>
<td>-</td>
</tr>
<tr>
<td>-53 +38</td>
<td>27.44</td>
<td>-</td>
<td>-</td>
<td>8.12</td>
<td>-</td>
</tr>
<tr>
<td>-38 +20</td>
<td>33.76</td>
<td>46.01</td>
<td>47.38</td>
<td>23.92</td>
<td>11.10</td>
</tr>
<tr>
<td>-20 +5</td>
<td>19.50</td>
<td>27.16</td>
<td>29.89</td>
<td>39.82</td>
<td>28.86</td>
</tr>
<tr>
<td>-5</td>
<td>19.30</td>
<td>26.83</td>
<td>22.73</td>
<td>20.26</td>
<td>60.04</td>
</tr>
</tbody>
</table>
Table II. Yxsjöberg (Sweden) old scheelite tailing analysis (-38 μm - sieve diameter)

<table>
<thead>
<tr>
<th>Oxide</th>
<th>Weight %</th>
<th>Mineral</th>
<th>Weight %</th>
</tr>
</thead>
<tbody>
<tr>
<td>SiO$_2$</td>
<td>32.85</td>
<td>Skarn minerals</td>
<td>37</td>
</tr>
<tr>
<td>CaO</td>
<td>23.50</td>
<td>SiO$_2$</td>
<td>32.85</td>
</tr>
<tr>
<td>Fe$_{23}$</td>
<td>19.71</td>
<td>CaF$_2$</td>
<td>21</td>
</tr>
<tr>
<td>Al$_{23}$</td>
<td>6.20</td>
<td>CaCO$_3$</td>
<td>5</td>
</tr>
<tr>
<td>S</td>
<td>3.30</td>
<td>CaWO$_4$</td>
<td>1.005</td>
</tr>
<tr>
<td>MgO</td>
<td>2.19</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Na$_2$O</td>
<td>1.42</td>
<td></td>
<td></td>
</tr>
<tr>
<td>K$_2$O</td>
<td>1.14</td>
<td></td>
<td></td>
</tr>
<tr>
<td>WO$_3$</td>
<td>.81</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
### Table III. Kemi slime (material 1) Analysis
(-38 μm - sieve diameter)

<table>
<thead>
<tr>
<th>Oxide</th>
<th>Weight %</th>
<th>Mineral</th>
<th>Weight %</th>
</tr>
</thead>
<tbody>
<tr>
<td>SiO₂</td>
<td>37</td>
<td>Mg₃Si₂O₅(OH)² (talc)</td>
<td>32</td>
</tr>
<tr>
<td>MgO</td>
<td>30</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cr₂O₃</td>
<td>16.08</td>
<td>Mg₃Si₂O₅(OH)⁴ (serpentine)</td>
<td>29.5</td>
</tr>
<tr>
<td>Fe₂O₃</td>
<td>10</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Al₂O₃</td>
<td>6</td>
<td>(Fe,Mg)(Cr,Al)₀₂ (Mg,Fe)(Cr,Al)₀₈ (OH)</td>
<td>38.5</td>
</tr>
<tr>
<td></td>
<td></td>
<td>(chromite and Cr bearing chlorite)</td>
<td></td>
</tr>
</tbody>
</table>
Table IV. Geevor slime tailing Analysis
(-38 μm - sieve diameter)

<table>
<thead>
<tr>
<th>Oxide</th>
<th>Weight %</th>
<th>Mineral</th>
<th>Weight %</th>
</tr>
</thead>
<tbody>
<tr>
<td>SiO₂</td>
<td>57.5</td>
<td>Al₂SiO₄(OH)₂ (kaolinite)</td>
<td>29.7</td>
</tr>
<tr>
<td>Al₂O₃</td>
<td>19.3</td>
<td>SiO₂ (silica)</td>
<td>27.8</td>
</tr>
<tr>
<td>FeO</td>
<td>12.8</td>
<td>H₂KA₁(SiO₃)₂ (mica)</td>
<td>19.6</td>
</tr>
<tr>
<td>K₂O</td>
<td>4.6</td>
<td>Fe₂O₃</td>
<td>12.8</td>
</tr>
<tr>
<td>CaO</td>
<td>1.5</td>
<td>1/2(CaO) (MgO)</td>
<td>10.2</td>
</tr>
<tr>
<td>TiO₂</td>
<td>0.6</td>
<td>(SiO₂)(H₂O)...</td>
<td>0.7</td>
</tr>
<tr>
<td>Sn</td>
<td>0.55</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cu</td>
<td>0.164</td>
<td>SnO₂</td>
<td>0.704</td>
</tr>
<tr>
<td>W</td>
<td>0.016</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

As obtained the Yxsjöberg old scheelite tailing assays 0.41% WO₃ and 34% SiO₂. The -53 μm fraction (41 weight%) and the -38 μm fraction (25.5 weight%) assay 0.83% WO₃ and 0.81% WO₃ respectively.

The Kemi chromite slimes (material 1 and material 2 (table I)) are the feed to the Jones magnetic separator of the slime circuit. The chemical analysis of material 1 indicated that it was from the Elijarvi open pit of the Kemi deposit (19). There is a substantial difference between the mineral compositions of the ores from the current (1984) operating Elijarvi and Viiia open pits (19) and therefore the nature of the feed to the Kemi concentrator is inconsistent. Material 2 of the Kemi chromite slimes was obtained about six months after the first material.

The Geevor material was a tailing (99% -75 μm, 95% -38 μm) of the
Geevor slime plant, Cornwall. It consists of more than fifty percent of clayey matter which makes it a very difficult material to treat. And also, the small crystallites of cassiterite (less than 15 μm) in the Geevor ore are often interlocked or massed in the gangue grains (27).

Zeta potential studies. The zeta potentials were determined by a microelectrophoresis equipment - Laser Zee™ model 500, available at the Division of Process Technique, University of Oulu, Finland. This model incorporates a unique patented prism technique (28) which makes the necessary procedure several times faster than the conventional equipment.

Table V gives the particulars of the minerals used in the zeta potential studies. The -20 μm fractions of the minerals were prepared by prolonged grinding in a powered agate mortar in distilled water and stored in poly-ethylene plastic bottles in distilled water. The storage period was less than a week.

Table V Details on minerals used for zeta potential studies

<table>
<thead>
<tr>
<th>mineral</th>
<th>grade %</th>
<th>origin</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>WO 3</td>
<td>Cr 0 2 3</td>
</tr>
<tr>
<td>scheelite</td>
<td>77.7</td>
<td></td>
</tr>
<tr>
<td></td>
<td>SiO 2</td>
<td>Sn 2</td>
</tr>
<tr>
<td></td>
<td>Fe tot.</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Yxsjöberg(Sweden)</td>
</tr>
<tr>
<td></td>
<td></td>
<td>(gravity,magnetic</td>
</tr>
<tr>
<td></td>
<td></td>
<td>and electrostatic</td>
</tr>
<tr>
<td></td>
<td></td>
<td>concentrate)</td>
</tr>
<tr>
<td>chromite</td>
<td>47.7</td>
<td>1.6</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Kemi, Finland</td>
</tr>
<tr>
<td>cassiterite</td>
<td>79</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Geevor, Cornwall, U.K.</td>
</tr>
<tr>
<td>hematite</td>
<td>65</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Malmberget, Sweden</td>
</tr>
<tr>
<td>calcite</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Stora Vika, Sweden</td>
</tr>
</tbody>
</table>
Before every measurement a sufficient quantity of the mineral sample was gently ground in a pestle and mortar with a small amount of liquid in which the measurement is to be made, in order to ensure a clean surface. The sample preparation for measurements and the procedure adopted during measurements were as found in the operating manual. Readings were repeated four times, with a new sample in the chamber for every two readings, and averaged for one zeta potential measurement.

The zeta potentials of scheelite were determined in distilled water, Mg salt solutions and pulp liquids. The zeta potentials of calcite were measured in distilled water and $K_2CO_3$ salt solutions. One set of measurements was made in distilled water containing both $K_2CO_3$ and Mg salts. $H_2SO_4$ and NaOH were used as pH regulators.

The analysis of water used to make the pulps is given in Table VI.

Table VI  Analysis of water

<table>
<thead>
<tr>
<th>Concentration (mg/l)</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Cl</td>
<td>63</td>
</tr>
<tr>
<td>SO$_4$</td>
<td>11</td>
</tr>
<tr>
<td>NO$_3$</td>
<td>&lt;2</td>
</tr>
<tr>
<td>Ca</td>
<td>22 (total hardness expressed as Ca)</td>
</tr>
<tr>
<td>Fe</td>
<td>0.17</td>
</tr>
<tr>
<td>HCO$_3$</td>
<td>32</td>
</tr>
</tbody>
</table>

The results are shown in figures 4, 5, 6 and 7. The addition of $2.46 \times 10^{-3}$ m/l Mg salt gives an iep value of about 8.8 and also increases the zeta potential to high positive values within a narrow pH range (Fig. 4). In the pulp liquid the zeta potential of scheelite
remains in the negative low zeta potential side (-10 to -14 mV) over the pH range of 7 to 10.5 (fig. 6).

The addition of 3.6x10^{-3} m/l K_2CO_3 fails to increase the negative potential of calcite appreciably. Also, the addition of 2.46x10^{-3} m/l Mg salt brings the zeta potentials back to near zero (Fig. 7).

The zeta potentials of chromite were obtained in distilled water, Al salt solutions (Fig. 8) and pulp liquid. HCl and NaOH were used as pH regulators.

Al salt is suitable to be tested as an aggregant in the case of Kemi chromite slimes due to the following reasons:

i. Heteroaggregation is possible between serpentine and chromites under alkaline conditions. Serpentine could have a positive zeta potential over a wide pH (29). And also, serpentine content of the Kemi chromite slimes is rather high (about 29.5% in material 1). Addition of silicate causes stable dispersion of the serpentine as well as the other gangue mineral talc and thereby will prevent the heteroaggregation of the gangue minerals on the chromites. However, the non-selective adsorption of silicate on chromite is undesirable. Expulsion of silicate from chromite is possible by the addition of chromate ions (16).

ii. Al specie have shown a strong affinity to aluminium chromites (17).

iii. There is a possibility to study the effect of "aluminium hydroxide" polymolecular complexes on the surface of chromite.

The supernatant liquid of the pulp for the zeta potential studies, was obtained by conditioning 150 grams of Kemi slime (material 1) in 850 ml of water, in the presence of 10 ml of 1% sodium metasilicate, 40 mg of Na_2CrO_4.4H_2O and 1.5x10^{-4} m/l Al_2(SO_4)_3 for 48 hours at natural pH, followed by settling and filtration. The zeta potential of the chromite was about -25 mV at pH 7.5-8.5.
A mildly alkaline test pH was decided for the chromite slimes in order to facilitate the interaction between chromate and aluminium specie and also to prevent the excessive dissolution of serpentine.

Michell and Cosio (2,3), in a fairly extensive study did a detailed observation of the electrokinetic behaviour of Cornwall cassiterite. It was found to have a pzc of pH 3.7. After conditioning with 25 mg/l copper sulphate, the cassiterite was seen to have a positive potential over a very wide pH range and a high degree of mobility below a pH of 5.3. Barium chloride at 25 mg/l did not have any mentionable influence except shifting the pzc of pH 3.7 to an iep value within pH 5. In the presence of 25 mg/l ferric chloride, positive values were obtained over the pH range of 2.5 to 9.8. They found that their measurements were comparable with previous similar measurements.

In the experiments they carried out with a synthetic feed to quantify the effects of cations in slime gravity concentration, copper sulphate and ferric chloride were found to have the maximum influence (copper salt being the best) at a concentration around 25 mg/l.

By way of comparison, the zeta potentials of cassiterite obtained from Geevor were measured in distilled water (Fig. 9). The pzc value obtained agrees well with that obtained (pH 3.7) by Michell and Cosio (2,3).

Zeta potentials of hematite were measured in Cu salt solution and also in Al salt solution. From figure 10 it can be seen that the presence of $8 \times 10^{-4}$ m/l CuSO$_4$ shifts the zeta potential to the left of the values obtained for hematite in distilled water. It appears that the SO$_4^{2-}$ ions have greater influence on hematite than the copper ions.

All the reagents used were of analytical grade.
Slime gravity experiments. Tests were carried out on a Mozley laboratory separator (30). The original stainless steel tray of the separator was replaced by a fibreglass tray. The flat tray arrangement is very suitable for treating slimes.

The tray is inclined at a shallow angle to the horizontal in the longitudinal direction and is reciprocated with a simple harmonic motion in a direction perpendicular to this longitudinal direction. Tray slope, water flowrate, strokes per minute and amplitude are equipment based operating variables. This equipment did not have a feeder and the suggested feeding arrangement was to pour a pulp of very high density (about 50% solids by weight) onto the tray and then to start the separator. This arrangement is not suitable for feeding a conditioned pulp of 15% solids. Hence, a cone feeder with a stirrer was fitted to the separator and the pulp was fed gradually after the separator was started.

Yxsjöberg old scheelite tailing (-53 μm fraction)

Tests were made with this material by varying only the pH. The intention is to test the seemingly contradicting conclusions on the dependence of slime gravity processes' efficiency on pH. Rao and Sirois (4), from the work they did on a slime hematite ore, concluded that the greater the difference between apparent iep and conditioning pH, the better would be the concentration efficiency. Burt (5), in a later study on a slime cassiterite ore concluded that at high pH, where good dispersion could be expected, the efficiency would be low.

The operating conditions of the tests are included in Table VII. The pH was adjusted by H₂SO₄ and NaOH.
Table VII The operating conditions for the laboratory experiments

<table>
<thead>
<tr>
<th></th>
<th>Yxsjöberg (-53 µm fraction)</th>
<th>Yxsjöberg(-38 µm fraction)/Kemi (material 1)/Kemi (material 2)/Geevor tailing</th>
</tr>
</thead>
<tbody>
<tr>
<td>pulp density</td>
<td>50% solids by wt. *</td>
<td>15% solids by wt. **</td>
</tr>
<tr>
<td>conditioning time</td>
<td>48 hours</td>
<td>48 hours</td>
</tr>
<tr>
<td>feeding time</td>
<td>-</td>
<td>1 minute</td>
</tr>
<tr>
<td>operating time</td>
<td>3 minutes</td>
<td>4 minutes</td>
</tr>
<tr>
<td>strokes/mt.</td>
<td>100 rpm</td>
<td>80 rpm</td>
</tr>
<tr>
<td>amplitude</td>
<td>3&quot; (76.2 mm)</td>
<td>3&quot; (76.2 mm)</td>
</tr>
<tr>
<td>slope</td>
<td>0</td>
<td>0.75</td>
</tr>
<tr>
<td>irrigation water</td>
<td>2.5 l/mt.</td>
<td>2 l/mt.</td>
</tr>
<tr>
<td>tray</td>
<td>stainless steel</td>
<td>fibreglass</td>
</tr>
</tbody>
</table>

* Rao and Sirois, 1974 (Ref. 4)
** 24 hours for the Kemi (material 2)
*** Including feeding time
**** 1.8 l/mt. for the Geevor tailing

Yxsjöberg old scheelite tailing (-38 µm fraction)

This material was used for the aggregation tests and 150 grams samples were used for each test. First 650 ml of water was added to 10 ml of 1% sodium metasilicate solution while stirring. The 1% silicate solution was prepared by diluting a 10% silicate solution. This silicate containing water was added to 150 grams of the material. One test was done by conditioning this pulp at 15% solids by weight and at a pH about 10.8.
For further tests three levels of aggregant (Mg) were decided. The levels were 1.07 g., 1.50 g. and 2.00 g. of MgCl₂·6H₂O per test. The pulps were treated as follows: (i) pulp containing silicate was prepared as described above and conditioned for 15 minutes. (ii) modifying agent NaF was added - 3.57 ml of 10⁻¹ m/l - and conditioned for 15 minutes. (iii) modifying agent K₂CO₃ was added - 2 grams dissolved in 100 ml of water and conditioned for 15 minutes. The Mg salt was added as 100 ml of dilute solution to the above pulp. The tested pH levels were 7.2-7.7, 9.1-9.35 (natural pH) and 10.7.

Conditioning was carried out by stirring the pulp at a fixed low rate of shear, in a 1000 ml cylinder. The speed of the stirrer had to be raised during the time of adding the aggregant and the pulp was stirred at this high rate of shear for a further 15 minutes.

Kemi slime (material 1)/Kemi slime(material 2)

Pulps containing silicate were prepared in the same way as for the Yxsjöberg material (-38 μm fraction). The first test was done by conditioning this pulp at 15% solids by weight and at pH of 7.9±0.4.

For further tests, the pulps were treated as follows: (i) pulp containing silicate was conditioned for 45 minutes (ii) modifying agent (Na₂CrO₄·4H₂O), 40 mg/test and 20 mg/test were added to material 1 and material 2 respectively as dilute solutions and conditioned for 45 minutes (iii) in the case of material 1, the tested levels of aggregant -Al₂(SO₄)₃·18H₂O- were 100 mg and 150 mg per test; for material 2 the tested levels were 10 mg, 25 mg, 50 mg and 100 mg. Al was added to the pulps of material 2 as chromate loaded Al salt solutions. The chromate loaded aluminium salt solutions were prepared by adding 20 mg of the modifying agent (Na₂CrO₄·4H₂O) to the Al salt solution, while stirring. Conditioning was carried out in the same way as for Yxsjöberg material (-38 μm fraction). The test pH was 7.9±0.4 and was adjusted by HCl.
Geevor tailing

The Geevor tailing (99% -75 μm; 95% -38 μm) equivalent to 150 grams was weighed. 375 ml of water, containing 17.25 mg of sodium metasilicate, estimated on the basis of hematite present, was added to the above tailing while stirring. Silicate was added as 1% solution. This pulp was sieved carefully on a 75 μm sieve, in order to remove all oversized particles and organic impurities. A 75 μm sieve had to be used to facilitate the easy passage of particles. Another 300 ml of water was used to wash the sieve carefully and thoroughly and no material was allowed to be lost during sieving. This pulp was diluted to 15% solids and conditioned for the first test.

Subsequent tests were done in the presence of Cu salt. The pulps were treated as follows: (i) pulp was prepared in the above said way and conditioned for 15 minutes (ii) salt was added as a dilute solution - 20 mg/test and 75 mg/test of CuSO₄.5H₂O - and conditioned in the same way as for Yxsjöberg material (-38 μm fraction) (iii) pulp was diluted to 15% solids and conditioned at a decided pH - pH 7 and pH 6. The pH was adjusted by HCl.

Kemi slime (material 2)

Pilot tests were carried out on a pilot scale Duplex concentrator (31), available at Bureau de Recherches Geologiques et Minieras (BRGM), France. The operating conditions maintained are given in Table VIII.
Table VIII  The operating conditions for the pilot tests on Kemi slime (material 2)

<table>
<thead>
<tr>
<th></th>
<th>Set 1</th>
<th>Set 2</th>
</tr>
</thead>
<tbody>
<tr>
<td>pulp density</td>
<td>30% solids by weight</td>
<td>15% solids by weight</td>
</tr>
<tr>
<td>conditioning time</td>
<td>24 hours</td>
<td>24 hours</td>
</tr>
<tr>
<td>feeding time</td>
<td>1.5 minutes</td>
<td>3.5 minutes</td>
</tr>
<tr>
<td>irrigating time</td>
<td>1 minute</td>
<td>1 minute</td>
</tr>
<tr>
<td>flushing time</td>
<td>20 seconds</td>
<td>20 seconds</td>
</tr>
<tr>
<td>feed rate</td>
<td>106.51 kg/hr.</td>
<td>59.00 kg/hr.</td>
</tr>
<tr>
<td>strokes/mt.</td>
<td>80 rpm</td>
<td>80 rpm</td>
</tr>
<tr>
<td>amplitude</td>
<td>3&quot; (76.2 mm)</td>
<td>3&quot; (76.2 mm)</td>
</tr>
<tr>
<td>slope</td>
<td>1°</td>
<td>1°</td>
</tr>
<tr>
<td>irrigation water</td>
<td>15 l/mt.</td>
<td>15 l/mt.</td>
</tr>
</tbody>
</table>

Tests were carried out in the presence of sodium metasilicate only and with different levels of aggregant (aluminium salt) additions. Due to unavoidable circumstances the pulps were pumped contrary to the requirement of being fed by gravity as laid down in the Part 1. And this probably made the tests in the presence of aggregant a failure. Further tests have not been carried out yet.

RESULTS AND DISCUSSION

Yxsjöberg old scheelite tailing (-53 μm fraction)

The results of the tests with this material where only the pH was the variable are given in Table IX. The efficiency is high at high pH and low at low pH (Fig. 11) which is seemingly not in agreement with the conclusions of Burt (5). Burt based his conclusions on the following: (i) apparent viscosity of the pulp (ii) electrokinetic potentials of the concentrating surface and the valuable and gangue mineral particles.
<table>
<thead>
<tr>
<th>pH</th>
<th>con. wt./feed wt. %</th>
<th>grade % (WO$_3$)</th>
<th>recovery %</th>
<th>efficiency</th>
</tr>
</thead>
<tbody>
<tr>
<td>4</td>
<td>2.8</td>
<td>13.2</td>
<td>44.53</td>
<td>6.66</td>
</tr>
<tr>
<td>6</td>
<td>2.08</td>
<td>17.19</td>
<td>43.01</td>
<td>8.58</td>
</tr>
<tr>
<td>7.6</td>
<td>3.12</td>
<td>12.34</td>
<td>46.22</td>
<td>6.43</td>
</tr>
<tr>
<td>7.6</td>
<td>1.73</td>
<td>19.67</td>
<td>41.06</td>
<td>9.46</td>
</tr>
<tr>
<td>9</td>
<td>2.14</td>
<td>16.31</td>
<td>42.13</td>
<td>7.94</td>
</tr>
<tr>
<td>11</td>
<td>1.19</td>
<td>28.09</td>
<td>40.09</td>
<td>13.47</td>
</tr>
<tr>
<td>11</td>
<td>1.18</td>
<td>27.51</td>
<td>39.21</td>
<td>12.88</td>
</tr>
</tbody>
</table>

Grade of the feed 0.83% WO$_3$

No middling recirculation, no concentrate upgrading

The apparent viscosity cannot be significant in a typical slime plant pulp of only 10% solids by weight as the one Burt (5) used. Such a deslimed feed of 10% solids by weight corresponds to less than about 4% solids by volume. And also it can be shown, assuming an average particle size of 20 µm and a tetrahedron packing, that the centre to centre distance between the particles is about 55 µm. Our viscosity studies with a scheelite tailing which consisted of about 25% -5 µm particles showed that there is no appreciable viscosity effect (32). Burt's feed was a cassiterite slime sample from Cornwall which usually contains quartz. Heteroaggregation is possible between hematite and quartz also between hematite and cassiterite at low pH which can lead to a drop in efficiency.

Moreover it can be discussed that the prevailing high electrokinetic potentials of a concentrating surface can have only a marginal effect. However the coarser particles of a feed which can settle on a concentrating surface without being influentially repelled by the electrokinetic potential of the concentrating surface can retard the subsequent settling of the finer particles and thereby cause a drop in
recovery. How far it would affect the efficiency of concentration depends on the relative amounts of valuable mineral particles in different size ranges, the magnitude of the electrokinetic potentials of the different minerals and the nature of the job. For example, if there are large amounts of valuable mineral particles, say, in the -10 μm range it should affect the efficiency as well. With such a feed pH conditioning alone is inadequate.

There is ample opportunity for heteroaggregation in the Yxsjöberg scheelite tailing, between skarn minerals and the rest at low pH. The state of dispersion improves with increasing pH due to the disruption of the non-selective coagulums. From table IX it can be seen that the grade improves markedly with pH against a comparatively low drop in recovery. This results in a steep increase in efficiency, (Fig. 11), suggesting that a dispersed pulp should give an efficiency increase by preventing heteroaggregation. However no oversimplified and generalised statement can be made on the relationship between efficiency and pH without taking into account the distribution of valuable mineral particles in different size groups, the magnitude of the electrokinetic potentials of different mineral particles and the nature of the job.

Yxsjöberg old scheelite tailing (-38 μm fraction)

Results of the experiments with this material under different electrokinetic environment are given in Table X. Recovery increases with increasing pH at every level of Mg salt and also recovery increases with increasing dosages of Mg salt at every level of pH (Fig. 12). This is presumably due to the fact that more and more "magnesium hydroxy" complexes are being formed and are attracted towards the scheelite particles. However maximum efficiency is obtained at the lowest pH only and also efficiency drops to its lowest when the pH is at its highest (Table X). The reason for this should be that some of the "magnesium hydroxide" polymolecular complexes are being attracted to and as well as enmesh the gangue particles. It is noteworthy that the point of zero charge of magnesium hydroxide is reported to be in the range of pH 12 to 12.5 only (33).
Table X  Yxsjöberg old scheelite tailing

<table>
<thead>
<tr>
<th></th>
<th>con wt./feed wt. %</th>
<th>grade (%) (\text{WO}_3)</th>
<th>recovery (%)</th>
<th>efficiency</th>
</tr>
</thead>
<tbody>
<tr>
<td>A 7</td>
<td>13.14</td>
<td>4</td>
<td>64.89</td>
<td>2.38</td>
</tr>
<tr>
<td>A N</td>
<td>16.97</td>
<td>3.1</td>
<td>64.94</td>
<td>1.66</td>
</tr>
<tr>
<td>A 11</td>
<td>25.33</td>
<td>2.3</td>
<td>71.93</td>
<td>1.17</td>
</tr>
<tr>
<td>B 7</td>
<td>20.10</td>
<td>2.8</td>
<td>69.48</td>
<td>1.54</td>
</tr>
<tr>
<td>B N</td>
<td>19.83</td>
<td>2.9</td>
<td>70.98</td>
<td>1.67</td>
</tr>
<tr>
<td>C 7</td>
<td>14.90</td>
<td>3.8</td>
<td>69.90</td>
<td>2.42</td>
</tr>
<tr>
<td>C N</td>
<td>18.33</td>
<td>3.3</td>
<td>74.69</td>
<td>2.16</td>
</tr>
<tr>
<td>HPH</td>
<td>20.32</td>
<td>2.9</td>
<td>72.75</td>
<td>1.73</td>
</tr>
</tbody>
</table>

Grade of the feed 0.81% \(\text{WO}_3\)

A: 1.07 g/test of \(\text{MgCl}_2\cdot6\text{H}_2\text{O}\); B: 1.50 g/test of \(\text{MgCl}_2\cdot6\text{H}_2\text{O}\);
C: 2.00 g/test of \(\text{MgCl}_2\cdot6\text{H}_2\text{O}\); 7: pH 7.2-7.7; N: pH 9.1-9.35 (nat. pH);
11: pH 10.7; HPH: with \(\text{Na}_2\text{SiO}_3\) at a high pH

Kemi slime (material 1)/Kemi slime (material 2)

The results of the laboratory experiments under various electrokinetic environment are tabulated in Tables XI and XII.

In the presence of a limited amount of sodium metasilicate, 100 mg per test, and at a pH of about 7.9, the recovery is 82.4% at a grade of 51.9% \(\text{Cr}_2\text{O}_3\) for material 1 (Table XI). It is noteworthy that it was obtained without recirculating or concentrate upgrading (the maximum grade possible is only about 55% \(\text{Cr}_2\text{O}_3\)) which strongly suggests the suitability of slime gravity methods to treat the Kemi slimes. In a similar attempt with the Kemi slime (material 2) recovery was 85.6% at a grade of 47% \(\text{Cr}_2\text{O}_3\) (Table XII). The considerable drop in grade compared with obtained with material 1 under identical conditions is most probably due to the fact that the latter material is finer in size. The coarsest gangue particles of material 2 were difficult to be washed down under the operating conditions employed.
Table XI  Kemi slime (material 1)

<table>
<thead>
<tr>
<th>reagents added (mg/test)</th>
<th>details of the concentrate</th>
<th>wt. %</th>
<th>Cr O (%)</th>
<th>recovery(%)</th>
<th>effi.</th>
</tr>
</thead>
<tbody>
<tr>
<td>SiO  CrO  Al</td>
<td></td>
<td></td>
<td>Cr 0 (%)</td>
<td></td>
<td></td>
</tr>
<tr>
<td>2 4</td>
<td></td>
<td></td>
<td>2 3</td>
<td></td>
<td></td>
</tr>
<tr>
<td>100 - - con.</td>
<td></td>
<td>19.52</td>
<td>52.62</td>
<td>63.86</td>
<td>51.72</td>
</tr>
<tr>
<td>mid.</td>
<td></td>
<td>5.99</td>
<td>49.69</td>
<td>18.52</td>
<td></td>
</tr>
<tr>
<td>con.+mid.</td>
<td></td>
<td>25.51</td>
<td>51.93</td>
<td>82.38</td>
<td>70.32</td>
</tr>
<tr>
<td>100 40 100 con.</td>
<td></td>
<td>18.41</td>
<td>54.08</td>
<td>61.93</td>
<td>52.07</td>
</tr>
<tr>
<td>mid.</td>
<td></td>
<td>9.49</td>
<td>46.77</td>
<td>27.61</td>
<td></td>
</tr>
<tr>
<td>con.+mid.</td>
<td></td>
<td>27.91</td>
<td>51.59</td>
<td>89.54</td>
<td>78.01</td>
</tr>
<tr>
<td>100 40 150 con.</td>
<td></td>
<td>14.53</td>
<td>48.23</td>
<td>43.59</td>
<td>28.08</td>
</tr>
<tr>
<td>mid.</td>
<td></td>
<td>10.45</td>
<td>52.61</td>
<td>34.20</td>
<td></td>
</tr>
<tr>
<td>con.+mid.</td>
<td></td>
<td>24.99</td>
<td>50.06</td>
<td>77.79</td>
<td>61.46</td>
</tr>
</tbody>
</table>

* salt weights
Table XII  Kemi slime (material 2)

<table>
<thead>
<tr>
<th>reagents added (mg/test)</th>
<th>details of the concentrate</th>
</tr>
</thead>
<tbody>
<tr>
<td>SiO  CrO  Al</td>
<td>wt. %</td>
</tr>
<tr>
<td>2  4</td>
<td></td>
</tr>
<tr>
<td>100  -  -</td>
<td>con.  36.13</td>
</tr>
<tr>
<td></td>
<td>mid.  14.18</td>
</tr>
<tr>
<td></td>
<td>con.+mid. 50.31</td>
</tr>
<tr>
<td>100  40  10</td>
<td>con.  38.33</td>
</tr>
<tr>
<td></td>
<td>mid.  15.69</td>
</tr>
<tr>
<td></td>
<td>con.+mid. 54.03</td>
</tr>
<tr>
<td>100  40  25</td>
<td>con.  38.37</td>
</tr>
<tr>
<td></td>
<td>mid.  14.94</td>
</tr>
<tr>
<td></td>
<td>con.+mid. 53.31</td>
</tr>
<tr>
<td>100  40  50</td>
<td>con.  37.83</td>
</tr>
<tr>
<td></td>
<td>mid.  15.07</td>
</tr>
<tr>
<td></td>
<td>con.+mid. 52.91</td>
</tr>
<tr>
<td>100  40  100</td>
<td>con.  38.57</td>
</tr>
<tr>
<td></td>
<td>mid.  14.83</td>
</tr>
<tr>
<td></td>
<td>con.+mid. 53.41</td>
</tr>
</tbody>
</table>

* salt weights

** of the 40 mg Na₂CrO₄·4H₂O, 20 mg was used to prepare the chromate loaded Al salt solution.
Al addition has caused efficiency to increase at 100 mg/test and 10 mg/test levels with material 1 and material 2 respectively. In the case of material 1, the distribution of chromite decreases in the concentrate but increases in the middling when the aggregant concentration is raised to 100 mg and then 150 mg (Figs. 13a and 13b). Interesting points emerge when the corresponding grades are considered (Fig. 13c).

Some useful comments can be made on the way in which Al could have influenced the efficiency. Charge reduction is possible not only by the Al ions such as Al(OH)$_2$$^+$ and even Al$_8$(OH)$_2$$^{2+}$ (34) but also by the charged "aluminium hydroxide" polymolecular specie. These inorganic polymolecular complexes can exert their influence on bridging mechanisms in a manner similar to that of the organic polyelectrolytes such as polyacrylamide (20). More evidence for the presumed electrostatic nature of the polymolecular aluminium hydroxides, and its effects on chromite surface can be found by carefully going through the work of Sobieraj and Laskowski (17).

Geevor tailing

The results of the experiments with the Geevor tailing in the presence of sodium metasilicate and sodium metasilicate-Cu salt combinations are presented as a grade-recovery curve (Fig. 14). It is evident from the figure that the performance improves in the presence of a small amount of Cu salt only. Both grade and recovery increase (a shift in the right direction on the grade-recovery curve - Fig. 14).

More experiments with the Geevor slime, devoid of -1 to -2 µm particles (quartz basis), since it will ensure the absence of a good amount of clays could be made. And, the selective dispersion of hematite, the other major relatively high specific gravity mineral, must be investigated. Read (15) observed that a halide (NaF) could
competitively displace the calgon and he postulated a specific 
adsorption between halides or cation-halide complexes and hematite. 
Sodiumfluosilicate may also be tried. Zeta potential studies in the 
presence of these modifiers or dispersants can clarify such positions. 
And also, a trivalent cation such as ferric ion must be tested. Ferric 
ion gave good results in Cosio's study (2). Once again, zeta potential 
studies are necessary to fathom the degree of influence.

Kemi slime (material 2)

The results of the pilot-plant studies in the presence of sodium 
metasilicate are presented in figure 15. In figure 16, a comparison 
is made between the pilot duplex concentrator performance and Mozley 
laboratory separator performance.

A recovery of 84% at a grade of 42% is obtained in the presence 
of a limited amount of sodium metasilicate. Irrespective of the tested 
levels of pulp densities the pilot test results are much better than the performance of the Jones separator at the Kemi slime circuit. The recovery at the slime circuit is about 46.9-56.49% at a grade of 37% Cr₂O₃ (Table XIII) with middling recirculation (35). A general comparison is made with the Sala-HGMS also on its performance with a 
-104 μm chromite feed (Table XIII) (36). The grades obtainable in gravity treatment of Kemi slimes should substantially increase if the feed is sized at a small mesh, say 45 μm.
Table XIII  Fine chromite recovery

<table>
<thead>
<tr>
<th>Source</th>
<th>Feed grade Cr₂O₃ (%)</th>
<th>Feed size μm</th>
<th>Stages of operation</th>
<th>Concentrate grade Cr₂O₃ (%)</th>
<th>Recovery (%)</th>
<th>Concentrate wt/ feed wt (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sala HGMS</td>
<td>14.0 **</td>
<td>-104 μm</td>
<td>two concentrate upgraded</td>
<td>38.7</td>
<td>77.1</td>
<td>30.30</td>
</tr>
<tr>
<td>Kemi Jones</td>
<td>16.6</td>
<td>90% -75μm</td>
<td>one middling recirculated</td>
<td>37.0</td>
<td>46.9</td>
<td>26.28</td>
</tr>
<tr>
<td>Slime Gravity</td>
<td>19.83</td>
<td>100% -75μm 84% -38μm</td>
<td>one no concentrate upgrading no middling recirculation</td>
<td>42.03</td>
<td>84.34</td>
<td>39.79</td>
</tr>
</tbody>
</table>

* maximum grade 67.90 Cr₂O₃ % (on the basis of FeO Cr₂O₃ )

** source of the feed unknown
CONCLUSIONS

The need to effectively manipulate the particlesurface characteristics in slime gravity processes has been stressed for sometime now. For example Burt and Stoelzle (37) firmly believed that this is one area giving scope for a major breakthrough in the increase of efficiency of gravity concentration and stressed the need for further research work. Osborne (38) considered that a slime gravity concentrator (The Bartles-Mozley separator) is highly suited to treatment of flocculated suspensions.

An attempt is made here, in this work, to fathom the importance of particle-surface characteristics in treating natural slimes. The study shows that the efficiency of a slime gravity process is dependent on the electrokinetic environment:
- appropriate dispersion improves the performance by preventing heteroaggregation,
- addition of appropriate kinds and amounts of inorganic electrolytes to natural systems improves, but on a limited scale, the performance through selective aggregation,
- it appears that the inorganic polymolecular complexes play their roles in a manner similar to that of the organic polyelectrolytes,
- nature of the aggregates (whether it is compact or loose and open) is a critical factor,
- detailed basic studies on the specific roles of relevant inorganic ions and complexes are necessary for further development.
REFERENCES


32. E.Forssberg and R.Sivamohan, Dynamic Flow Characteristics of Thin Films, Accepted for publication in Scan. Jour. of Metallurgy.


Fig. 1 Surface energy as a function of distance (after Van Olphen, Ref. 7).
Fig. 1 Surface energy as a function of distance (after Van Olphen, Ref. 7).
Fig. 2A Effect of Al\(^{3+}\), Cr\(^{3+}\) and Fe\(^{3+}\) ions on flotation of Albanian chromite
with anionic concentration of 10\(^{-4}\) m/l (pH 4.15 ± 0.15) curve 1, AlCl\(_3\); curve 2, CrCl\(_3\) or FeCl\(_3\) respectively (after Sobieraj and Laskowski, 1973. Ref. 17).
Fig. 2B Comparison of flotation-pH domain of Albanian chromite (curve 1) with flotation-pH domain in the presence of $10^{-4}$ m/l AlCl$_3$ (curve 2). Hatched area designates depression region. Sodium laurate collector at concentration $10^{-4}$ m/l (after Sobierajand Laskowski, 1973, Ref. 17).
Fig. 3 Effect of a high molecular weight anionic polymer (A 100, cyanamid) on the distribution of galena (feed wt. 6 grams; 100% -20 μm, 85% -10 μm), when treated on a slime gravity concentrator (Mozley laboratory separator).
Fig. 4  Zeta potentials of Yxsjöberg scheelite in distilled water and Mg salt solutions:

a. in distilled water only
b. in distilled water containing $2.46 \times 10^{-3}$ m/l MgCl$_2$
c. in distilled water containing $7.38 \times 10^{-3}$ m/l MgCl$_2$
d. in distilled water containing $1.00 \times 10^{-2}$ m/l MgCl$_2$
Fig. 5 Zeta potentials of Yxsjöberg scheelite in distilled water and pulp liquids:

a. in distilled water only
b. in the pulp liquid obtained by conditioning Yxsjöberg tailing at 45% solids in water at pH 11, adjusted by NaOH, for 48 hours
c. pulp liquid obtained in the same way as above but at natural pH
d. pulp liquid obtained in the same way as above(b) but adjusted by Ca(OH)$_2$
Fig. 6  Zeta potentials of Yxsjöberg scheelite in distilled water and pulp liquid:

a. in distilled water only

b. pulp liquid obtained by conditioning 150 grams of the -38 µm of the scheelite tailing in 850 ml of sodium metasilicate containing water, 10 ml of 1% solution, in the presence of 3.57 ml of $10^{-1}$ m/l NaF, 2 grams of $K_2CO_3$ and 1.07 grams of $MgCl_2 .6H_2O$ for 48 hours at a pH of about 10.45.
Fig. 7  Zeta potentials of calcite in distilled water and $K_2CO_3$ and Mg salt solutions:
   a. in distilled water only
   b. in distilled water containing $2.5\times10^{-3}$ m/l $K_2CO_3$
   c. in distilled water containing $3.6\times10^{-3}$ m/l $K_2CO_3$
   d. in distilled water containing $3.6\times10^{-3}$ m/l $K_2CO_3$ and $2.46\times10^{-3}$ m/l $MgCl_2$
Fig. 8 Zeta potential of Kemi chromite in distilled water and Al salt solutions:

a. in distilled water
b. in distilled water containing $1.5 \times 10^{-4}$ m/$l$ Al$_2$(SO$_4$)$_3$

c. in distilled water containing $1.5 \times 10^{-3}$ m/$l$ Al$_2$(SO$_4$)$_3$

d. in distilled water containing $1.5 \times 10^{-2}$ m/$l$ Al$_2$(SO$_4$)$_3$
Fig. 9 Zeta potentials of Geevor cassiterite in distilled water
Fig. 10 Zeta potentials of hematite in distilled water and Al and Cu salt solutions:

a. in distilled water only
b. in distilled water containing $1.5 \times 10^{-4}$ m/l $\text{Al}_2(\text{SO}_4)_3$
c. in distilled water containing $8.0 \times 10^{-4}$ m/l $\text{CuSO}_4$
Fig. 11 Effect of pH on the efficiency of concentration of Yxsjöberg scheelite tailing
Fig. 12 Effect of electrokinetic environment on the recovery of scheelite from Yxsjöberg tailing
Fig. 13A  Effect of Al salt on the recovery of chromite, in the concentrate and the middling (Kemi slime, material 1)
Fig. 13B Effect of Al salt on the grade of chromite, in the concentrate and the middling (Kemi slime, material 1)
Fig. 14 Effect of electrokinetic environment on the recovery and grade of cassiterite from Geevor tailing
Fig. 15 Pilot Duplex concentrator performance on silicate treated Kemi chromite slime at different densities
Fig. 16 Comparison of pilot Duplex concentrator performance with Mozley laboratory separator