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RAPID FLOTATION USING A MODIFIED BUBBLE-INJECTED HYDROCYCLONE AND A SHALLOW-DEPTH FROTH SEPARATOR FOR IMPROVED FLOTATION KINETICS

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ABSTRACT

As a part of the Bureau of Mines' efforts to improve the efficiency of the United States' domestic minerals industry, the Bureau has developed a rapid froth flotation system which divides flotation into two discrete unit operations: bubble-particle attachment and bubble-pulp separation. A modified bubble-injected hydrocyclone developed by the Bureau of Mines was used as the bubble-particle attachment unit which mixed a bubble slurry with an ore slurry under highly turbulent conditions. Then the mixture immediately flows into a relatively quiescent froth separation unit where the bubbles quickly separated from the ore pulp. A shallow-depth froth separator was used to minimize the rising distance required to recover even the smallest size bubbles (100 μm) and to quickly recover the mineral laden bubbles. The tailings pulp only remained in the froth separator long enough to recover the bubbles and then exited through the conical bottom. Combining these two units, a rapid flotation system was formed that successfully floated silica from phosphate in one ninth of the retention time for conventional mechanical cells and floated coal from ash in one eighteenth of the retention time for conventional mechanical cells. The hydrodynamics of the rapid flotation system along with fundamental parameters for scale-up are presented in this paper.

INTRODUCTION

Conventional froth flotation is one of the most widely used mineral beneficiation processes. Unfortunately, long residence times are often required to achieve complete flotation [3]. These slow flotation kinetics have been associated with the hydrodynamics in the flotation cell. While flotation cell manufacturers have optimized the mixing hydrodynamics, little effort has been made to improve the flotation hydrodynamics within the flotation cell [1].

Conventional flotation has both a turbulent region, where bubble-particle attachment occurs, and a quiescent region, where the ore-laden froth is removed from the cell [5]. The flotation rate of a single particle in this environment is a function of the probabilities of bubble-particle collision, attachment, and the motion of the bubble moving to the froth [6]. To increase the flotation rate of a flotation cell the number of bubble-particle collisions must be increased. One way to increase the number of collisions is to increase the number of bubbles either by decreasing the bubble size or by increasing the air flow rate to the flotation cell. Unfortunately, the bubble size is limited by the mechanical mechanism in a conventional cell and, at high air flow rates, the mechanical cell's agitation is less effective at breaking up the air into small bubbles. Another method to increase the number of collisions is to raise the turbulence within the cell by elevating the fluid velocity [2,3]. Increasing the fluid velocity can increase the relative velocities of the bubbles to the particles and increase the probability of bubble-particle collision. However, there are problems with increasing the agitation in the flotation cell. Increasing the turbulence can also cause entrainment, where the bubble-particle aggregate gets carried out with the tailings, or increasing the turbulence can cause detachment, where the turbulent conditions dislodge the particle from the bubble. Thus, the process of attaching the particle to the bubble and the subsequent separation of the bubble from the pulp interact significantly in a conventional flotation cell, making it virtually impossible to optimize one of the processes without sacrificing the effectiveness of the other.

As a part of the Bureau of Mines' efforts to improve the efficiency of the domestic minerals industry, the Bureau has developed a rapid froth flotation system which separates flotation into two discrete unit operations: bubble-particle attachment and bubble-pulp separation (Figure 1) [5]. In the bubble-particle attachment unit, a bubble slurry is mixed with an ore slurry under highly turbulent conditions. The mixture is then rapidly fed into a quiescent bubble-pulp separation unit where the bubbles quickly separate from the ore pulp.

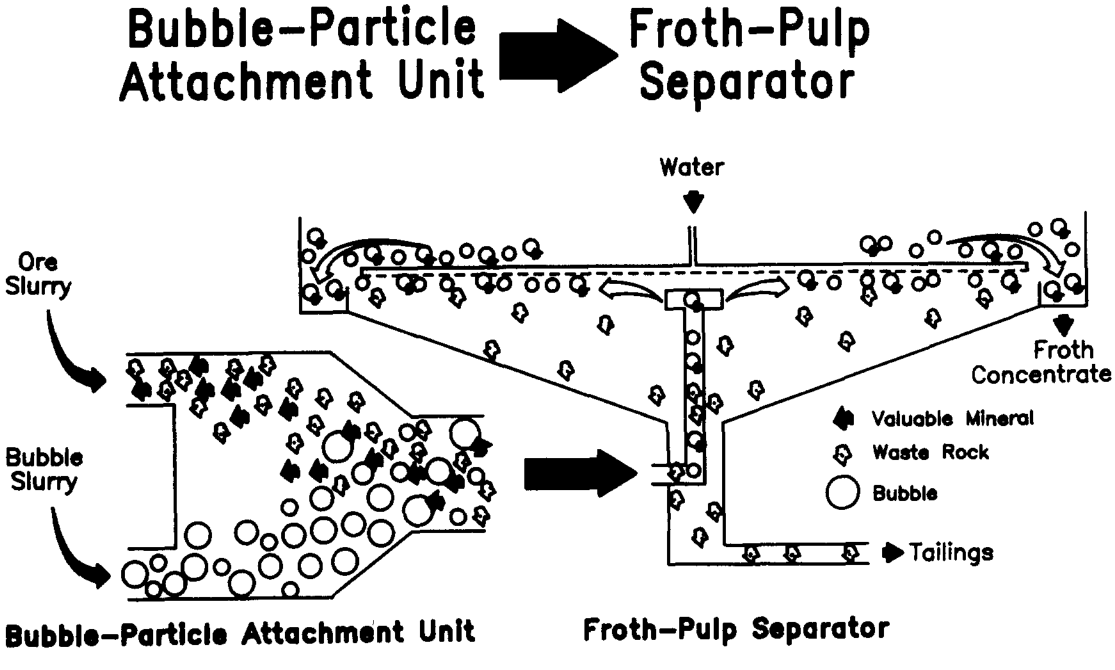


Fig.1 Schematic diagram of the rapid flotation system.

In this investigation, a modified bubble-injected hydrocyclone developed by the Bureau of Mines [5,6] was evaluated as a bubble-particle attachment unit (Figure 2). The bubble-injected hydrocyclone is similar to a conventional hydrocyclone. The ore slurry enters the cyclone tangentially at the top of the cyclone. The cyclone has a second tangential entry port in the form of a slit running the length of the cylindrical section of the cyclone. A bubble slurry is pumped from a bubble generator and enters through the slit-shaped port. The cyclonic action of the fluid forces the bubbles inward toward the center and the ore particles outward toward the wall. The bubbles and particles pass through each other at high velocities, causing rapid bubble-particle contact. The froth and pulp phases exit through the apex while still mixing vigorously. The froth-pulp mixture immediately enters the shallow-depth froth separator (Figure 3). A sparger at the top of the unit disperses the froth-pulp mixture which provides the quiescent conditions necessary for adequate bubble-pulp separation. The relatively large diameter froth separator has a nominal depth of flow (approx. 2 cm) which minimizes the rising time of even the smallest bubbles (100 μm). As the material moves outward from the sparger, the ore-laden bubbles quickly rise to the top of the froth layer and overflow at the edge of the separator. The pulp moves out the bottom of the unit.

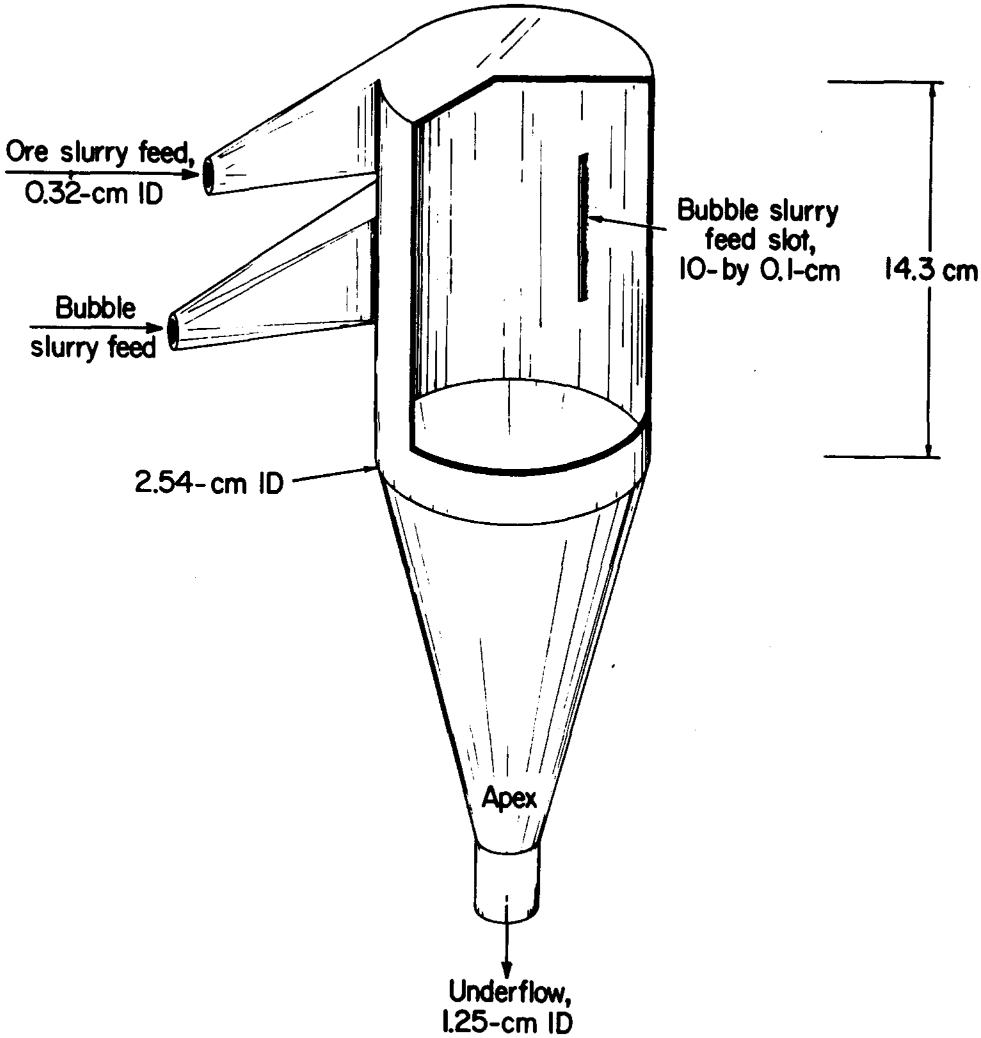


Fig.2 Modified bubble-injected hydrocyclone for use as a bubble-particle attachment unit.

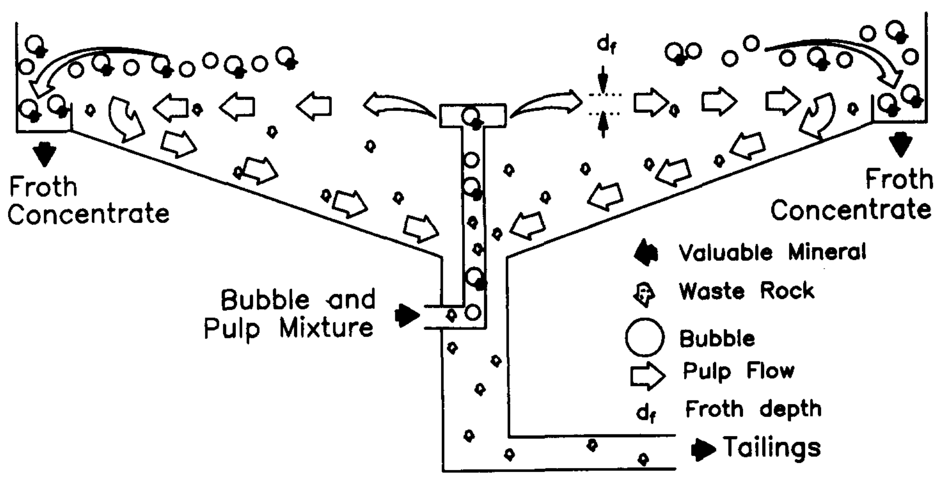


Fig.3 Schematic diagram of the shallow-depth separator and slurry flow characteristics in the shallow-depth separator.

Rapid flotation often requires multiple stages because adequate mineral recovery is not always obtained after one stage. For the second stage, the tailings product from the first stage was dewatered and then fed through the system for the second stage. This sequence was repeated for any number of stages. The concentrates from each stage can be combined to form a final concentrate for the multi-stage flotation process.

Mineral Samples and Reagents

To study the rapid flotation system two mineral systems were evaluated. To study the response of coarse particles, a sample of amine phosphate tailings was tested to rapidly float quartz away from the phosphate. For fine particles, a coal slurry was tested to float the coal from the ash minerals. The phosphate-bearing amine flotation tailings sample was obtained from a Florida phosphate operation and contained 9.5 pct P_2O_5 in a quartz matrix. The particle size of the sample ranged between 300 and 38 μm with the P_2O_5 evenly distributed among the different particle sizes. The sample was conditioned at pH 6.5 with cornstarch to depress the phosphate and then conditioned with Armac C†, an aliphatic amine acetate salt, to float the quartz. Betz M150 frother was used to stabilize the bubbles. The flotation goal was to float a quartz product, low in phosphate, that could be discarded as tailings and to obtain a phosphate-rich product which would be suitable for recycling to the phosphate flotation circuit.

The rapid flotation system was also tested with a fine coal sample from a coal cleaning operation. The slurry was 4.5 pct solids and the coal contained 26 pct ash. Only 33 pct of the material was greater than 38 μm size with the ash evenly distributed among the particle sizes. The slurry was allowed to settle and was dewatered to 10 pct solids for handling purposes. The "as received" pH of the slurry was 7.5 and that pH was maintained throughout the flotation testing. Fuel oil was used as the collector and MIBC frothing agent was used to stabilize the bubbles.

Amine Tailings Flotation Procedure

The flotation system consisted of a slurry reservoir, a bubble generator, a bubble-particle attachment unit and a froth separator. Water used throughout the test was adjusted to the proper pH by adding hydrochloric acid or sodium hydroxide and was treated with the appropriate amount of frothing agent. The amine tailings sample was first conditioned with cornstarch for 30 s to depress the phosphate and then immediately conditioned with Armac C for an additional 30 s to float the quartz. The conditioned sample was placed in the slurry reservoir and diluted to the proper percent solids. The ore pulp was pumped into the bubble-injected bubble-particle attachment unit. The bubble-particle attachment unit was a modified bubble-injected hydrocyclone with an inner diameter of 2.5 cm, a cone angle of 85° and an overall length of 26 cm (Figure 2). The ore slurry entered the chamber through a 0.38 cm ID inlet while the bubble slurry entered through a slot along the length of the cyclone. The dimensions of the slot were 0.1 cm wide and 10 cm in length. The vortex finder of the cyclone was capped off to force the bubble-particle slurry to exit through the apex.

The bubble generator was a conventional flotation cell (10 L) with the normal impeller replaced by a 16.5 cm diameter, high-speed spinning disk rotating at 3,300 rpm. Air striking the spinning disk was quickly transformed into small sized bubbles approximately 100 μm size. These bubbles were significantly smaller than those generated from a conventional flotation cell. The frother was used in the bubble generator to stabilize these small bubbles.

†Reference to specific products does not imply endorsement by the Bureau of Mines.

The bubble slurry was pumped from the bubble generator. The water to ore ratio in the bubble-injected hydrocyclone bubble-particle attachment unit was equivalent to 10 pct solids for all the tests. At different aeration levels, the ore slurry percent solids was adjusted so that when combined with the bubble slurry, the percent solids in the bubble-injected hydrocyclone bubble-particle attachment unit was maintained at 10 pct solids. The bubble-pulp mixture was then sent to the froth separator where the concentrate and tailings products were separated. Three different sized shallow-depth froth separators were employed (Figure 3). The bubble-pulp mixture entered the unit through a sparger which was centrally located near the top of the cone shaped tank. The bubbles quickly rose to the surface as the slurry spread out toward the side of the tank. The froth overflowed at the outer edge and the tailings pulp drained out of the unit through the bottom. Unit 1 had a 160 cm^2 surface area and an effective volume of 0.8 L. Unit 2 had a 314 cm^2 surface area and an effective volume of 1.7 L. Unit 3 had a 670 cm^2 surface area and an effective volume of 5.2 L.

The flotation process was continued until the ore slurry in the reservoir was depleted. At that time, 2 L of wash water was added to the ore reservoir to flush the system of any remaining ore. The concentrate and tailings were dried, weighed, and analyzed for phosphate and silica. The flotation conditions are summarized in table 1.

TABLE 1 Reagent and conditioning scheme for phosphate flotation

Ore charge........................g	500
pH................................	6.5
Frothing agent....................	Betz M150
Frothing agent concentration....ppm	25
Air in bubble generator.........pct	35
Conditioner #1....................	Cornstarch
Concentration..................g/kg	.25
Conditioning time..............min	.5
Conditioner #2....................	Armac C
Concentration..................g/kg	.38
Conditioning time..............min	.5
Solids during conditioning......pct	75

Conventional flotation tests were also carried out in a Denver flotation cell in order to compare the results with those obtained from the rapid flotation technique. The cell was set at 1,100 r/min and air was introduced at 4 L/min. Identical reagent and pH conditions were used. Timed samples of the flotation concentrate were taken up to 10 min to determine the flotation kinetics of the amine tailings sample in the Denver flotation cell.

Coal Flotation Procedure

The same flotation system was used for the coal flotation tests. The coal sample was conditioned with fuel oil for 10 minutes. The conditioned pulp was placed in the slurry reservoir and diluted to the proper percent solids. The coal pulp was pumped into the bubble-injected hydrocyclone bubble-particle attachment unit and mixed with the bubble slurry which was pumped from the bubble generator. Inside the bubble-injected hydrocyclone bubble-particle attachment unit the water to coal ratio was equivalent to 4.5 pct solids for all the tests. The bubble pulp mixture was then sent to the froth separator where the concentrate and tailings products were separated. The flotation process was continued until the ore slurry in the reservoir was depleted. Again, 2 L of wash water was added to the reservoir to flush the system of any remaining ore.

Since the coal was not completely floated after the first stage, multiple passes through the system were required. To accomplish this, the first stage tailings were dewatered and returned to the ore slurry reservoir at the appropriate percent solids and the flotation sequence was re-started. This sequence was repeated for as many as six stages. The coal was not re-conditioned with additional reagent for any of these multi-stage flotation tests. The concentrates and tailings were dried, weighed and analyzed for percent ash. Table 2 lists the coal flotation conditions.

TABLE 2 Reagent and conditioning information for coal flotation

Ore charge......................g	225
pH............................	7.5
Frothing agent..................	MIBC
Frothing agent concentration..ppm	20
Air in bubble generator.......pct	20
Conditioner......................	Fuel oil
Concentration................g/kg	0.33
Conditioning time.............min	10
Solids during conditioning....pct	10

Conventional coal flotation tests were also carried out in a Denver flotation cell in order to compare the results with those obtained from the rapid flotation technique. The cell was set at 1,100 rpm and air was introduced at 4 L/min. Identical reagent and pH conditions were used. Timed samples of the flotation concentrate were taken up to 10 minutes to determine the flotation kinetics of the coal in the Denver flotation cell.

Amine Phosphate Tailings Flotation

Experimental Design

Flotation tests were designed to study the effect of mixing intensity in the bubble-injected hydrocyclone bubble-particle attachment unit, the residence time of the bubbles in the froth separator, and the air to ore ratio. Three different flow rates were used to represent different mixing intensities within the bubble-injected hydrocyclone. The fluid head loss through the system was measured to determine the power consumed by the mixing action of the bubble-injected hydrocyclone and the sparger of the froth separator.

The power consumed by the bubble-particle attachment unit, a measure of the mixing intensity, was nearly proportional to the square of the total flow rate through the system. To independently measure the effect of mixing intensity, the subsequent separation of the bubbles from the pulp was conducted under virtually the same hydrodynamic conditions in the froth separator. As the mixture flowed into the froth separator, shown in figure 3, the pulp moved radially from the center of the unit towards the outer edge. The depth of this horizontal flow was fairly shallow. The effective volume of the froth separator was the disk shaped region at the top and was roughly equal to the surface area times the depth of the flow. Bubbles must rise out of the pulp during the time that the pulp is in this effective region. This time shall be referred to as the effective bubble residence time, because any bubbles that have not risen to the surface by the time the pulp reaches the outer edge of the froth separator will be swept down with the tailings as the pulp exits the unit. The effective bubble residence time becomes:

$$t_b = d_f \cdot A_s/Q \qquad (1)$$

where d_f is the depth of the flow, A_s is the surface area of the froth separator, and Q is the volume flow rate. The depth of the effective volume would largely depend upon the type of sparger used to distribute the flow radially. By using the same sparger throughout the test work, the effective depth would remain relatively constant even as the flow rate varied. Therefore, for this research, the same sparger was used at every flow rate and with every froth separator. This kept the d_f relatively constant as the pulp radiated from the sparger.

To maintain similar hydrodynamics within the froth separator, the effective bubble residence time also needed to be constant. With the sparger holding the d_f constant, the ratio of the froth separator's surface area to the total flow rate was proportional to the effective bubble residence time. Keeping that ratio constant would also keep the effective bubble residence time constant. Three froth separators were constructed with surface areas of 160, 314, and 670 cm^2. Corresponding flow rates of 5.1, 10.0, and 21.3 L/min were selected to maintain the same ratio of surface area to flow rate, which reflected a constant effective bubble residence time within the three froth separators. Therefore, the three different total flow rates, representing three mixing intensities in the bubble-injected hydrocyclone bubble-particle attachment unit, were tested with the same effective bubble residence time in the froth separator.

Three air to ore ratios of 1.5, 2.5, and 3.5 mL/g were also tested in the bubble-injected hydrocyclone bubble-particle attachment unit. The air to ore ratio could be adjusted by changing the relative proportions of the ore pulp and bubble slurry being used at each total flow rate. A three by three factorially designed experiment shown in table 3 was conducted to determine the effect of mixing intensity at three levels and air to ore ratio at three levels in the bubble-injected hydrocyclone bubble-particle attachment unit. The factorial design will simultaneously study the effects of mixing intensity and air to ore ratio at all possible combinations of the levels [5].

TABLE 3 Operating conditions for phosphate flotation tests

Air/ore, mL/g	Total flow rate, L/min	Mixing intensity power, W	Froth separator surface area, cm^2	Surface area to flow rate, $cm^2 \cdot min/L$
1st experimental design holding surface area to flow rate constant				
1.5	5.1	4.1	160	31
1.5	10.0	15.3	314	31
1.5	21.3	144.0	670	31
2.5	5.1	4.0	160	31
2.5	10.0	17.0	314	31
2.5	21.3	149.8	670	31
3.5	5.1	3.9	160	31
3.5	10.0	18.5	314	31
3.5	21.3	156.0	670	31
2nd experimental design holding mixing intensity constant				
1.5	10.0	15.3	160	16
1.5	10.0	15.3	314	31
1.5	10.0	15.3	670	67
2.5	10.0	17.0	160	16
2.5	10.0	17.0	314	31
2.5	10.0	17.0	670	67
3.5	10.0	18.5	160	16
3.5	10.0	18.5	314	31
3.5	10.0	18.5	670	67

Another three by three factorially designed experiment shown also in table 3 was conducted to determine the effect of three different effective bubble residence times at constant mixing intensity in the bubble-injected hydrocyclone bubble-particle attachment unit and at the three different air to ore ratios. This was accomplished by maintaining the total flow rate at 10 L/min and testing each of three different size froth separators. Again, the effective bubble residence time was proportional to the ratio of surface area to total flow rate. As the air to ore ratio varied, the intensity within the bubble-injected hydrocyclone attachment unit also changed. The relative proportion of the bubble slurry to the total flow altered the mixing intensity slightly. However, at the same air to ore ratio, the mixing intensity remained relatively constant for all three effective bubble residence times.

RESULTS AND DISCUSSION

Although the goal of this flotation process is recovery of phosphate, the flotation rate is a function of the quartz recovery because the quartz is being floated. To determine the flotation rate, the first order kinetics equation was used:

$$\text{Recovery} = 100 - 100\, e^{-kt_r} \tag{2}$$

where k is the first order flotation rate constant and t_r is the residence time of a particle in the flotation system. Solving the equation for the flotation rate,

$$k = \{\ln(100/[100 - \text{Recovery}])\} / t_r \quad \text{min}^{-1} \tag{3}$$

The speed of flotation among different tests would be reflected in the first order flotation rate constant calculated for each test.

The results of the first factorial experimental design, where the froth separator surface area to flow rate ratio was held constant, are shown in table 4 and figure 4. Analysis of variance of the concentrate grade, quartz recovery, selectivity index, and quartz flotation rate was conducted and, as shown in table 5, the statistical f-test at a 95-pct confidence was performed to determine the significance of the results. The mixing intensity within the bubble-injected hydrocyclone bubble-particle attachment unit had a significant effect upon the silica grade of the quartz concentrate. In general, the highest grade of the quartz concentrate was obtained at the lowest mixing intensity. The effect of air to ore ratio was also statistically significant and the highest grade of the quartz concentrate was obtained at the 1.5 mL/g air to ore ratio. However, a first level interaction between the mixing intensity and the air to ore ratio was also statistically significant indicating that both factors react to the level of the other factor. This interaction is evident when both the mixing intensity and the air to ore ratio were at the lowest level but the silica grade of the quartz concentrate was not the best. However, if only one parameter, the mixing intensity or the air to ore ratio, was at the lowest level, then the silica grade of the quartz concentrate was maximized. At the low mixing intensity, 4 W, and 2.5 mL/g air to ore ratio, a 91.2 pct silica concentrate was obtained, and at the low air to ore ratio, 1.5 mL/g, and 15 W mixing intensity, a 90.5 pct silica concentrate was obtained.

Statistically, the mixing intensity had no significant effect upon the amount of quartz recovered. The variations in the quartz recovery were not that much different than the experimental error. However, the air to ore ratio had a significant effect upon the quartz recovery. The best quartz recovery was obtained at the 2.5 mL/g air to ore ratio. As the amount of air to ore increased, the number of bubbles also increased which resulted in more bubble-particle attachments and a higher recovery of the quartz. However, at the 3.5 mL/g air to ore ratio, the quartz recovery dropped. This was probably a function of the bubble generator. The bubbles were continuously generated in a constant volume unit and only a portion of the bubble slurry was pumped to the bubble-injected hydrocyclone bubble-particle attachment unit. As the air to ore ratio increased, the flow of water and air to the bubble generator also increased. While the percentage of air in the bubble slurry remained at 35 pct volume, the bubble sizes were obviously larger. Less agitation energy was imparted to each unit volume of water and air at these higher flow rates and the larger bubbles were less effective resulting in a decline in the quartz recovery at the high air to ore ratio.

TABLE 4 Concentrate grade, recovery, selectivity index (S.I.) and flotation rate for various air/ore ratios and total flow rates using the phosphate ore and holding the surface area to flow rate ratio constant†

Air/ore, mL/g	Total flow rate, L/min	Mixing intensity power, W	Concentrate grade, pct silica	Recovery, pct	S.I.††	Flotation rate, min^{-1}
1.5	5.1	4.1	85.5	61.2	2.06	3.55
1.5	10.0	15.3	90.5	51.3	2.93	3.22
1.5	21.3	144.0	85.6	47.7	1.99	2.39
2.5	5.1	4.0	91.2	63.6	2.89	3.79
2.5	10.0	17.0	81.7	79.4	2.51	6.99
2.5	21.3	149.8	78.1	70.2	2.23	4.47
3.5	5.1	3.9	84.8	45.6	1.80	2.28
3.5	10.0	18.5	83.7	52.0	1.91	3.21
3.5	21.3	156.0	81.3	35.2	1.58	1.61

†Froth separator surface area to flow rate = 31 $cm^2 \cdot min/L$.

‡Selectivity Index = $\sqrt{\frac{(Qartz\ Recovery) \cdot (Phosphate\ Rejection)}{(Phosphate\ Recovery) \cdot (Quartz\ Rejection)}}$

Throughout the various tests the grade and recovery displayed the typical inverse relationship where an increase in grade resulted in a lower recovery and vice versa. The selectivity index was calculated for each test to quantitatively judge between the various tests [4]. The selectivity index is defined as the geometric mean of the relative recoveries and relative rejection of two minerals in a separation process. The relative recovery is the ratio of quartz recovery in the quartz concentrate divided by phosphate recovery in the quartz concentrate. The relative rejection is the ratio of phosphate recovery in the unfloated product divided by the quartz recovery in the unfloated product. Unfortunately, there was not a statistically significant difference between the mixing intensities or the air to ore ratios when the selectivity index was evaluated. The experimental error masked any effect of these parameters.

TABLE 5 Analysis of variance results from the statistical f-test for significance at a 95-pct confidence level of the concentrate grade, recovery, selectivity index (S.I.), and flotation rate for various air/ore ratios and total flow rates holding the surface area to flow rate ratio constant†

Factor	Concentrate grade, pct silica	Recovery, pct	S.I.	Flotation rate, min^{-1}
Effect of mixing intensity	Sgnt‡	Not§	Not	Not
Effect of air to ore ratio	Sgnt	Sgnt	Not	Sgnt
Interaction of mixing intensity to air to ore ratio	Sgnt	Not	Not	Not

†Froth separator surface area to flow rate = 31 $cm^2 \cdot min/L$.
‡Significant effect.
§No significant difference as compared with the experimental error.

Only the air to ore ratio had a statistically significant effect upon the flotation rate. The fastest flotation rate was obtained at the 2.5 mL/g air to ore ratio. Selective flotation of quartz without floating the phosphate was accomplished by maximizing the flotation rate of quartz while still maintaining a low phosphate flotation rate. The best flotation at the fastest quartz flotation rate occurred at the 2.5 mL/g air to ore ratio and at 4 W mixing intensity (5.1 L/min flow rate). A 91.2 pct silica concentrate was obtained at a 3.79 min^{-1} flotation rate and with 63.6 pct recovery of the quartz. At the 4 W mixing intensity in the bubble-injected hydrocyclone bubble-particle attachment unit, the quartz flotation rate was high enough to recover 63.6 pct of the quartz. But that mixing intensity was also low enough to keep the phosphate flotation rate low and produce a high grade quartz flotation concentrate.

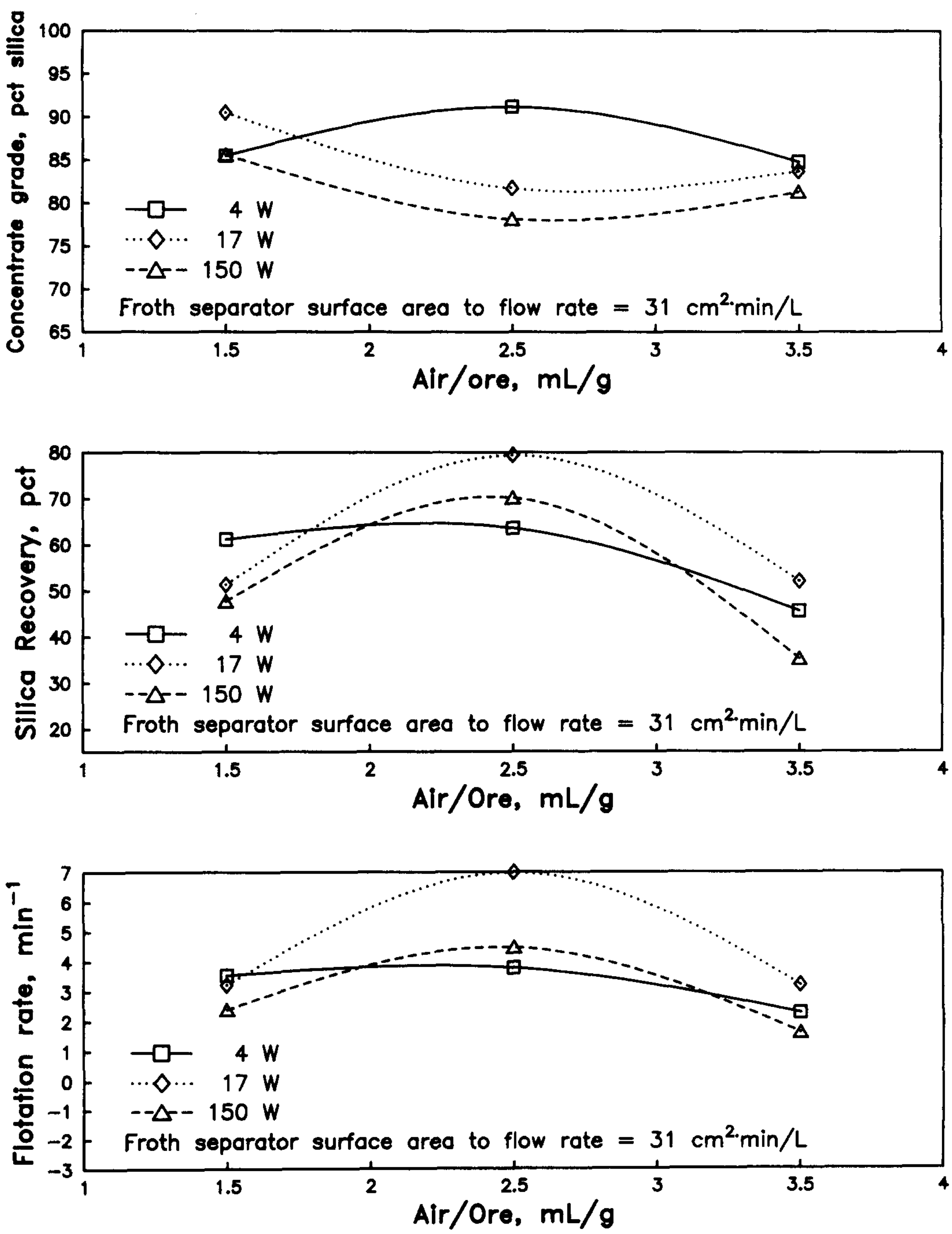


Fig.4 Concentrate grade, recovery and flotation rate for various air/ore ratios and total flow rates using the amine phosphate tailings sample and holding the surface area to flow rate ratio constant.

The results of the factorial design experiments to measure the effect of the bubble residence time at a constant mixing intensity are shown in tables 6 and 7. Figure 5 displays the results graphically. Both bubble residence time and air to ore ratio had a statistically significant effect upon the quartz concentrate grade. The best grade was obtained at the bubble residence time corresponding to 31 $cm^2 \cdot min/L$ surface area to flow rate ratio and at the 1.5 mL/g air to ore ratio. As the air to ore ratio increased, the flotation rate of phosphate also increased, thereby lowering the quartz concentrate grade. While there is a measured statistically significant difference in the quartz concentrate grade for all three bubble residence times, the actual values were relatively close.

TABLE 6 Concentrate grade, recovery, selectivity index (S.I.) and flotation rate for various air/ore ratios and surface area/flowrate ratios holding the mixing intensity constant†

Air/ore, mL/g	Surface area to flow rate, $cm^2 \cdot min/L$	Concentrate grade, pct silica	Quartz recovery, pct	S.I.	Quartz flotation rate, min^{-1}
1.5	16	87.8	59.0	2.64	6.62
1.5	31	90.5	51.3	2.93	3.22
1.5	67	83.8	68.8	2.44	2.04
2.5	16	82.8	75.8	2.29	10.43
2.5	31	81.7	79.4	2.51	6.99
2.5	67	83.4	57.3	1.94	1.48
3.5	16	79.8	65.0	1.70	7.64
3.5	31	83.7	52.0	1.91	3.21
3.5	67	83.8	24.4	2.01	.48

†Mixing intensity power ave = 17.0 W, total flow rate = 10 L/min.

TABLE 7 Analysis of variance results from the statistical f-test for significance at a 95-pct confidence level of the concentrate grade, recovery, selectivity index (S.I.), and flotation rate for various air/ore ratios and surface area to flow rate ratios holding the mixing intensity constant†

Factor	Concentrate grade, pct silica	Recovery, pct	S.I.	Flotation rate, min^{-1}
Effect of bubble residence time	Sgnt‡	Not§	Not	Sgnt
Effect of air to ore ratio	Sgnt	Sgnt	Not	Sgnt
Interaction of bubble residence time to air to ore ratio	Sgnt	Not	Not	Not

†Mixing intensity power ave = 17.0 W, total flow rate = 10 L/min.
‡Significant effect.
§No significant difference as compared with the experimental error.

Only the air to ore ratio had a statistically significant effect upon the quartz recovery. The best quartz recovery was obtained at the 2.5 mL/g air to ore ratio. The 2.5 mL/g air to ore ratio produced more bubbles than the 1.5 mL/g air to ore ratio which resulted in the increased quartz recovery. However, the 3.5 mL/g air to ore ratio did not increase the quartz recovery, because the bubble size appeared to increase at this high air flow rate.

Again, the selectivity index was not significantly affected by either the bubble residence time or the air to ore ratio. However, there was the typical inverse relationship between the concentrate silica grade and the quartz recovery. The flotation rate was significantly affected by both the bubble residence time and the air to ore ratio. Because there was no significant difference in the quartz recovery due to the bubble residence time, only the volume of the froth separator caused the flotation rate to change. As expected the small volume froth separator with its short residence time had the largest flotation rate. The fastest flotation rate was obtained at the 2.5 mL/g air to ore ratio.

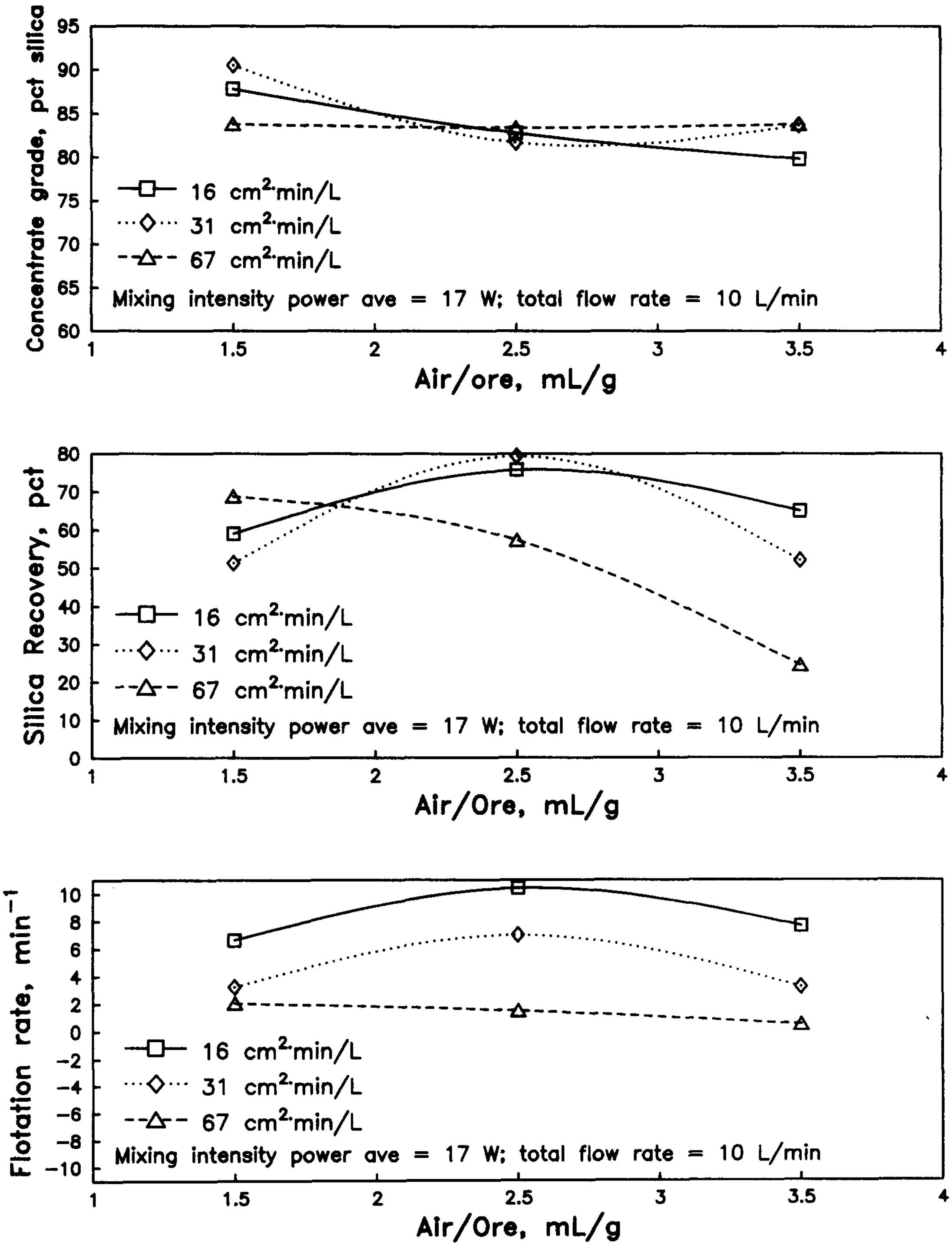


Fig.5 Concentrate grade, recovery, and flotation rate for various air/ore ratios and surface area/flow rate ratios holding the mixing intensity constant.

While the flotation process recovered a quartz concentrate, the goal of this flotation was to rapidly obtain a high-grade quartz flotation concentrate to discard as a tailings and to recover most of the phosphate in the unfloated product for recycling to the phosphate flotation circuit. The best quartz grade products were obtained using either the 1.5 mL/g air to ore ratio at 15 W mixing intensity level with the 160 cm^2 surface area froth separator, the 1.5 mL/g air to ore ratio at the 15 W mixing intensity level with the 314 cm^2 surface area froth separator, or the 2.5 mL/g air to ore ratio at the 4 W mixing intensity level with the 314 cm^2 surface area froth separator. The fastest flotation rate between those three conditions was obtained at the 1.5 mL/g air to ore ratio at 15 W mixing intensity level with the 160 cm^2 surface area froth separator. About 59 pct of the quartz was rapidly floated at a 6.62 min^{-1} flotation rate. The unfloated product contained 15 pct P_2O_5 and recovered 82 pct of the phosphate. Table 8 compares the results of the best rapid flotation test with the results of the conventional silica flotation test. The silica grade of the flotation concentrate was the same for both the rapid and conventional flotation systems. The phosphate recovery was higher for the rapid flotation system than for the conventional flotation system, but the phosphate product grade was lower for the rapid flotation system than for the conventional flotation system. The lower grade phosphate product is not a problem because this product will be recirculated to the phosphate recovery circuit. However, the biggest difference between the two techniques was the flotation speed. The capacity (tons per hour per cubic meter) of the rapid flotation system was over nine times larger than the conventional flotation system so that a physically smaller rapid flotation unit could be used to replace a larger conventional flotation circuit of the same capacity.

TABLE 8. Comparison of the rapid flotation system and conventional laboratory flotation of the amine tailings sample

	Flotation system	
	Rapid	Conventional
Discard product.........pct silica	87.8	87.3
Discard product...........pct P_2O_5	3.8	3.7
Phosphate product.........pct P_2O_5	14.5	24.0
Phosphate recovery.............pct	82.1	71.3
Quartz flotation rate........min^{-1}	6.62	1.25
Flotation capacity..........$t/h/m^3$	39.5	4.3

Coal Flotation

Experimental Design

The coal flotation tests were also designed to study the effect of mixing intensity in the bubble-injected hydrocyclone bubble-particle attachment unit, the residence time of the bubbles in the froth separator, and the air to ore ratio. Three different flow rates (5.1, 10.0, and 21.3 L/min) were used to represent different mixing intensities within the bubble-injected hydrocyclone. Two air to ore ratios (1.0 and 1.5 mL/g) were also investigated. The first series of coal flotation tests was run at different mixing intensities while keeping the conditions in the froth separator relatively constant by maintaining a constant froth separator surface area flow rate ratio. The second series of tests was run at a constant mixing intensity, but at three different effective bubble residence times. The operating conditions are summarized in table 9.

Results and Discussion

Again, the speed of flotation among different tests would be reflected in the first order flotation rate constant calculated for each test. A single stage rapid coal flotation was not possible in the rapid flotation system, because the level of coal recovery was too low. Multiple flotation stages were required to obtain at least 90 pct recovery of the coal. Each flotation stage added to the total residence time of the flotation process so that the plot of coal recovery versus time shown in figure 6 would reveal the flotation rate. The curve appears to fit the first order kinetics equation. However, rather than attempting to fit each test's recovery versus time curve to the first order equation, the first order rate constant at 90 pct coal recovery was calculated using equation 2 and the residence time for 90 pct coal recovery. This K_{90} value would be used throughout the various coal flotation tests as a measurement of the flotation speed.

TABLE 9 Operating conditions for coal flotation tests

Air/ore, mL/g	Total flow rate, L/min^{-1}	Mixing intensity power, W	Froth separator surface area, cm^2	Surface area to flow rate, $cm^2 \cdot min/L$
Experiments holding surface area to flow rate constant				
1.0	5.1	4.6	160	31
1.0	10.0	49.0	314	31
1.0	21.3	242.2	670	31
1.5	5.1	4.3	160	31
1.5	10.0	49.1	314	31
1.5	21.3	241.1	670	31
Experiments holding mixing intensity constant				
1.0	10.0	49.0	160	16
1.0	10.0	49.0	314	31
1.0	10.0	49.0	670	67
1.5	10.0	49.1	160	16
1.5	10.0	49.1	314	31
1.5	10.0	49.1	670	67

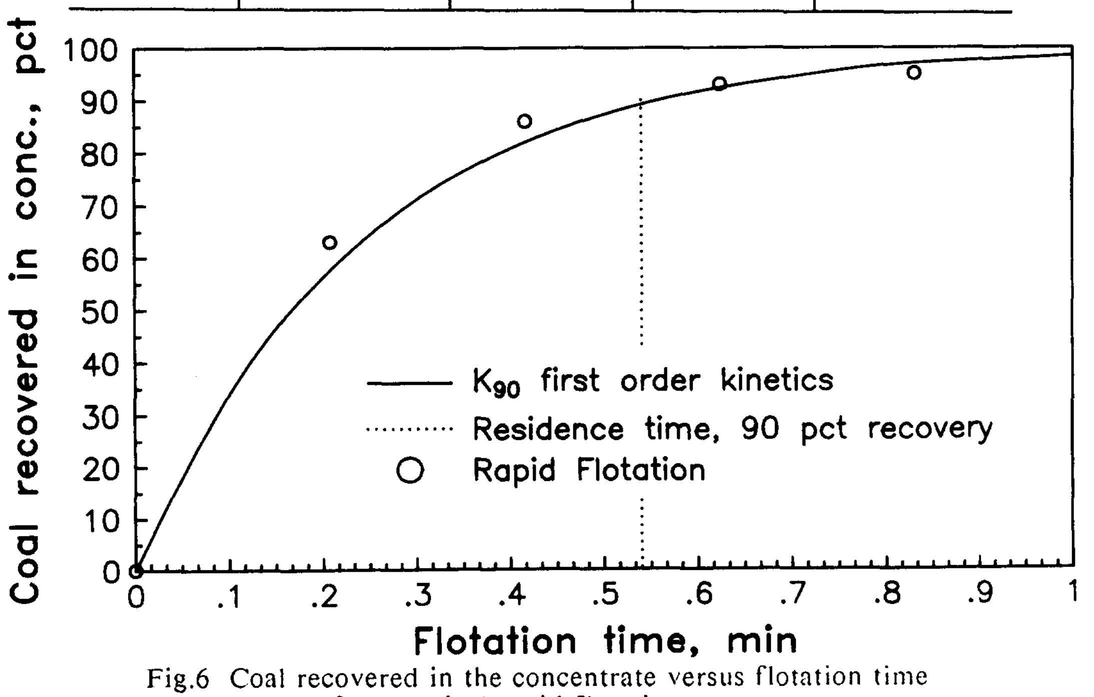


Fig.6 Coal recovered in the concentrate versus flotation time for a typical rapid flotation test.

The results of the typical rapid flotation test shown in figure 6 are also shown in table 10. The first stage recovered 63 pct of the coal in a concentrate containing only 6.3 pct ash. The second stage recovered an additional 23 pct of the coal in a concentrate containing 6.1 pct ash. Combining these two concentrates, the two stage system recovered 86 pct of the coal in a concentrate containing 6.3 pct ash. While the second stage only recovered 23 pct of the coal, the second stage actually recovered 62 pct of the coal that was still present in the first stage tailings. The third stage flotation recovered an additional 7 pct of the coal, but at a low grade of 26.3 pct ash. Combined with the two previous stages, the three stage process recovered 93 pct of the coal in a concentrate containing 8.1 pct ash. Relative to the coal present in the second stage tailings, the third stage recovered 50 pct of the coal that was still present in the stage tailings. The fourth stage recovered more ash than coal and served no real purpose.

TABLE 10 Rapid flotation results from a typical 4 stage coal flotation test

	Flotation concentrate			Cumulative	
Product	Pct ash	Pct coal	Recovery, pct	Pct coal	Recovery, pct
1st stage	6.3	93.7	63	93.7	63
2nd stage	6.1	93.9	23	93.8	86
3rd stage	26.3	73.7	7	91.9	93
4th stage	39.9	60.1	2	91.9	95
Tailings	82.0	18.0	5	NA	NA
Composite	24.6	75.4	100	NA	NA

NA - Not applicable.

In order to compare the results of these various multi-stage tests, the grade of the combined coal concentrate was determined at the point where 90 pct of the coal was recovered. In this way each of the tests would be compared at an acceptable 90 pct recovery level. The residence time for all of the stages needed to obtain 90 pct recovery would be used to calculate the flotation rate (K_{90}). The results from the experiments holding the froth separator surface area to flow rate ratio constant are shown in table 11 and figure 7. The best grade and fastest flotation were obtained using the 49 W mixing intensity and the 1.5 mL/g air to ore ratio in the bubble-injected hydrocyclone bubble particle attachment unit.

TABLE 11 Concentrate grade, selectivity index (S.I.) and flotation rate at 90 pct recovery for various air/ore and mixing intensities holding the froth separator surface area to flow rate ratio constant†

Air/ore, mL/g	Total flow rate, L/min^{-1}	Mixing intensity power, W	Concentrate grade, pct coal	K_{90} flotation rate, min^{-1}
1.0	5.1	4.6	89.9	3.81
1.0	10.0	49.0	90.3	3.16
1.0	21.3	242.2	87.4	1.53
1.5	5.1	4.3	79.6	3.00
1.5	10.0	49.1	91.9	4.13
1.5	21.3	241.1	87.1	1.13

†Surface area to flow rate ratio = 31 cm^2·min/L

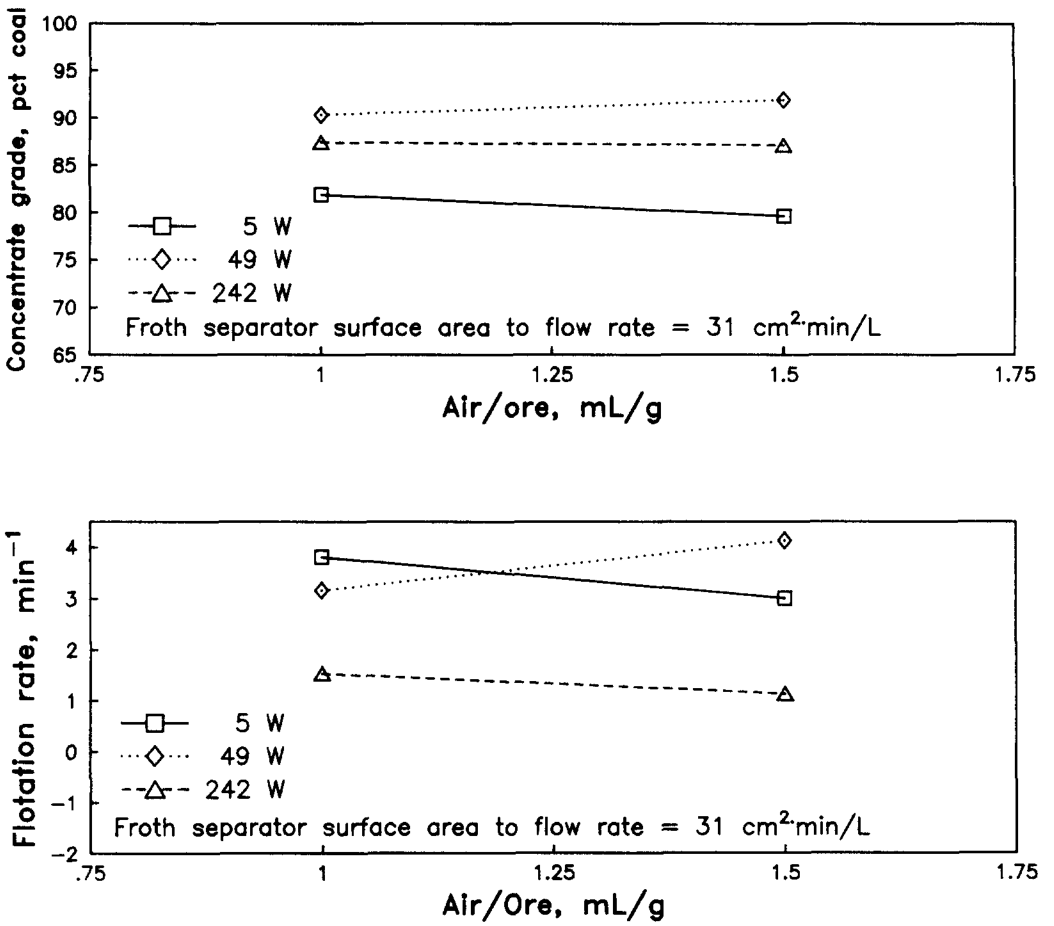


Fig.7 Coal concentrate grade and flotation rate at 90 pct recovery for various air/ore and mixing intensities holding the froth separator surface area to flow rate ratio constant.

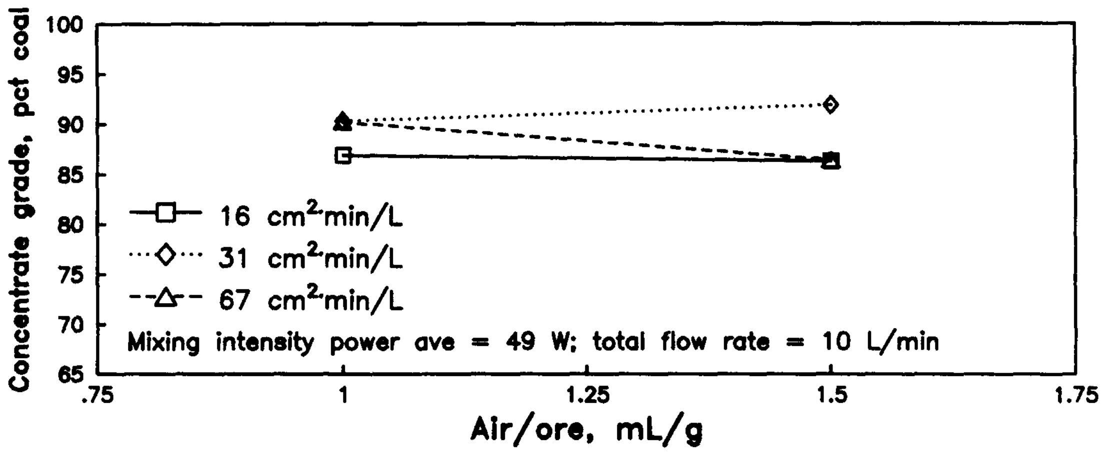
Holding the mixing intensity in the bubble-injected hydrocyclone bubble-particle attachment unit constant at 49 W, tests were also conducted with each of the three froth separator sizes. Again, the ratio of froth separator surface area to flow rate was directly proportional to the effective bubble residence time. The results are shown in table 12 and figure 8. The best flotation rate was obtained at the 31 cm^2·min/L surface area to flow rate ratio. At the shorter effective bubble residence time, some of the bubbles did not have enough time to leave the pulp and were swept down with the flotation tailings. This required additional flotation stages to obtain 90 pct coal recovery. At the longer effective bubble residence time, the bubbles were on the surface of the separator too long. Some of the bubbles coalesced and burst, releasing the attached coal which was swept down with the flotation tailings. Again, this required additional flotation stages to obtain 90 pct coal recovery.

At the best operating conditions of 1.5 mL/g air to ore ratio, 49 W mixing intensity, and 31 cm^2·min/L ratio of froth separator surface area to flow rate, 93 pct of the coal was recovered in a concentrate containing only 8.1 pct ash. These results are compared with the conventional laboratory flotation test in table 13. Both flotation systems were able to recover over 90 pct of the coal, but the rapid flotation system produced a better grade 18 times faster. This difference in the speed of flotation is graphically shown in figure 9. In approximately 40 s the rapid flotation system recovered over 90 pct of the coal. Whereas, conventional flotation required 10 min to obtain 90 pct recovery of the coal.

TABLE 12 Concentrate grade and flotation rate at 90 pct recovery for various air/ore and surface area to flow rate ratios holding the mixing intensity constant†

Air/ore, mL/g	Surface area to flow rate, $cm^2 \cdot L/min$	Concentrate grade, pct coal	Flotation rate, min^{-1}
1.0	16	86.9	2.51
1.0	31	90.3	3.16
1.0	67	90.2	0.72
1.5	16	86.3	2.76
1.5	31	91.9	4.13
1.5	67	86.4	0.58

†Mixing intensity power = 49 W, flow rate 10 L/min.



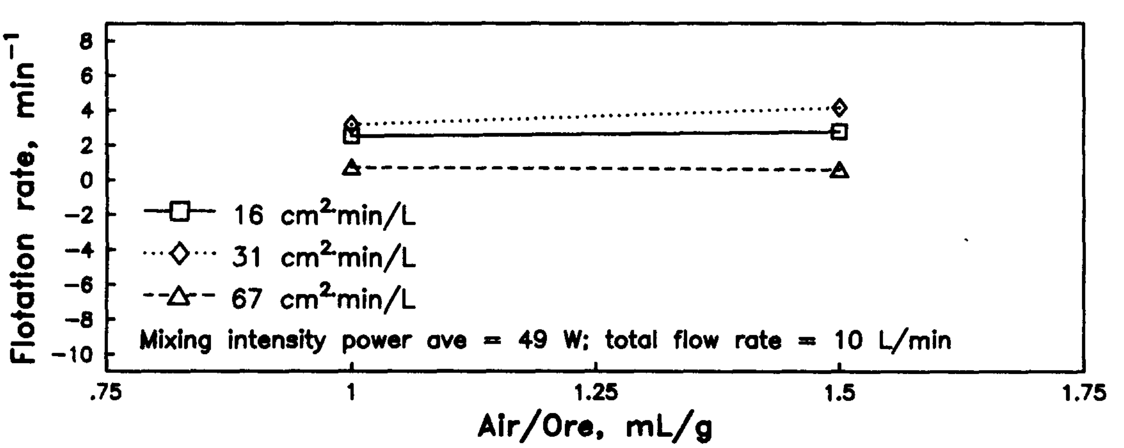


Fig.8 Coal concentrate grade and flotation rate at 90 pct recovery for various air/ore and surface area to flow rate ratios holding the mixing intensity constant.

TABLE 13 Comparison of the rapid flotation system and conventional laboratory flotation for coal flotation

	Flotation system	
	Rapid	Conventional
Coal recovery.............pct	93.4	90.0
Concentrate grade....pct coal	91.9	90.5
Residence time............min	0.66	10.0
Flotation rate..........min^{-1}	4.13	0.23
Capacity..............$t/hr/m^3$	4.38	0.32

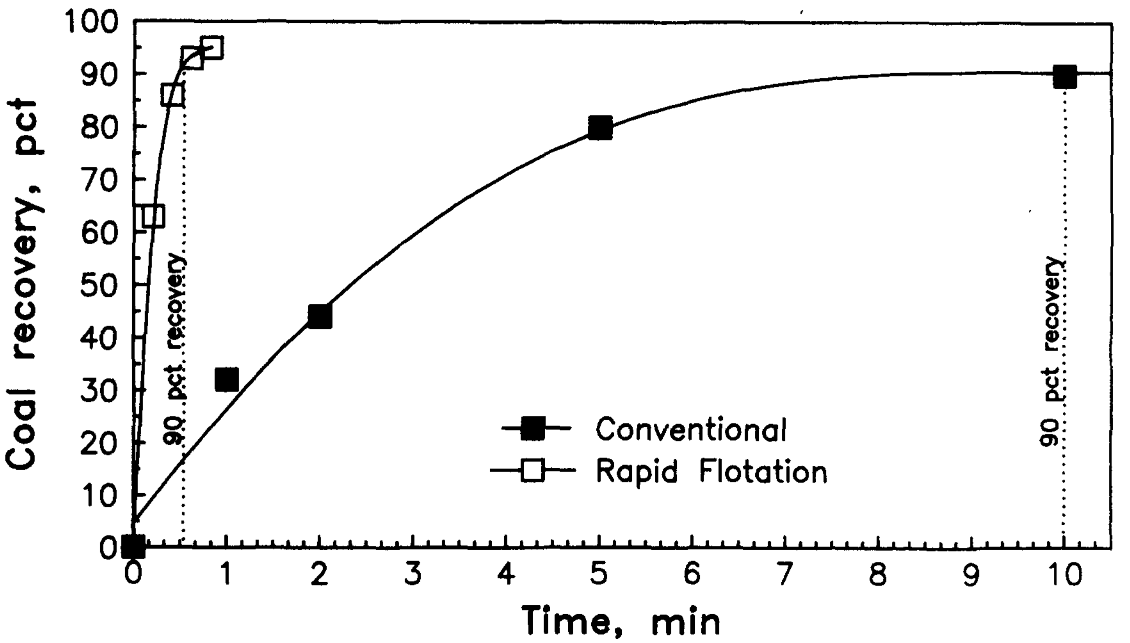


Fig.9 Comparison of coal flotation kinetics with the rapid flotation system and the conventional flotation cell.

CONCLUSIONS

The Bureau's rapid flotation system illustrates how separating the bubble-particle attachment and bubble-pulp separation can improve the overall flotation kinetics. Each unit can be optimized individually to obtain the best performance of its function. There was an optimum mixing intensity and slurry residence time in the bubble-injected hydrocyclone bubble-particle attachment unit for the best performance. There was also an optimum froth separator surface area to flow rate ratio for the froth separator (effective bubble residence time). Matching these two optimized units substantially improved the flotation kinetics and improved flotation kinetics allow for much smaller flotation circuits to replace the larger conventional flotation circuits of the same capacity. The rapid flotation system could be scaled-up by using the mixing intensity of the bubble-injected hydrocyclone bubble-particle attachment unit, air to ore ratio, and the froth separator surface area to flow rate ratio parameters. Larger scale units should be designed to match those parameters.

The rapid flotation system was shown to be effective on an amine tailings sample. The best quartz grade at the fastest flotation rate was obtained with a 1.5 mL/g air to ore ratio, a 15 W mixing intensity in the bubble-injected hydrocyclone bubble-particle attachment unit, and a froth separator with 31 $cm^2 \cdot min/L$ ratio of surface area to flow rate. About 59 pct of the quartz was rapidly floated at a 6.62 min^{-1} flotation rate. This was nine times faster than the conventional flotation system. The unfloated product contained 15 pct P_2O_5 and recovered 82 pct of the phosphate. This product was suitable for recirculating to the phosphate flotation circuit.

The rapid flotation system also demonstrated its effectiveness on coal. The best operating conditions of 1.5 mL/g air to ore ratio, 49 W mixing intensity, and 31 $cm^2 \cdot min/L$ ratio of froth separator surface area to flow rate, recovered 93 pct of the coal in a concentrate containing only 8.1 pct ash. The rapid flotation system was 18 times faster than the conventional coal flotation system. The flotation system had an optimum mixing intensity in the bubble-injected hydrocyclone bubble-particle attachment unit. In addition, there was an optimum froth separator surface area to flow rate ratio for the froth separator. Each of the two unit operations could be independently optimized to provide a rapid flotation process with improved flotation kinetics.

REFERENCES

1. Comment by N. Arbiter, *Challenges in Mineral Processing Conference*, Berkeley, California, (Dec. 1998).

2. Azbel, K., & Liapis, A. I., Motion of Solid Particles in Liquid Medium, *Handbook of Fluids in Motion*, 895-926 (1983).

3. Bischofberger, C. & Schubert, H., On the Hydrodynamics of Flotation Machines, *Inter. J. of Mineral Processing*, **5**, 131-142 (1978).

4. Kelley, E. C., & Spottiswood, D. J., *Introduction to Mineral Processing*, Wiley and Sons, (1982).

5. Spears, D. R., & Jordan, C. E., A Novel Flotation System Based on the Bubble-injected Hydrocyclone, *Soc. of Mining Engineering. AIME preprint 91-142*, 8 pp (1991).

6. Stanley, D. & Jordan, C., Bubble-Injected Hydrocyclone Flotation Cell, 1989, *U.S. Patent No. 4,971,685*, (Nov. 20, 1990).