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REVIEW OF AGGLOMERATION PRACTICE AND FUNDAMENTALS IN HEAP LEACHING

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This article reviews agglomeration practices for precious metal and copper heap leaching. Both industries prefer drum to conveyor agglomeration, particularly for clayey ore or ore having a high fines content. Precious metal heap leaching operations opt for cement in a dosage from 2.5 to 10 kg cement/t of ore (5–20 lb/ton) added to a cyanide solution. Copper ores are agglomerated with water and up to 40 kg sulfuric acid/t of ore (80 lb/ton) without binder. The agglomerate physical characteristics, with the exception of their strength, can be measured precisely and automatically. The impact of agglomeration on the in situ physical characteristics of the heap, other than the observable ponding and slumping, is not understood well. The most substantial benefits of agglomeration include up to 90% metal recovery from poorly permeable ores, shorter leach cycles, extra metal recovery from already-leached tailings, and better environmental heap closure.

Keywords: agglomeration, gold, copper, heap leaching, practice, fundamentals

INTRODUCTION AND OUTLINE

Agglomeration is the consolidation of solid particles into larger shapes by means of agitation alone (i.e., without application of mechanical

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pressure in molds, between rolls or through dies). The work of T. C. Scrutton in 1905 is one of the earliest references to agglomeration. Scrutton agglomerated ore by rolling it down a chute inclined at 60° and stacked the agglomerates in a vat leach (very large vessel filled with ore and completely saturated with leaching solution). The initial work on iron ore pelletization began a few years later, in 1911, and quickly expanded to include other materials, including manganese, fluorspars, and phosphate, for better furnace operation. Advances in agglomeration for heap leaching occurred 25 years later, in 1937 precisely, with Shepard's work on the agglomeration of gold tailings with lime and calcium carbonate.

There are numerous publications, proceedings, and conferences about the matured practice, fundamentals, and benefits of agglomeration in the ferrous mining industry. There are relatively few, very recent references on agglomeration for heap leaching authored by a couple of pioneers such as Gene McClelland and Paul Chamberlin. This review article summarizes the fundamental and practical advances of agglomeration for heap leaching. This article discusses, in order 1) the preliminary ore characterization that determines the agglomeration requirements, 2) the most common agglomeration equipment employed, 3) the nature, dosage, and chemistry of the solution and binders added, 4) the methods to evaluate the agglomerate quality, 5) the impact of stacking and irrigation methods, and 6) the benefits of agglomeration. The review of more than 100 publications produced two compilation tables (Tables 1 and 2) about the past and current practice of agglomeration in copper and precious metal heap leaching sectors. The two tables contain data about 1) the type and size of agglomeration equipment employed, 2) the nature and amount of solution added, 3) the nature and dosage of binders added, 4) the curing, stacking, and irrigation practices, and 5) the measured impact of agglomeration on copper, gold, and silver recovery. These tables are referred to throughout this article. The article utilizes SI units, with imperial (e.g., lb/ton for dosage) or American (e.g., GPM/ft^2 for irrigation rate) units listed in brackets. In this article, the unit "kg/t of substance" means "kg of substance per tonne of ore," unless otherwise indicated.

The most significant advances in agglomeration for heap leaching occurred in the late 1970s to early 1980s at the U.S. Bureau of Mines in Reno, Nevada (Potter 1983). This organization had previously developed in the 1960s cyanide heap leaching processes for recovering

Table 1. Agglomeration practice in the copper heap leaching industry

Operation			Agglomeration Conditions				Leaching					
Operation	Production (tpd)	Initial particle size (mm)	Agglomeration equipment	Sulfuric acid (kg/t)	Water (W)/ raffinate (R) (kg/t)	Final moisture (%)	Curing (d)	Stacking height (m)	Stacking equipment	Irrigation ($L/m^2/h$)	Leaching time (d)	Copper recovery (%)[1]
Mine-for-leach, Morenci, Phelps Dodge (O'Brien et al. 2003)	85,000	P_{80} 12.5	2 drums, 14 ft wide, 40 ft long	2.1–3.1	W/R	7	0	6.85	Mobile and radial stacker	6.1 (18″ × 18″ drip emitter grid)		
Mantoverde (Trincado et al. 2003)	23,000	P_{100} 15	Drum					6.8		10–12		
Nifty Copper Operation (Readett et al. 2003)	68	P_{80} 17		15	W		5–10	9	Radial stacker	6–10 (31″ center)		
(Holle 1996)	8,400	P_{80} 12	Drum, 8.8 ft wide, 26 ft long, 6°	6–10	7–80 W	7–8						
Quebrada Blanca (Canello and Schnell 1995; Schnell 1997)	13,200	P_{100} 9 P_{80} 6	2 drums, 9.7 ft wide, 29 ft long, 5°	5–7	65 W	9–10	5	6–6.5		6–9 on/off (0.4 m × 0.8 m)	>300 d	80%
El Abra (Jenkins and Canello 1997)	8,250	P_{100} 19	3 drums, 14 ft wide, 43 ft long, 6°	16–20	80–100 W/R		2	6–10	Mobile stacker	15	45	85%

(*Continued*)

Table 1. Continued

Operation			Agglomeration Conditions				Leaching					
Operation	Production (tpd)	Initial particle size (mm)	Agglomeration equipment	Sulfuric acid (kg/t)	Water (W)/ raffinate (R) (kg/t)	Final moisture (%)	Curing (d)	Stacking height (m)	Stacking equipment	Irrigation ($L/m^2/h$)	Leaching time (d)	Copper recovery (%)[1]
Zaldívar, Placer Dome Inc. (Bouffard 2004)	50,000	P_{80} 12	Conveyor	12	W	8	15–30	9	Mobile stacker	8 (drip emitter)	300	80%
Doña Inés de Collahuasi S.A.		P_{100} 9		28–35	80–100 W		1	5		10–15	60	68–85%
Lince		P_{100} 9		35	50 W			4		24	30	80%
Cerro Colorado		P_{100} 12.5		7	60 W		2	6		12	210	80%
Lo Aguirre		P_{93} 6		15–30	60–70 W		1	3–4		6–12	180–360	63–68%
El Salvador, owned by Codelco (Chile) (Pino 1995)		P_{97} 12.5		11–20	55–60 W		1–2	5		10	240–360	75%

Ivan Mine	P_{100} 12.5	20–50	150 W			20–25	30 (oxide)	84% (oxide)
Sociedad Punta del Cobre S.A. Planta Biocobre	P_{100} 12.5	25	65 W	1		20	30–120	80% (oxide)
Radomiro Tomic, Codelco (Chile)	P_{100} 51	5–8	40 W		8	10	45	78% (oxide)
Empresa Mineral Mantos Blancos S.A. Manto Verde	P_{100} 10	26	90 W	1	5	15	50	83% (oxide)
Tesoro Project, Anaconda Chile S.A.	P_{100} 25	20	100 W	1	3–5	15	30–60	76% (oxide)
La Cascada, owned by Sociedad Minera La Cascada Ltd.	P_{100} 9.5	32	0 W	<1	1.9		7–8	75% (oxide)
Minera Disputada de Las Condes,	P_{100} 8	60	W	1	3	36	14	65% (oxide)
Compania Minera de Tocopilla	P_{90} 9.5	45	130 W	1	1.8		20	75% (oxide)

[1]The designation (*oxide*) following the percent copper recovery indicates an operation processing copper oxide ores only.

Table 2. Agglomeration practice in the precious metal heap leaching industry

Operation			Agglomeration equipment				Agglomeration conditions				Leaching				
Operation	Production (tpd)	Maximum particle size (mm)	Stockpile	Belt	Reverse belt	Drum (# of drums, diameter in m, length in m, angle in °, speed in rpm)	Cement (kg/t)	Other (kg/t)	Solution	Moisture after agglomeration (%)	Curing (h)	Stacking height (m)	Irrigation ($L/m^2/h$)	Leaching time (d)	Gold recovery (%, unless noted)
Arizona gold heap leaching	240	9.5				1, 0.9, 2.1, 5, 17	5		1 kg NaCN/t solution	10	48	2.1	81	5	90%
Colorado gold vat leaching	1,500	12.5				1, 2.6, 9.8, 8, 10	1.5	Fly ash −3.5	5 kg NaCN/t solution	13	8	vat		3	90%
Arizona silver heap leaching	180–200	12.5			×			Lime −3.5	Water	10–12	72	32	18.3	7	90% (before 37% in 90 d)
N. Nevada gold heap leaching	2,500	15.3	×				3.5–5		Water	9–13 (zones in the heap retain 30%)	48–72	3.7	11.0	20–30 (before 50 d)	60% gain
E. Nevada gold heap leaching	5,160–10,800	19				1, 2.7, 6.4, N/A, N/A	2–5		NaCN solution	8		4.6	9.8	20–80	70–90%
Central Nevada tailings operation (McClelland, 1986)		P_{65} 200#				1, 2.6, 6.7, 4,10.5	5	Lime –25	Water	12–14		4.9	7.3	24	76%
Southeastern California silver operation (McClelland, 1986a)	24,000	P_{90} 200#				2, 3.0, 9.1, N/A, N/A	17.5		Water	12–15		4.0	24.4	25 leach 40 rinse	73%

Alligator Ridge (DeMull and Womakc, 1984; Strachan and van Zyl 1987)		19		×		Lime −1.5–5	0.6 kg NaCN/t solution	5–11		3.7	14.6	30–40	70%
Cerro Rico by Compania Minera del Sur S.A., Bolivia	400	P_{50} 100#		N/A, 2.0, 7.0, N/A, N/A	10		1 kg NaCN/t solution	15–17		6	17.8		
Anaconda's Darwin Silver Mill (Milligan and Engelhardt 1983)	3,960	P_{87} 100#		×	17.5	Lime −5		15.7	96	1.4		35	25–72%
Kennecott Barneys Canyon Mining Co (LeHoux 1997)													
Haile Gold Mines Inc. (Phifer 1988)	1,600	49		N/A, 1.8, 7.9, 7, 9	4–7.5		NaCN solution	12	48	4.3		30–45	90%
Masbate, Philippines (Pizzaro et al. 1986)	1,200	12.5		N/A, 2.7, 8.8, N/A, N/A		Lime −9	NaCN solution	9–12	24	3.4	17.1	20	58%
Houston International Minerals Borealis Project (Outzen 1983)	1,400	37	×		5		0.25 kg NaCN/t solution			4.6	14.6		82–84%
Gooseberry Mine (Butwell 1990)		P_{85} 200#		N/A, 3.0, 9.8, 12.5, N/A	7.5–11			17.6		9.1	7.3	60–70	85% gold 75% silver
Brewer Gold (Pautler et al. 1990)				×	4	Nalco 9760 −0.12	NaCN solution	10–12	72		12.2	90 leach 15 rinse	7% gain

(*Continued*)

Table 2. Continued

Operation			Agglomeration equipment				Agglomeration conditions				Leaching				
Operation	Production (tpd)	Maximum particle size (mm)	Stockpile	Belt	Reverse belt	Drum (# of drums, diameter in m, length in m, angle in °, speed in rpm)	Cement (kg/t)	Other (kg/t)	Solution	Moisture after agglomeration (%)	Curing (h)	Stacking height (m)	Irrigation ($L/m^2/h$)	Leaching time (d)	Gold recovery (%, unless noted)
Cripple Creek, CO (Gomes 1983)	1,500					N/A, 2.4, 9.8, N/A, N/A	5		NaCN solution	13	8	2.7 (vat)			
Tombstone, AZ (Gomes 1983)	4,800				×							3.2		7	90% Ag
Maggie Creek, NV (Gomes 1983)	2,000		×				3.5	Fly ash – 1.5		13					

precious metals from low-grade ores. Cyanide heap leaching is the preferred process to treat low-grade gold and silver ores that cannot be economically ground and cyanide-leached in stirred tanks. Heap leaching consists of stacking run-of-mine or crushed ore to form a pile several feet high, then irrigating the crest (top) of the heap with a solution that percolates, by gravity, through the pile to solubilize the metal value and, finally, separating the soluble metals from the pregnant solution (called *pregnant* because of its high metal value, in contrast to the *barren* solution—low in metal value—applied on top of the heap) discharging at the bottom (toe) of the pile. In 2001, there were approximately 80 to 120 precious metal heap leach operations worldwide, of which 34 were based in the United States (Kappes 2002). In 2001, 22 of the 34 American precious metal heap leach operations were located in Nevada.

Some commercial heaps commissioned prior to the U.S. Bureau of Mines agglomeration studies were suffering from low solution permeability, which resulted in lower gold and silver recovery and a longer leaching cycle. The wide range of particle sizes in the heap, the segregation of these sizes during stacking and irrigation, and the ore mineralogy were the ultimate culprits of low permeabilities.

The U.S. Bureau of Mines determined the optimum practical requirements (moisture level, binder type, binder dosage, curing time, agglomeration equipment, and residence time) for agglomeration of precious metal ores and tailings. Original methods (e.g., drainage rate from a flooded column), albeit atypical of heap leaching environments, were designed to compare quantitatively various agglomeration conditions. A decade later, in 1979, the work culminated in the first pilot-scale heap leach test on crushed and agglomerated ore in Eastern Nevada. The first commercial agglomeration heap leach utilizing agglomerates began operation in 1980. According to Gomes (1983), there were 36 commercial operations in the Western United States in 1983 that agglomerated ore.

The 2001 statistics compiled by Kappes revealed that 12 gold operations heap-leached run-of-mine ore, 7 heap-leached run-of-mine and crushed ore, and 24 heap-leached crushed ore. Thirty-two operations reported an average heap height of 8.9 m, an average irrigation time of 70 d during the leach, and an average solution application of 1.3 t/t of ore. Thirty-three percent of 24 operations that heap leached crushed ore only did not agglomerate; 66% agglomerated crushed ore prior to stacking. The survey did not indicate the proportion of run-of-mine and run-of-mine/crushed operations that agglomerated.

Advances in agglomeration in precious metal heap leaching paralleled those in copper heap leaching. Chilean copper production boomed from 20,000 tpa in the 1980s to 1,000,000 tpa in the 1990s (Scheffel 2002). The phenomenal expansion of the heap leach, solvent extraction, and electrowinning procress is attributed to the development of the "thin layer leaching" concept. Sociedad Minera Pudahuel commercialized this concept for copper oxide ores in 1980 and then, for copper sulfide ores *as* "bacterial thin layer leaching," in 1985. The ore, crushed to less than 12–16 mm (1/2"), is agglomerated typically with concentrated sulfuric acid, stacked in heaps 6–8 m (20–26 ft) tall, and cured for a couple of days prior to irrigation with raffinate solution. The majority of copper operations agglomerate in one form or another by mixing the ore with concentrated sulfuric acid and water (Table 1).

Pelletization, granulation, and agglomeration are common terms describing a size enlargement process. The term agglomeration is most often employed for ores containing both fine and coarse particles whereby fine particles coat larger particles. Lipiec and Bautista (1998) define these agglomerates as *rim agglomerates*. The Geobiotics agglomerates (Kohr, 1998) are another form of *rim* agglomerates whereby a slurry of sulfide concentrate is sprayed onto inert, coarse particles. The agglomeration of ore containing only fine particles of less than typically 74 μm (200#), such as tailings or iron ore, is referred to as pelletization. As shown in Figure 1, the pellet structure is more homogeneous than the agglomerate structure.

The size enlargement chapter in Perry's *Chemical Engineering Handbook* offers a comprehensive outlook on kinetics, equipment, and modeling of agglomeration processes. Certain topics directly applicable to agglomeration processes for heap leaching were extracted and are summarized below.

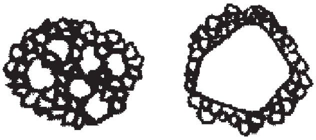

Figure 1. Simplistic representation of a pellet (left) and an agglomerate (right).

The binding process occurs in typically four consecutive or parallel stages:

- Wetting—coating of particles with a liquid film, of great importance for tailings because of the comparable size between the water droplet and the solid material
- Growth—characterized by three phases: nucleation, followed by coalescence of nuclei to form an agglomerate and layering of fines onto the nuclei or agglomerate
- Consolidation—bed agitation intensity and compaction pressure applied to an agglomerate that reduce its porosity
- Breakage—classified as shatter, fragmentation, wear, and abrasion of agglomerates

A population balance model can describe mathematically the rates of nucleation, layering, coalescence, and breakage. Such equations consider the changes in the number of agglomerates and in their mass. For instance, the layering process does not change the number of agglomerates, but increases their mass. The coalescence process (two agglomerates forming one) has the opposite effect. Breakage can help produce more uniformly sized agglomerates by splitting large, poorly consolidated agglomerates. To this author's knowledge, there were no available references on the modeling of crushed ore, except for some glimpses into the distribution of particles making up agglomerates of various sizes (Bouffard 2003).

There are four types of bonding mechanisms:

- Solid bridges—created by the crystallization of dissolved substances, the hardening of bonding agents such as glue and resins, or the chemical reactions between wetting fluid, binder, and ore. Cement curing for precious metal ore agglomeration is an example of the latter type. Glue and resins are expensive and may interfere with the heap leach chemistry.
- Mobile liquid binding—created by the surface tension and capillary water suction that exist in three forms:
    - Pendular state corresponding to discrete lens-shaped rings at the points of contact of particles

  - Funicular state corresponding to a network of liquid interspersed with air and particles
  - Capillary state corresponding to the complete saturation of pores between particles

- Intermolecular and electrostatic forces—created by the short-range forces that attract very fine particles ($<1\,\mu m$) under agitation.
- Mechanical interlocking—created among fibrous particles under agitation or compression.

Of the four types of bonding mechanisms, solid and liquid bridges are the most common in the agglomeration of crushed ore and tailings. Solid bridges are likely to better survive wetting in the more saturated heap than liquid binding forces.

ORE CHARACTERIZATION

Several publications have stressed the essential need to characterize the physical, chemical, and mineralogical properties of the ore to be heap leached. With regard to the physical characteristics, the methods include: particle size distribution; silt vs. clay content; Atterberg limits of plasticity and solidity; weathering and swelling characteristics; triaxial strength and internal friction angle; and permeability.

The particle size distribution (or ore gradation) obtained by wet screening constitutes the starting point for further testing. Most consultants (Heinen et al. 1979; McClelland 1986a; Garcia and Jorgensen 1997; Kinard and Schweizer 1987) concur that the proportion of fines smaller than $50–75\,\mu m$ (200–270#) determines the need for agglomeration. Garcia and Jorgensen (1997) recommended agglomerating ores containing more than 5% of $-74\,\mu m$ (200#) fines. Garcia and Jorgensen (1997) recommended agglomerating with binder if $-74\,\mu m$ (200#) fines account for more than 10–15%.

The Atterberg limits are classified as liquid and plastic limits. The Atterberg liquid limit defines the moisture content at which the material changes from a liquid to a plastic state. The Atterberg plastic limit defines the moisture content at which the material changes from a plastic to semisolid state. If the liquid limit is greater than 20 and the plastic limit is greater than 10, the ore appears to contain high clays. The liquid and plastic limit can be measured using the ASTM D4318 standard.

The triaxial strength and internal friction angle, measured using the ASTM D4767 standard, helps to determine whether the heap will have sufficient static and seismic strength. A friction angle of 30–37% indicates sufficient stability. Lower friction angles could be evidence of poor stability.

ASTM standards D4546 and D5890 measure the swelling index of soils and clay mineral component of geosynthetic clay liners. A common procedure for measuring the swelling index consists of placing 0.1 kg of material in a cylinder equipped with a porous plate and submerged in water. The height difference after 24 h of soaking is a direct measure of the swelling.

The greater the permeability of a bed of rock or soil, the greater is the ability of the bed to carry larger flows of solution without flooding and to drain rapidly. The solution application rates in heap leaching are so small that the fluid can be assumed to obey Darcy's law:

$$\mathbf{u} = -K\nabla P = -\frac{k'\rho g}{\mu}\nabla P \tag{1}$$

where
$\mathbf{u}$: vector flow velocity (m/s)
P: hydrostatic head (m)
g: gravitational acceleration (9.81 m/s^2)
K: saturated permeability of the bed (m/s)
k': intrinsic permeability (m^2)
ρ: fluid density (kg/m^3)
μ: liquid viscosity (kg/m/s)

There are numerous ASTM standards (D5084, D2434, D5856, D5093, and D6391) to measure the permeability of a bed of rocks, i.e., when all pores in the bed are filled with solution. The constant and falling head permeameters are popular methods.

On a scale from excellent to worse permeability, clean gravel has the largest permeability, ranging typically between 10 to 100 cm/s. Gravel drains very easily. Clean sand ranks second to gravel, with a typical permeability of 1 cm/s. Very fine sand has a much lower permeability of 10^{-5} cm/s and drains poorly. Organic and inorganic silts, mixture of sands, silts, and clays have a very low permeability of 10^{-6} cm/s. Lastly, impervious soils made up of clays have the worst permeability, ranging from 10^{-7} to 10^{-9} cm/s. These soils drain very slowly, if at all, due to the swelling of the clays.

This paragraph presents three examples of permeability measurements on unleached and leached ore. The permeability of the unleached agglomerated high-clayey ore decreased rapidly from 0.2 cm/s to 10^{-4} cm/s with an applied normal stress of 25 psi (Garcia and Jorgensen, 1997). This load corresponds to a heap height of 8 m (26 ft). With increasing loads of up to 200 psi, the permeability was reduced further to only 10^{-7} cm/s. Uhrie et al. (2003) found that the permeability decreased with increasing clay content. The permeability of an ore containing low clay remained above 10^{-1} cm/s under loads corresponding to an equivalent heap height of 183 m (600 ft). In comparison to the guidelines referred to in the previous paragraphs, the high-clayey ore tested had a very high permeability of 0.003 cm/s under load. Kinard and Schweizer (1987) measured the permeability of leached clayey agglomerates, originally 0.3–1.0 cm in diameter, after cyanide heap leaching. The agglomerates were recovered from the 9-m (29.5-ft) tall heap with a backhoe, after measuring the heap density at different depths. The permeability ranged from 10^{-4} to 4×10^{-7} cm/s and was inversely proportional to the bulk density. The bulk density was surprisingly low, ranging from 1.19 to 1.43 t/m^3 (74–89 lb/ft^3), but not correlated with heap depth.

A decision to agglomerate should be based on the proper evaluation of the physical, chemical, and mineralogical characteristics of the ore. Agglomeration alone may suffice to increase solution permeability. In other instances, agglomeration of ores of high fines content, high clay content, or brittle in nature may still not provide adequate solution and air permeability. A viable process consists of screening out (desliming) fines and heap leaching the remaining coarse particles. Trent Parker and Harmel Dawson of Dawson Metallurgical Laboratories were awarded, in November 1979, the first patent related to screening (U.S. patent 4,173,519). It is worth noting, though, that the remaining coarse particles may still decrepitate in contact with the leaching solution. This is more problematic for copper or zinc sulfide ores, which contain a larger percentage of metal value, compared to gold ores. Phifer (1988) noted a significant improvement in the gold recovery from 30% to 80% after screening out –74 μm (200#) fines, which accounted for 30–40% of a run-of-mine ore. A 3-m (10-ft) tall heap stacked with nonscreened run-of-mine ore had previously slumped to 2.2 m (7 ft). Fines had migrated due to heavy rainfall. A hybrid tank/heap flowsheet should increase the overall recovery, because of the larger recovery achievable from fines leached in tanks and the larger recovery obtained from coarse particles in the heap. In addition, it is possible to envision crushing the remaining coarse particles, without generating too many more fines, to further increase the recovery. An alternative to screening consists of screening the ore into fine and coarse fractions and regulating the rate of each fraction fed to the agglomerator.

EQUIPMENT

Estimates proposed by Rose et al. (1990) and Kappes (2002) indicate that agglomeration and stacking account for 6–10% of the total capital costs and 10–21% of the total operating costs (Table 3). The cost of cement alone is about $US 1.00/t of the $1.15/t, i.e., 90% of the agglomeration and stacking operating costs.

According to Kappes' (2002) recent survey of heap leach design and practice in the precious metal industry, of the 24 of the 43 responding mining companies that crushed ore before heap leaching, 8 did not agglomerate, 5, including Barney's Canyon and La Quinua operation at Yanacocha, turned to belt agglomeration, and 11 utilized rotating drums.

This section discusses belt conveyor and drum agglomeration—the two principal industrial-scale agglomeration equipment employed for heap leaching. Some references to rotating disc agglomerator and pug mills are included.

Belt Agglomeration

Conveyor belts are well suited to agglomerate ore containing typically less than 15% of –104 μm (150#) fines. Belt conveyors can be operated in three ways.

When all belt conveyors are inclined at about the same angle (about 15°) and are moving in the same direction, agglomeration occurs when particles touch each other at the transfer point between belts or when they bounce on the belt upon landing (Figure 2). The belt typically moves at a rate of 1.25–1.50 m/s (250–300 ft/min) (Chamberlin 1986). Dispersion bars hanging at the discharge of a belt improves the mixing of the

Table 3. Influence of the scale of the heap leaching operation on the capital and operating costs of the agglomeration/stacking unit relative to the total cost (all figures in US dollars)

	Production	400 tpd (Rose et al. 1990)	1000 tpd (Rose et al. 1990)	3000 tpd (Kappes, 2002)	15,000 tpd (Kappes, 2002)	30,000 tpd (Kappes, 2002)
Capital	Agglomeration/stacking	\$460,000	N/A	\$1,000,000	\$3,500,000	N/A
	Total	\$4,600,000	N/A	\$14,000,000	\$53,600,000	N/A
	Percentage	10%	N/A	7%	6.5%	N/A
Operating	Agglomeration/stacking	N/A	\$1.15/t	\$1.20/t	\$1.10/t	\$1.10/t
	Total	N/A	\$7.20/t	\$11.50/t	\$8.30/t	\$5.20/t
	Percentage	N/A	16%	10%	13%	21%

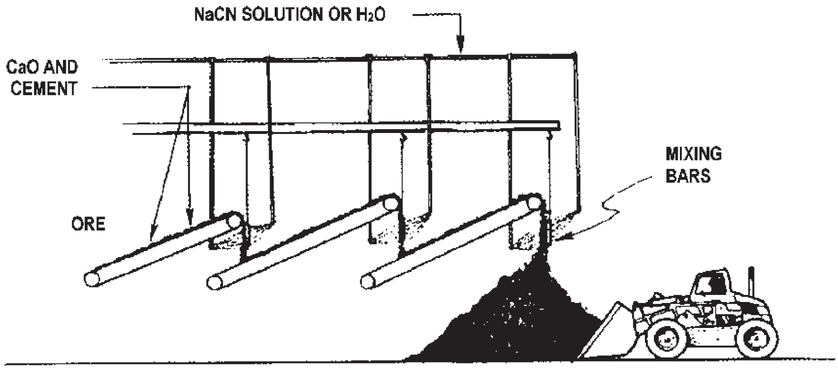

Figure 2. Belt conveyor agglomeration. Reproduced from Chamberlin (1986). Reproduced with permission from *Mining Engineering Magazine*.

ore in its free fall of 1.2–1.8 m (4–6 ft). The number of required transfer points increases with increasing fines content. The number of transfer points also depends on the nature of the solution addition. LeHoux (1997) found that the addition of a slurry rather than water required 3–4 times more drop points.

The solution can be sprayed at the transfer points or along the belts. Too little solution results in excessive dusting at the transfer points. Too much solution addition results in spillage at the transfer points, damage to the belt, more frequent shutdown for cleanup, and solution running down the belt, as well as compaction, of the agglomerates upon landing on the pile. Startup and shutdown may also lead to solution running onto an empty belt and washing off the pile. Because some ores cannot absorb the prescribed amount of moisture all at once at the transfer points, LeHoux (1997) recommends staging the points of solution addition at the tail of the conveyor trains to avoid buildup of slimes underneath the equipment and solution running. In his experience, adding a polymer or a filtration aid reduced splashing and allowed for large solution additions.

When the ore falls from a low-angle conveyor moving relatively slowly onto a high-angle (35–55°) conveyor moving rapidly in the opposite direction of the low-angle conveyor (Figure 3), agglomeration occurs at the transfer point and primarily on the high-angle belt due to the opposite forces in action: the forward momentum of the belt attempting to move agglomerates to the top, counterbalanced by the gravity forcing the agglomerates to roll down to the bottom of the high-angle belt.

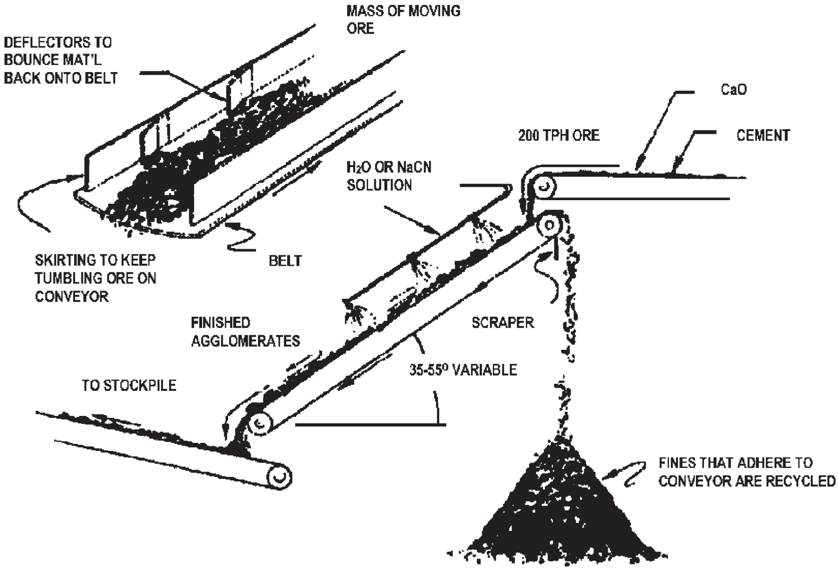

Figure 3. Reverse, high-angle belt conveyor agglomeration. Reproduced from Chamberlin (1986). Reproduced with permission from *Mining Engineering Magazine.*

Too high of a belt angle would cause the agglomerates to slide down the belt rather than roll. McClelland et al. (1985) rated the quality of agglomerates produced from tailings with a reverse belt as inferior to the quality obtained from a drum agglomerator, even though both equipment had the same agglomeration time of only 10–15 s.

When the ore falls from a low-angle belt onto a vibrating conveyor, particle agglomeration occurs at the transfer point and when agglomerates slightly bounce on the vibrating conveyor and hit each other.

Drum Agglomeration

Drum agglomeration consists of injecting ore into a cylindrical, inclined drum that rotates to impart rolling, cascading, and tumbling (Figure 4). Particles are set in motion by the balance between gravitational and centripetal forces. Commercial drums average 6.4 m (21 ft) in length and 2.1 (6.8 ft) in diameter. The drum will rarely exceed 15 m (50 ft) in length. Another drum is installed in parallel to handle larger throughput. The solution is pumped through nozzles or perforated pipes located

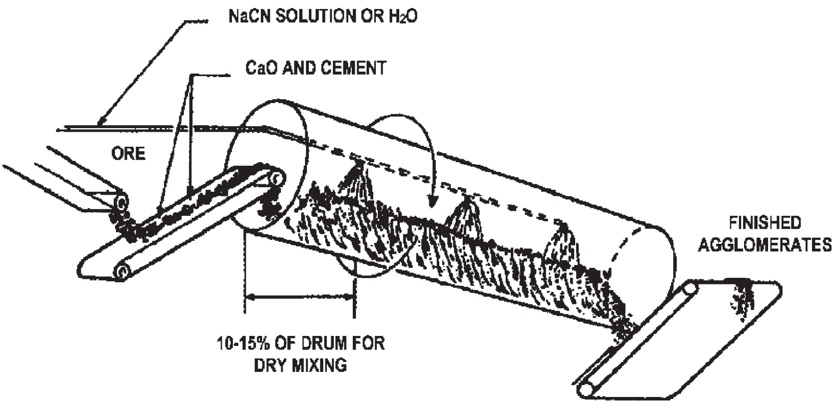

Figure 4. Drum agglomeration equipment. Reproduced from Chamberlin (1986). Reproduced with permission from *Mining Engineering Magazine.*

preferentially along the first 2/3 of the drum length. The drum may be rubber-lined to prevent corrosion and equipped with loose chains or rubber strips to prevent ore from sticking. Partially closing the front entrance of the drum with berms prevents ore spillage. There were no available references on the number and size of baffles.

Drum agglomeration is well suited for ores containing high clays or a large fines content. Chamberlin (1986) prefers drum agglomerator to belt conveyor when a binder must be added. In rare applications, a drum agglomerator may be equipped with screens on the discharge end to separate the oversize from the undersize. Recycle ratios between 2:1 and 5:1 are common in iron-ore pelletization and fertilizer granulation circuits. Recycling affects the moisture content, already susceptible to feed moisture variations, and the agglomerate size distribution.

There are three steps involved in the calculation of the drum throughput, starting with the calculation of the drum critical and normal rotation speed, then the drum residence time, and lastly the drum throughput.

The critical speed, i.e., the speed at which a single particle is held stationary in the drum due to centripetal forces, is given by

$$C = \sqrt{\frac{g \sin \theta}{2\pi^2 D}} \approx \frac{42.3}{\sqrt{D}} \tag{2}$$

where
C: critical speed (rpm)
D: drum diameter (m)
g: gravitational acceleration (m/min^2)
θ: angle of the drum from the vertical (80 and 90°)

The normal rotation speed, N, is a fraction, α, of the critical speed, C. The fraction, α, is equal to 30–50%. Slow rotation allows the agglomerate to roll rather than cascade.

An empirical equation, (3), derived from rotary dryer by the U.S. Bureau of Mines, calculates the residence time as a function of the drum length, the drum diameter, the rotation speed, the incline of the drum, and the angle of repose of agglomerates:

$$t = \frac{1.77\sqrt{\phi}L}{(90 - \theta)DN} \quad (3)$$

where
t: retention time (min)
ϕ: angle of repose of agglomerates (about 45°)
L: drum length (m)
$90-\theta$: drum incline ranging from 1 to 12.5° (typically 5–7°) from the horizontal
D: drum diameter (m)
N: normal rotation speed (rpm)

According to (3), longer drum, drum of larger diameter, drum inclined at a lower angle, and slow rotation increase the residence time. Assuming the material to move in a piston flow through the drum, the throughput can be calculated using:

$$Q = 1440\frac{\pi D^2 L}{4}\frac{\rho f}{t} \quad (4)$$

where
f: solid volume holdup (m^3 drum filled with solids/m^3 total drum volume), typically 10–20%
ρ: density of the solid volume holdup (t/m^3)
Q: agglomerate throughput (tpd)

Combining (2) and (3) into (4) yields

$$Q = 1440\frac{\pi D^2 L}{4}\frac{\rho f}{\left(\frac{1.77\sqrt{45}L}{(90-\theta)DN}\right)} = 1440\frac{\pi D^2 L}{4}\frac{\rho f}{\left(\frac{1.77\sqrt{45}L}{(90-\theta)D\left(\frac{\alpha\cdot 42.3}{\sqrt{D}}\right)}\right)}$$

$$= 4039D^{5/2}\rho f(90-\theta)\alpha \tag{5}$$

Assuming $\rho = 1.3\,\mathrm{t/m^3}, f = 0.15$, $90-\theta = 5°$, and $\alpha = 0.40$ (5) simplifies to:

$$Q = 1571D^{5/2} \tag{6}$$

Predictions from (2), (3), and (6) were compared to laboratory and industrial data from the heap leaching, fertilizer, and iron-ore industries.

Figure 5 compares the predicted and actual normal rotation speeds of drum agglomerators. The lower and upper curves correspond

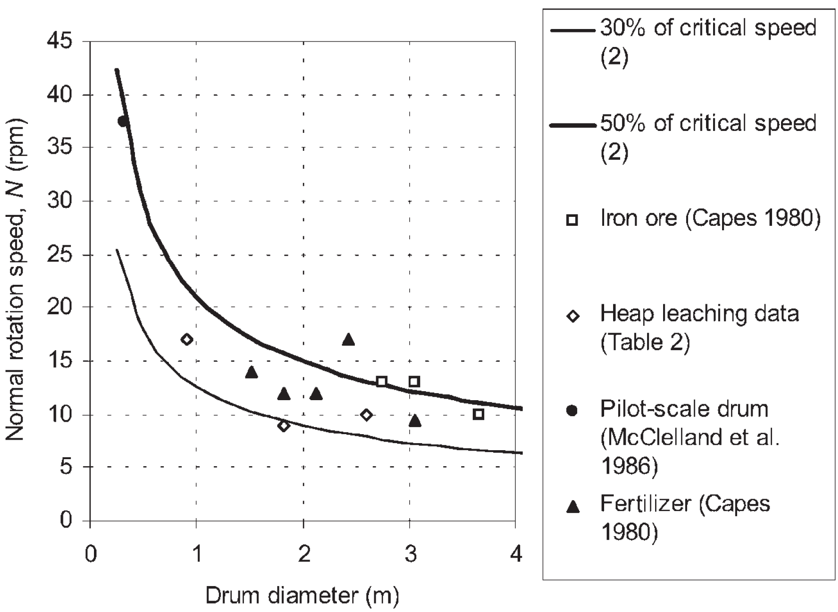

Figure 5. Comparison of the predicted and actual normal rotation speeds of laboratory and industrial drum agglomerators. The lower and upper curves define the boundaries of the predicted normal rotation speed, N.

to 30% and 50%, respectively, of the critical speed, C, as calculated from (2). There is very good agreement between the various sets of data and the predicted rotation speed. Half of the data points lie on the invisible 40% α-line. This value seems reasonable for future calculations.

Equation 3 was used to compare the predicted and actual residence times of the drums listed in Tables 1 and 2. The drum diameter and length data was obtained directly from these tables. If unavailable, the rotation speed was estimated from the reliable (2), assuming α equal to 40%. Equation 3 assumed θ and S equal to 45° and 5°, respectively. The predicted residence times ranged from 14 to 53 s. These agree well with the actual residence times in Tables 1 and 2. The actual residence times may be too short for proper agglomeration. Chamberlin (1986) suggested a residence time of at least 60 s for coarse ore and 240 s for fines. For better mixing and longer residence time, a drum can be equipped with a dam.

Equation 6 was used to predict the throughput of all agglomerators shown in Figure 5 and all agglomerators in Tables 1 and 2 for which the diameter was known. Figure 7 shows a poor agreement between the predictions of (6) and the actual throughputs. The discrepancies may be attributed to the empiricism of (3) and the assumption of piston flow behavior in the drum. If the flow behavior resembled more well-mixed conditions, the residence time would be shorter and, thus, the agglomerate throughput would be greater.

Attempts were made to derive a simple empirical equation that predicts the throughput as a function of the drum diameter. Such an equation exists for disc agglomerators, as will be shown in (10). The proposed equation takes the form:

$$Q = kD^m \tag{7}$$

Parameters k and m were evaluated by minimizing the sum of the squared difference between the predictions of (7) and the actual data shown in Figure 6. Parameters k and m were found to be 23.2 and 4.7, respectively, for values of drum diameter, D, ranging from 0 to 4.5 m, and for ore throughput, Q, expressed in tpd. This equation provides reasonable estimates of the ore throughput for drums up to 2.5 m in diameter (Figure 6). The wide data scatter above 2.5 m in diameter was not well predicted by this model.

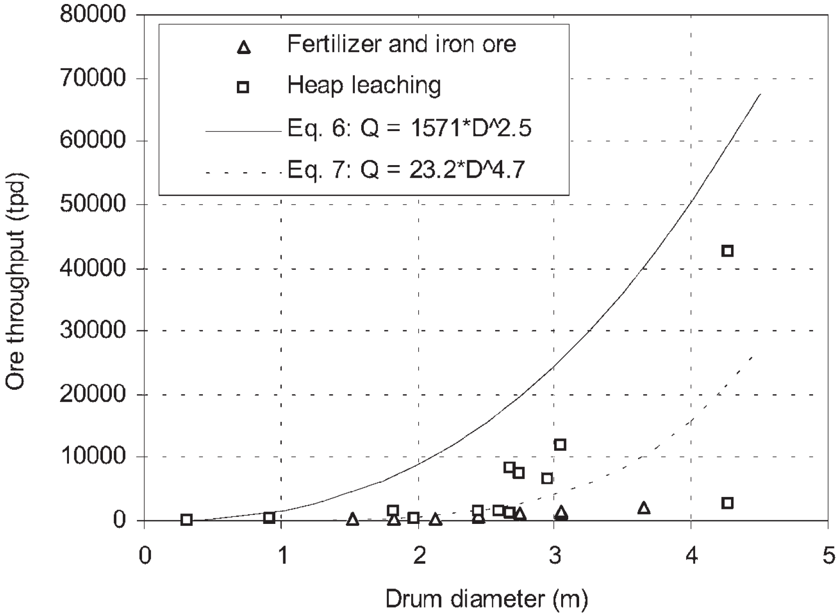

Figure 6. Comparison of the predictions of Eqs (6) and (7) and actual ore throughputs of drum agglomerators. Variables Q and D have units of tpd and metre, respectively.

Another empirical equation is proposed to predict the drum ore throughput as a function of the drum diameter, length, and rotation speed:

$$Q = kD^m L^n N^p \tag{8}$$

To estimate the four parameters, only data sets with known diameter, length, and rotation speeds were considered in the least-square minimization calculation. There were six sets from the fertilizer agglomeration tests, three sets from the iron-ore agglomeration tests, and four sets from the heap leaching agglomeration tests. The parameters k, m, n, and p took values of 823, 0.34, 1.08, and -0.88, respectively, for D expressed in meters, L in meters, N in rpm, and Q in tpd. Inserting these values in (8) yielded a very good fit of the 13 actual throughput values across a wide range of drum diameters (Figure 7). This equation also predicted very well the throughput of four incomplete sets from heap leaching agglomeration tests. However, the predictions were unreliable for throughputs above 5,000 tpd.

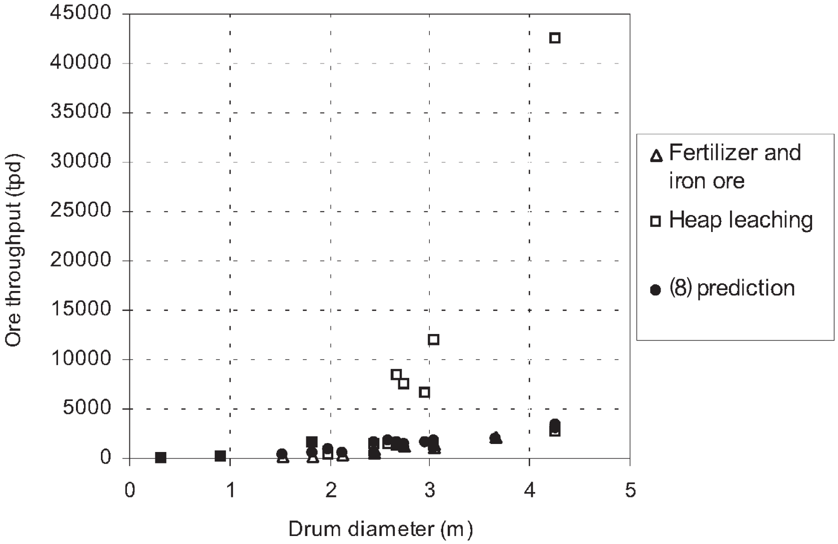

Figure 7. Comparison of the prediction of Eq. (8) and actual ore throughputs of drum agglomerators.

Since the length of the drum is typically three times longer than the drum diameter (Figure 8), (8) can be modified to include the length to diameter ratio. The revised equation is:

$$Q = 823\frac{D^{0.33}L^{1.08}}{N^{0.88}} = 2695\frac{D^{1.41}}{N^{0.88}} \tag{9}$$

More data on the rotation speed and residence time should be acquired to improve the fit of (9) for throughputs above 5,000 tpd.

Disc Agglomeration

A disc agglomerator (Figure 9), also known as disc granulator or pan granulator, consists of a rotating, tilted disc or pan with a rim. Disc agglomerators produce pellets of uniform size. There is little to no solid recycling. Solids and solution are continuously added to the disc. A coating of the feed material builds up on the disc and the thickness of this layer is controlled by scrapers or a plow, which may oscillate mechanically. Feed rate variations affect the rolling action. Solution is applied by a series of spray nozzles distributed across the face of the bed. A coarse spray onto fines favors the formation of nuclei for growth.

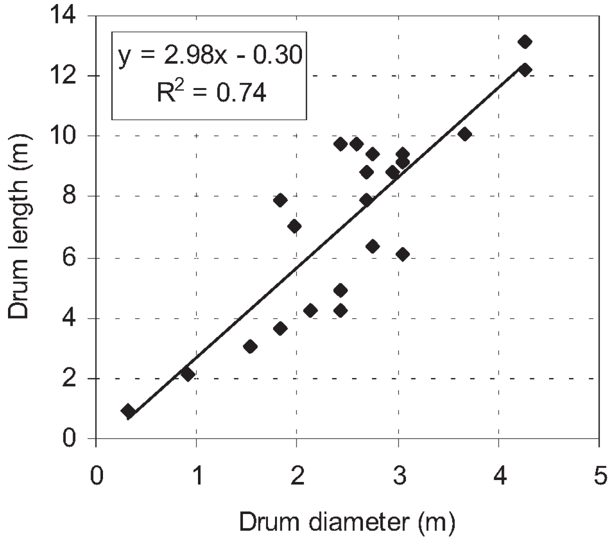

Figure 8. Influence of the drum diameter on the drum length. Data obtained from Tables 1 and 2.

Solid feed and spray nozzle locations influence significantly the agglomeration performance and pellet structure. Pellets collect in the eye of the pan, located above the bottom rim of finer pellets (Figure 9).

The normal rotation speed and the throughput can be calculated using similar equations as for the drum agglomerator. The critical rotation speed, C, is also given by (2), where θ is replaced by a value between 45–65°. To calculate the normal rotation speed, the fraction, α, takes a value between 50–75%. Since a disc can range in size from 0.3 m (1 ft) to 10 m (33 ft), the rotation speed, N, is calculated using (2) to range from 6 to 41 rpm when $\theta = 50°$ and $\alpha = 60\%$. The disc angle and the fraction of the critical speed are the two parameters to maintain constant in scaleup. If the rotation speed is too low, sliding occurs. If the rotation speed is too high, particles are thrown off the disc or openings develop in the disc, allowing spray blowthrough and uneven buildup on the disc bottom.

The residence time varies typically from 60 to 120 s. It can be increased by lowering the solid feed rate or the disc angle or by increasing the rotation speed or the bed depth. The solid flow pattern lies between well mixed and piston behavior.

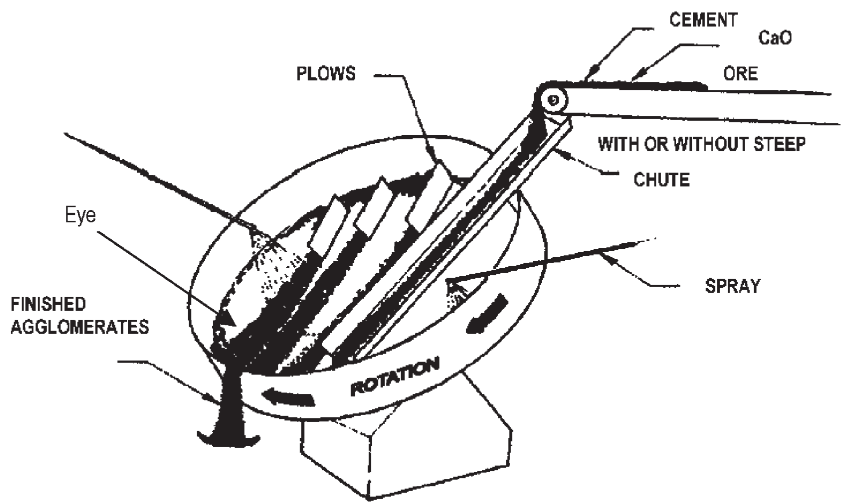

Figure 9. Disc agglomeration equipment. Reproduced from Chamberlin (1986). Reproduced with permission from *Mining Engineering Magazine*.

The thickness of the pan walls is typically 10–20% of the diameter. The throughput is proportional to the square of the pan diameter:

$$Q = kD^2 \tag{10}$$

where k takes a value between 0.5 and 1.2, Q has units of tph, and D has units of meters (Capes 1980). Equation 10 applies for throughputs ranging from 5 to 1,700 tpd.

Disc agglomerators are used in the iron ore, agricultural, and chemical industries, but not for heap leaching, possibly because of their lower throughputs of up to 2,400 tpd, compared to throughputs of more than 5000 tpd for heap leaching.

Other Agglomeration Equipment

Ores containing little clays and fairly coarse particles can also be agglomerated by far simpler means, such as dumping water and possibly a solution of low cyanide concentration into the haul trucks or spraying moisture (water or cyanide solution) on the sides of the heap throughout the construction. As the ore is rolling down the slopes, fines stick to coarse particles. This is referred to as stockpile agglomeration (Figure 10). Even the dozer moving the ore up or down the slopes can help particle agglomeration.

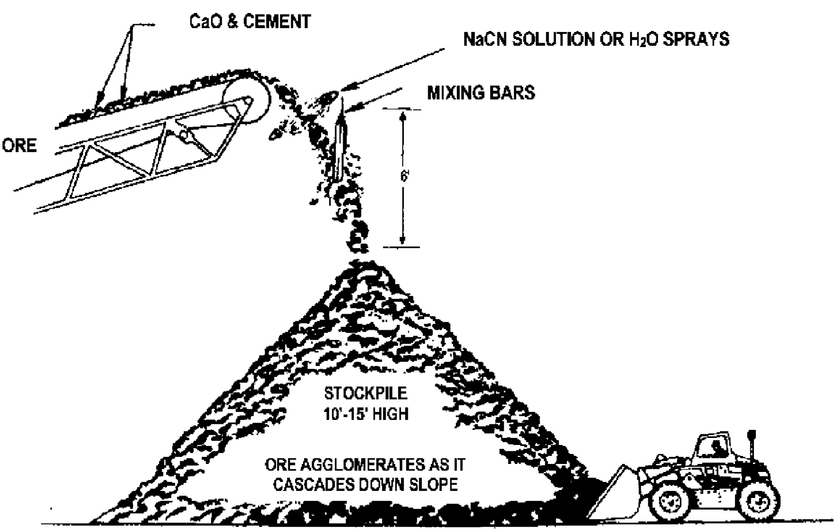

Figure 10. Stockpile agglomeration. Reproduced from Chamberlin (1986). Reproduced with permission from *Mining Engineering Magazine.*

Tibbals (1987) was a fervent proponent of pug mills, which are devices equipped with a horizontal trough in which a central shaft slowly rotates, to which is attached mixing blades, bars, rods, or paddles. In his opinion, the power consumption of pug mills is less than the drum agglomerator, the capital cost is 25–50% of the costs of the drum, and the mixing action is comparable to the drum. Pug mills were used at Haywood-Santiago and Florida Canyon operations (Nevada). Pug mills and disc agglomerators are very rarely used, if at all, in heap leaching, as confirmed by Kappes' recent survey (2002).

MOISTURE REQUIREMENTS

In the mining industry, agglomeration and pelletization utilize a liquid, such as a cyanide solution in the gold industry or a sulfuric acid solution in the copper industry, to bind particles together by a liquid film. Without the addition of a binder, the surface tension and van der Waals forces that hold the particles together are weak. Particle adhesion created by surface tension may fail when agglomerates retain more moisture under irrigation. In fact, overwet ore looses its ability to agglomerate. Curing is also nonessential when agglomerating with solution alone, unless if using sulfuric acid, thought to react with gangue minerals (e.g, kaolinite) to render them amorphous and to inhibit silica dissolution (Cruz et al. 1980; Farias et al. 1995). In fact, agglomerates inoculated with a microbial inoculum should remain moist to avoid cell desiccation. This can be achieved by not aerating the pile before the onset of irrigation.

Agglomerating gold ores with a cyanide solution does not affect the chemistry of the agglomeration, as opposed to the chemical bridges created by the hydration reaction between dicalcium silicate, tricalcium silicate, and water when using cement. Chamberlin (1980) reported, however, that a cyanide solution improved particle bonding compared to water alone, but did not have any effect on gold extraction. The uniform presence of cyanide in the heap before irrigation may shorten the time required for cyanide transport to the gold surfaces. Worstell (1987) believes this phenomenon to be the rate-limiting step in cyanide heap leaching, among oxygen gas/liquid transfer, oxygen and cyanide adsorption on the gold surface, electrochemical dissolution of gold, desorption of the gold–cyanide complex, and gold transport to the bottom of the pile. The benefits of adding cyanide during agglomeration would translate in an overall faster recovery. O'Brien (1982) demonstrated that the agglomeration of two gold ores with Portland cement and a cyanide solution reduced the leach cycle by 4–10 days compared to the Portland cement and water combination. The tests were performed in columns only, not in a heap.

Evidence suggests that the use of strong cyanide solution in agglomeration increases the cyanide consumption. The use of a strong cyanide solution can also be a safety hazard for toxic gas emanation at too low solution pH's. A dosage of 1 kg NaCN/t of ore (2 lb/ton) is fairly typical for agglomeration. At least half of the former and present gold heap leach operations that agglomerate(d) use(d) a cyanide solution rather than water alone (Table 2). There is no specific guideline for the appropriate moisture addition. The following equation has been proposed to estimate the moisture content of agglomerate:

$$\varpi = \frac{1}{1 + 2.17\rho_s/\rho_l} \quad \text{for size} > 30\,\mu\text{m} \tag{11}$$

where ω is the moisture content in kg liquid/kg dry ore, ρ_s is the solid true density, and ρ_l is the density of the liquid having similar properties as water (Perry's Chemical Engineering Handbook). For a solid density of 2.8 g/cm^3 and a water density of 1.0 g/cm^3, the moisture content is approximated to be 14 wt%. This equation does not account for the initial moisture content of the ore, its mineralogy, its particle size distribution, and the use of binders. All affect the agglomerate final moisture content. Tailings, which contain a greater proportion of fines and, thus, more surface area for wetting, typically require twice as much moisture as crushed ore (15–30 wt% vs 5–15 wt%). If the material to be agglomerated were too wet, it should be dried, as excessive moisture does not produce individual agglomerates, but rather clumps of agglomerates that do not roll down the slopes of the heap. A starting material that is too wet also limits the quantity of soluble reagents that can be mixed with the ore. To further support the negative impact on the agglomerate quality of adding too much moisture, Zárate and Guzmán (1987) found that the quality of cement/lime-based pellets was optimum at a moisture of 17 wt%, but declined at higher water dosages. All three methods employed in their work to measure the agglomerate quality (dip, flooding, and compaction) recommended the same optimum moisture content. Heinen et al. (1979) observed the same phenomenon with agglomerated tailings.

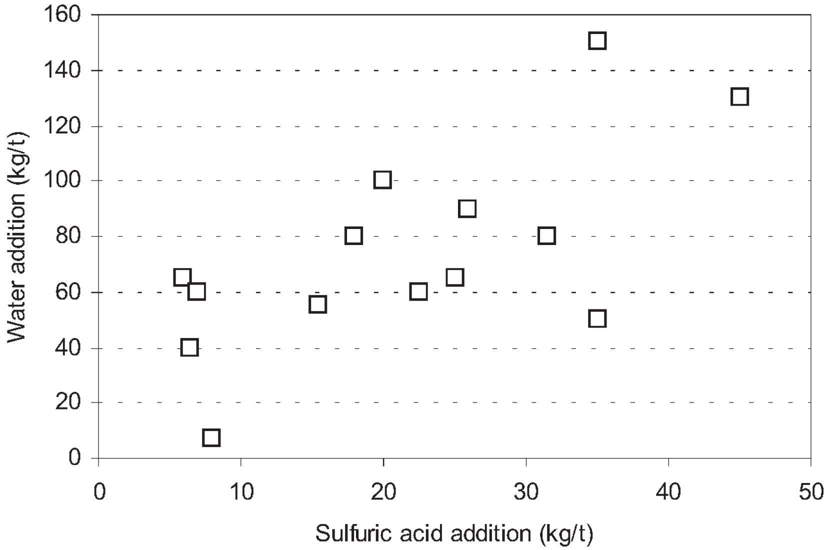

Figure 11. Survey of the sulfuric acid and water additions of 14 copper heap leaching operations. Each data point represents one plant.

Copper heap leaching operations agglomerate with water and sulfuric acid. Figure 11 illustrates the water and concentrated sulfuric additions for 14 copper heap leaching operations. On average, 15–25 kg sulfuric acid/t of ore is added to 60–100 kg water/t of ore (120–200 lb/ton). The final moisture content ranges between 7.5–12.5 wt%, a typical moisture for agglomerated crushed ore. There appears to be a proportional relationship between water and sulfuric acid. The high sulfuric acid and water additions on the right-hand side of Figure 11 could be associated to finely crushed ore having a larger acid demand. The low sulfuric acid and water addition on the left-hand side of Figure 11 could be related to the coarseness of the particles, to the higher than usual initial moisture content of the feed material, and possibly due to wet screening or to the low acid consumption by gangue minerals.

Holle (1996) found that larger amounts of acid added during agglomeration increased the copper recovery by 30%. Phelps Dodge's Morenci Mine-for-Leach operation (Arizona) recovered also 15% more copper with the addition of 5 kg/t of sulfuric acid (10 lb/ton, previously 0 kg/t) to agglomeration. The increase in copper recovery was proportional to the acid dosage (Uhrie et al. 2003).

Sulfuric acid was also used to agglomerate radioactive uranium-bearing tailings from Ranchers Exploration and Development Corporation in a 3.7 m (12 ft) wide by 10.1 m (33 ft) long drum rotating at 7–16 rpm (Scheffel 1982). 544,000 tonnes of agglomerates prepared with 45–68 kg/t (90–136 lb/tonnes) of sulfuric acid were stacked. Some success had been obtained using Polyox WSR 301 and sulfuric acid, but its addition was considered unnecessary.

Others have employed a hypochlorite solution to oxidize sulfidic refractory ore (Perez et al. 1990; Ahmadiantehrani et al. 1991). Perez et al. (1990) patented the use of a sodium or calcium hypochlorite solution in the amount of 2.5–22.5 kg/t (5–45 lb/ton) of chlorine and 2.5–10 kg/t (5–20 lb/ton) of cement or gypsum to destroy, modify, or passivate sulfidic minerals. Agglomerates cured for 1–3 days before rinsing with water and cyanide leaching. Previous tests by Ahmadiantehrani et al. (1991) had consumed up to 80 kg/t (160 lb/ton) of hypochlorite (equivalent to 55 kg/t or 110 lb/ton of chlorine) due to the oxidation of gangue minerals, but recovered 80% of the gold content. This consumption was reduced to 9.5 kg/t (19 lb/ton) of chlorine by agglomerating with cement and by operating at low temperatures (3–15°C). This did not affect the gold recovery.

The method of solution addition is not critical for crushed ore since there are several large particles that can act as a nucleus for fines adhesion. Tibbals (1987) suggested that the amount of energy input for agglomeration of crushed ore was more important than the method of solution addition. When the ore particle size is smaller than the droplet size, as for tailings, these authors (Tibbals 1987; Eisele and Pool 1987) agree that the moisture should be added as droplets rather than as an atomizing spray. Fines adhere to the droplet to create a nucleus. Atomizing spray do not form nuclei; the wet fines do not agglomerate.

BINDER REQUIREMENTS

The weak forces of adhesion between water and ore particles can be strengthened with binders. A binder can be a liquid or solid that forms a bridge, film, or matrix, or that causes a chemical reaction. The proportion of fines and clays in the ore determines the need and dosage of binder. For instance, Chamberlin (1986) recommended adding cement for gold ores containing more than 10% of material smaller than 75 μm (200#). Dry binders should be mixed with the ore, preferably in the crushing circuit, at the entrance of the drum agglomerator, or added on the first conveyor. Liquid and viscous binders should be mixed with the agglomerating solution for better distribution.

The pH of the heap leach operation determines the nature of the binder selected. There are numerous binders for alkaline pH's, including lime, cement, silicate, pozzolan, and polymer. Fourteen of nineteen past and present cyanide heap leaching operations agglomerated with cement (Table 2). Very few copper heap leaching operations add any binder to the sulfuric acid/water combination, possibly because of the limited selection of acid-tolerant and microbial-resistant binders (polymers and gypsum). The next sections describe the chemistry, dosage, and strength-producing mechanisms of several binders.

Lime, Pozzolan, and Silicate

To minimize cyanide losses as HCN at pH's less than 9, lime [$Ca(OH)_2$] is added to gold and silver ores in amounts from 1.5 to 25 kg/t (3–50 lb/ton) to provide alkalinity. Lime was found to be a less effective binder than Portland cement. Lastra and Chase (1984) recommended lime agglomeration for ores containing no clays, either on belts for coarse particles or on belts with vibrating chutes for particles smaller than 12 mm (1/2 in). Lime has long been known to strengthen clays through three mechanisms: 1) formation of carbonates; 2) destructuring the clay minerals by raising the pH; and 3) by ion exchange, in which natural monovalent cations adsorbed onto the clay minerals are exchanged by divalent cations. The latter mechanism reduces the diffuse double layer thickness, thereby minimizing clay swelling.

Litz (1993) and an article in the November 1992 issue of the Engineering & Mining Journal (Anonymous 1992) reported on the development of *Leach-It*, a modified lime patented (U.S. Patent 5,116,417) by Chemstar Lime Co., Phoenix, AZ. This product crystallizes into cementitious torbemorite and ettringite minerals, producing a dentritic structure.

Walker and Oliphant (1992) also developed and patented a mixture comprised of 10–80% calcerous component, such as quick lime, 5–50% siliceous-calcerous component, such as fly ash, and 10–80% sulfated component, such as gypsum. The mixture should be added in the amount of up to 2 wt% to precious metal ores for cyanide heap leaching.

Pozzolan, such as cement kiln dust, fly ash, granulated blast furnace slag, and other metallic slags, is a fine siliceous or siliceous and aluminous material that can be used as a binder. When mixed with lime and water at ambient temperature, the glassy, fine particulates produce cementitious compounds such as calcium silicate hydrate gel and calcium aluminosilicate. Pozzolan sets slowly in comparison to cement.

Solid or liquid silicates are produced by the fusion of typically 1.5–3 parts sand with 1 part sodium or potassium carbonate. They are employed as agglomeration binders, particularly if the agglomerates are to be dried. The sand to carbonate ratio is determined by the binding mechanism, by the setup time, and by the material agglomerated. Starch, glycerin, molasses, dextrin, and lime may be added as additives to increase the agglomerate strength.

Silicates produce different types of binding mechanisms, such as hydration, precipitation, polymerization, and surface charge modification. Polymerization occurs rapidly when the pH of liquid silicate drops below 10.7. Sodium silicates react almost instantly with multivalent metal cations to form the corresponding insoluble metal silicates. The metal-robbing property makes silicates undesirable in heap leaching where the dissolved metals must remain soluble.

A crushed ore containing uranium was agglomerated in a drum with 4.1–4.5 kg/t (8.2–9 lb/ton) of sodium silicate, 11–12.6 kg/t (22–25 lb/ton) of concentrated sulfuric acid, and 66–96 kg/t (132–192 lb/ton) of water to produce agglomerates 10–30 mm (3/8–1 in) in diameter (Videau and Roche 1990). The 1,000-tonnes demonstration-scale heap containing no silicates ponded and channeled. The heap agglomerated with silicates was irrigated at $20\,L/m^2/h$ ($0.008\,GPM/ft^2$). Even though the heap containing silicates eventually blew out in certain zones, this heap leached uniformly.

Lime, pozzolan, or silicates are not as popular as Portland cement for precious metal heap leaching.

Cement

Cement is the preferred binder for precious metal ore agglomeration. Cement is comprised of 50–70% tricalcium silicate ($3CaO{\cdot}SiO_2$), 15–30% dicalcium silicate ($2CaO{\cdot}SiO_2$), 5–10% tricalcium aluminate ($3CaO{\cdot}Al_2O_3$), 5–15% tetracalcium aluminoferrite ($4CaO{\cdot}Al_2O_3{\cdot}Fe_2O_3$), and various hydrated forms of gypsum ($CaO{\cdot}SO_3{\cdot}H_2O$) (Kosmatka et al. 2002). There are five types of cement (10, 20, 30, 40, and 50) under the Canadian Standards Association Standard A5 (CSA), and eight types of cement (I, IA, II, IIA, III, IIIA, IV, and V) under the ASTM C150 standard. CSA A5 types 10–50 are, respectively, essentially the same as ASTM C150 cements Types I–V.

Calcium silicates hydrate to form calcium hydroxide ($CaO{\cdot}H_2O$) and calcium silicate hydrate ($3CaO{\cdot}2SiO_2{\cdot}8H_2O$). Tricalcium aluminate participates in three reactions. It reacts with gypsum to produce ettringite ($6CaO{\cdot}Al_2O_3{\cdot}3SO_3{\cdot}32H_2O$), with ettringite to produce calcium monosulphoaluminate ($4CaO{\cdot}Al_2O_3{\cdot}3SO_3{\cdot}12H_2O$), or with calcium hydroxide to produce tetracalcium aluminate hydrate ($4CaO{\cdot}Al_2O_3{\cdot}13H_2O$).

Tetracalcium aluminoferrite reacts with water and calcium hydroxide to produce calcium aluminoferrite hydrate ($6CaO{\cdot}Al_2O_3{\cdot}Fe_2O_3{\cdot}12H_2O$). These reactions are summarized in Table 4. The strength of hydrated cement is due primarily to calcium silicate hydrate. Tricalcium silicate and aluminate hydrate and harden rapidly. The addition of silicates accelerates the set of cement.

Cement types I–V are not resistant to acids or highly corrosive substances. Calcium, sodium, and magnesium sulfates may attack calcium aluminate hydrates and calcium hydroxide to form ettringite, gypsum, and brucite (magnesium hydroxide). Cement with a low percentage of tricalcium aluminate is more resistant to sulfate.

Cement accounts for 7–15% of a concrete mixture, while it typically represents less than 1% in agglomeration of ore and tailings. Electron

Table 4. Cement compound hydration reactions

$$2(3CaO{\cdot}SiO_2) + 11H_2O \rightarrow 3CaO{\cdot}2SiO_2{\cdot}8H_2O + 3\,(CaO{\cdot}H_2O)$$

$$2(2CaO{\cdot}SiO_2) + 9H_2O \rightarrow 3CaO{\cdot}2SiO_2{\cdot}8H_2O + 1\,(CaO{\cdot}H_2O)$$

$$3CaO{\cdot}Al_2O_3 + 3\,(CaO{\cdot}SO_3{\cdot}2H_2O) + 26H_2O \rightarrow 6CaO{\cdot}Al_2O_3{\cdot}3SO_3{\cdot}32H_2O$$

$$2(3CaO{\cdot}Al_2O_3) + 6CaO{\cdot}Al_2O_3{\cdot}3SO_3{\cdot}32H_2O + 4H_2O \rightarrow 3(4CaO{\cdot}Al_2O_3{\cdot}3SO_3{\cdot}12H_2O)$$

$$3CaO{\cdot}Al_2O_3 + CaO{\cdot}H_2O + 12H_2O \rightarrow 4CaO{\cdot}Al_2O_3{\cdot}13H_2O$$

$$4CaO{\cdot}Al_2O_3{\cdot}Fe_2O_3 + 2(CaO{\cdot}H_2O) + 10H_2O \rightarrow 6CaO{\cdot}Al_2O_3{\cdot}Fe_2O_3{\cdot}12H_2O$$

microscopy showed calcium to be uniformly distributed throughout agglomerates of clayey particles (Heinen et al. 1979). The nature of the bridging holding the clayey particles was unidentifiable.

Keuhey and Coughlin (1983) reported a rare instance of lower gold extraction from cement-based agglomerates. Eight percent less gold was recovered after 32 d from a column containing agglomerates prepared with 10 kg/t (20 lb/ton) of cement type II. Cement has been found to be most effective in high siliceous ores (crushed rocks) and less effective in ores having high clay content.

Eisele and Pool (1987) recommend a dosage of 7.5 kg/t (15 lb/ton) of cement and 7.5 kg/t (15 lb/ton) of lime for tailings. They measured drainage rates in excess of 20,000 $L/m^2/h$ (compared to typically 6–12 $L/m^2/h$ for heap leaching) from five columns loaded with agglomerates prepared with these dosages of cement and lime. They also recommended this equal dosage of cement and lime for crushed ores. Based on three independent tests to assess the agglomerate quality, Zárate and Guzmán (1987) instead recommended twice as much of each binder, i.e., 8–12 kg/t (16–24 lb/ton) of cement and lime. On the other hand, McClelland (1986) and Heinen et al. (1979) recommended 2.8–5.5 kg/t (5–11 lb/ton) of cement, without or with lime at 3.9 kg/t (7.7 lb/ton).

Former and current industrial cyanide heap leaching operations agglomerate crushed ore with only 2.5 to 5 kg/t (5.10 lb/ton) of cement (only in a few instances with lime also). Tailings, which are typically comprised of particles 100 μm (150#) in size, require roughly twice as much cement as crushed ores, anywhere from 5 to 17 kg/t (10–35 lb/ton) ton) (Figure 12).

Gypsum

Sulfide heaps are irrigated with an acidic sulfate solution. Adding cement or lime to sulfide ores results in the precipitation of gypsum and jarosite [$Ca_{1/2}Fe_3(SO_4)_2(OH)_6$]. Lastra and Chase (1984) saw promise in gypsum and gypsum-derived binders. These precipitates could potentially coat the mineral surfaces and yield solution of too high pH's.

Polymers

Nalco Chemical Company (Illinois) and Betz Dearborn, Inc. (Pennsylvania) have developed several polymers for ore agglomeration under acidic or alkaline conditions. These polymers are either anionic, cationic, or

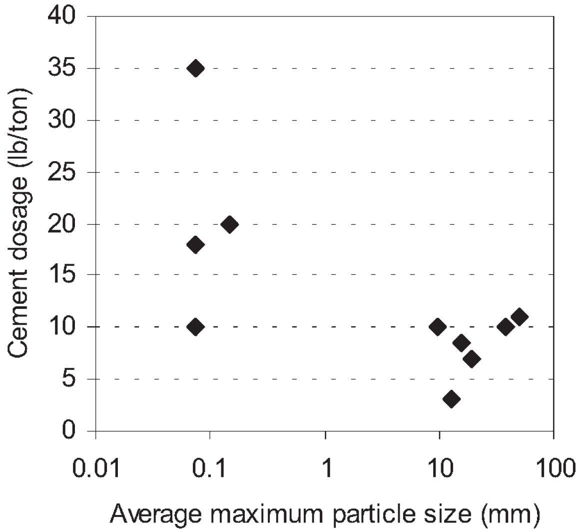

Figure 12. Comparison of the cement requirements for the agglomeration of precious metal tailings (left-hand side of graph) and crushed ore (right-hand side of graph).

both in nature. Tables 5–7, summarizing the chemicals patented by each company, show that acrylamide is the basis for most polymers. The combination of anionic 70/30 mole percent of polyacrylamide/poly(acrylamide/sodium acrylate) and cationic 90/10 mole percent of poly(diallyldimethylammonium chloride) to poly(acrylamide/diallyldimethylammonium chloride) in a dosage ranging from 0.1 to 2 kg/t (0.2–4 lb/ton) is a novel binder developed by Nalco Chemical Company (Illinois).

Table 5. Cationic polymer binders

Company	Inventor(s)	Patent	Agglomerating agent	Suited for
Betz Dearborn, Inc.	Polizzotti et al.	5,668,219	Block polymer containing a block of an ammonium cation or acrylamide having a molecular weight of up to 200,000	Copper or gold ores
Betz Dearborn, Inc.	Polizzotti et al.	5,512,636	Polymers of acrylamide and diallyl dimethyl ammonium chloride	Copper or gold ores

Table 6. Anionic polymer binders

Company	Inventor(s)	Patent	Agglomerating agent	Suited for
Betz Dearborn, Inc.	Polizzotti et al.	5,077,022 5,077,022 5,112,582 5,186,915 5,211,920	Polymers of acrylamide and acrylic acid having a molecular weight from 1 to 8 million, in combination with lime	Gold ores
Nalco Chemical Company	Gross	4,898,611 5,100,631	Polymers of acrylamide and acrylic acid of molecular ranging from 1 to 20 million, with or without cement	Gold ores
Nalco Chemical Company	Gross et al.	4,342,653 4,786,318 4,875,935	Polymers of acrylamide with carboxylate or sulfonate groups	Copper ores
Allied Colloids Limited	MacDonald	4,587,108	Polymers of acrylamide with 2-acrylamido-2-methyl propane sulphonic acid	Uranium ores

Acrylamide or its derivatives are the common base of most polymeric binders. They are of high to very high molecular weights. Considerable portions of the chain are unattached to the particle. The

Table 7. Other polymer binders

Company	Inventor(s)	Patent	Agglomerating agent	Suited for
Betz Dearborn, Inc.	Cifuentes	WO99/63123	Polypropylene glycol and alkylphenol ethoxylate in a paraffin oil solvent (known as PEG 400 MOT) having a molecular weight of 400	Gold ores
Nalco Chemical Company	Kerr	WO99/20803	Cationic and anionic polymers added sequentially having a molecular weight of up to 30 million, in combination with cement for gold ores	Copper and gold ores

free, dangling portions of the chain then contact and adsorb onto other particle surface. When added without lime or cement, anionic acrylamide chains likely adsorb onto the negatively charged rocks via hydrogen bonding. The frequent combination of lime or cement with acrylamide binders change the binding mechanism from hydrogen bonding to electrostatic bridging. The divalent calcium ions form an electrostatic bridge between the negatively charged particle surface and the negatively charged carboxyl groups of the acrylamide–acrylic acid copolymer.

Nalco Chemical Company (Illinois) has also developed polymeric binders that would also not inhibit the microorganisms present and would not be consumed by the microorganisms (Kerr 1999). Of the following binders (9704, CX-2131, CX-2134, CX-2185, CX-2194, 98DF108, and 97DF125) at doses ranging from 0.2 to 0.8 kg/t (0.4–1.6 lb/ton), products 9704, 98DF108, and 97DF125 yielded the smallest slump, between 12–19%, after 30 min of irrigation in columns loaded with a clayey ore. The baseline column without binder slumped by 31%. The performance of these binders in the field was comparable to the slump in columns. The columns did not slump further after leaching. Other indicators of the performance of these binders included: 1) the rate of ferrous oxidation as an indirect method of binder compatibility with microbial species, 2) the turbidity of the column discharge solution, and 3) the solution flow rate after compression (Kerr 1999). Binder 97DF125, a cationic polymer of medium molecular weight, was superior based on these criteria. Binders 97D125 and 9704 are commercially available at a cost ranging from $US 0.50/t of ore to $US 2.00/t of ore, based on the above dosages. The CX-binders remain research products to date.

Nalco binders were utilized by the Toquepala/Caujone Copper Mine in Chile and Asarco's Ray Mines Division in Arizona to belt-agglomerate copper ores. Lately, WMC Resources' Nifty Copper Operation (Australia) has utilized two other products commercialized by Nalco in a demonstration-scale heap. 30,000 tonne of copper ore were agglomerated with 1 kg/t (2 lb/tonne) of Nalco's Extract-Ore® 9560 and 5,000 tonne of ore were agglomerated with 1 kg/t (2 lb/ton) of a less-expensive Nalco polymeric binder. No baseline test without binders was conducted. Extract-Ore™ is a medium molecular weight latex copolymer of moderate anionic charge. The polymer consists of water-swollen micro gels, approximately 1 μm in diameter. When diluted in water, the micro gels instantly solubilize, exposing the functional groups along the polymeric backbone to the charged sites of the ore particles. Overall, 73.5% of the copper was recovered in 303 d from the two heaps agglomerated with the polymeric binders, in spite of unleached and partially leached zones identified in the heap at closure. 73.5% was less than the expected 90% obtained in a large column (Efthymiou et al. 1998), but above expectations based on previous other treatment schemes.

Newmont Gold Mines in Carlin, NV, and Philex Gold's Sibutad Project in the Philippines also employed Nalco binders and cement to agglomerate gold ores. Brewer Gold found that the column gold recovery was superior with the use of 125 g/t (0.25 lb/ton) of Nalco Extract-Ore^{TM} 9760 and 2.5 kg/t (5 lb/ton) of cement compared to an ore agglomerated with 7.5 kg/t (15 lb/ton) of cement (Pautler et al. 1990). A column containing ore agglomerated with 125 g/t (0.25 lb/ton) of Nalco 9760 drained more slowly than a column containing ore agglomerated with cement. In the field, the dual mixture of cement and Nalco 9760 performed better than cement alone. Two 10.7-m (35-ft) tall heaps were stacked side-by-side and irrigated at 12.5 $L/m^2/h$ (0.005 GPM/ft^2). One half was agglomerated with cement and Nalco 9760; the other half was agglomerated with cement. Half of the solution applied on the heap without Nalco 9760 discharged from the heap with Nalco 9760. This suggests severe channeling.

The Betz Dearborn Company (Trevose, Pennsylvania) developed binders HL 9120 and HL 9121 for the agglomeration of gold ores (Polizzotti et al. 1997; Polizzotti 1993). Binder HL 9120 (U.S. patents 5,077,021, 5,077,022, 5,186,915, 5,211,920) is an anionic 70/30 to 90/10 acrylamide/acrylic acid polymer of medium molecular weight (1–10 million) added in a dose of 50 g/t (0.1 lb/ton) to 2.5 kg/t (5 lb/ton) of lime. Binder HL 9121 (U.S. patent 5,472,675) is a cross-linked borated polyvinyl alcohol added in a dose of 50 g/t (0.1 lb/ton) to 3 kg/t (6 lb/ton) of lime and 3 kg/t (6 lb/ton) of cement. The influence of the binder on the gold recovery and drainage rate was measured in columns. Binder HL 9121 improved the gold extraction by 5% under simulated heavy rainfall conditions (Polizzotti et al. 1997). The column agglomerated with the binder drained five times faster than the column agglomerated with cement (Polizzotti 1993).

Others Binders

The Research Centre for the Mining and Metallurgical Industry of Cuba has developed a proprietary binder termed *Additive 1* to agglomerate clayey copper ores (Serrano 2003). Serrano (2003) claims that *Additive 1* is low cost, resistant to acid solutions, and forms porous pellets with good mechanical resistance. A column containing a clayey ore agglomerated with 25 kg/t (50 lb/ton) of *Additive 1* slumped by 34%. A column containing clayey ore mixed with coarse ore and agglomerated with 50 kg/t (100 lb/ton) of *Additive 1* and 20 kg/t (40 lb/ton) of sulfuric acid slumped by 15%. However, a column containing the same material without coarse ore slumped by 28%. Although the dosage of *Additive 1* is large and its benefits were undermined by the coarse ore addition, *Additive 1* did not affect the copper recovery or the acid consumption.

Because of its sticky nature, molasses seems an appealing binder, acting as glue to agglomerate particles. The forces of adhesion are relatively weak. What's more, under irrigation, these forces weaken due to the molasses dissolution. However, if it is combined with hydrated lime, an exothermic reaction occurs between sucrose and lime that forms calcium sucrate, a rigid and stable material that bonds to particles. Because the reaction is fast, the reagents should be mixed immediately before agglomeration. The molasses/lime combination has been used for many years for briquetting, pelletization, and other applications involving coal fines, metal ores, fly ash, limestone, and steel mill waste. There is no mention of its use in the mining industry.

The U.S. Bureau of Mines has also tested a high molecular weight polyethylene oxide (PEO, also known as Polvox). This chemical is an effective flocculant for colloidal silica and clays in basic solution. Heinen et al. (1979) observed an improved drainage rate with the addition of 0.05 kg/t (0.1 lb/ton) of PEO mixed with lime.

CURING

Curing refers to hydration reactions between calcium silicates and water that form calcium hydroxide and calcium silicate hydrate. The latter is by far the most important cementitious component in concrete. Calcium silicate hydrate forms dense bonds between particles.

Cement does not cure by drying. Heinen et al. (1979) observed that agglomerates cured and dried broke down upon wetting. If the relative humidity of the mixture drops below 80%, cement stops gaining strength. Hydration resumes after resaturation; the strength increases again. Approximately 40 kg (88 lb) of water per 100 kg (220 lb) of cement is necessary for curing. If an ore is agglomerated with 5 kg/t (10 lb/ton) of cement and has a moisture content of, say, 8 wt% before irrigation, there should be 4 times more water available than the required amount for proper curing.

The strength continues to increase provided that unhydrated cement is still present, that the concrete temperature remains favorable, and that sufficient space is available for hydration products to form. Although 28 d of curing is the standard in the concrete industry, 8 to 24 h sufficed in previous agglomeration studies of crushed ore (Chamberlin 1986; McClelland 1986b; Eisele and Pool 1987; Zárate and Guzmán 1987). Seventy-two hours was preferable for tailings (Eisele and Pool 1987).

Herkenhoff (1987) recommends letting cement-based agglomerates cure in a separate pile for 72 h. Forming a separate pile does not seem necessary for two reasons. First, if the agglomerates can withstand being stacked in a separate pile and then moved to the heap, then they should as easily withstand the impact of stacking in the heap. Second, given that the installation of solution header lines and emitters on top of the heap takes longer than 3 d, agglomerates should have more than enough time to cure before the solution is applied.

During curing substances, like cyanide and ferrous ions present in the agglomerates, may be oxidized before the start of irrigation. Reactions of the binder or of the agglomeration solution may occur during the period between agglomeration and irrigation. Their reaction products may affect the leach chemistry during irrigation.

AGGLOMERATE QUALITY

The agglomerate quality can be defined in terms of:

- the size distribution of the agglomerate, particularly the uniformity of the agglomerate size
- the agglomerate moisture content immediately after agglomeration and during irrigation

- the agglomerate strength, and
- the agglomerate internal porosity

The iron-ore, pharmaceutical, and fertilizer industries have developed quantitative and less-subjective techniques for measuring the agglomerate quality. The reader is referred to the review paper of Pietsch (1985) for details on these methods. In this author's opinion, the unavailability of quantitative techniques lies in the inconsistencies of the feed material, in the specificities of the agglomeration conditions dictated by the ore type, and in the general acceptance that the slow, difficultly controllable, and ever-larger heap leaching processes overwrite the imperfections of the agglomerates produced.

Quantitative techniques employed in the heap leaching industry include the use of sieves to measure the size distribution of moist and dry agglomerates. Nevertheless, very few records were found in the literature about the size distribution of agglomerates.

Moist agglomerates can be screened onto a standard vibrating Gilson screen, which, if operated for a few minutes, can produce better agglomerates by the bouncing action imparted by the vibrations. Care should be exercised not to blind the screen to achieve rapid separation. Snap freezing agglomerates with liquid nitrogen prior to sizing may alleviate these inconvenients (Hall 1986).

Although more tedious, dry agglomerates can be screened by gently manually rolling agglomerates over screens. These techniques are very practical in the laboratory and in the field during periodic sampling. For more frequent and online assessment of quality, some companies have developed image analysis software that calculate the agglomerate size distribution from a reference length and a digital picture taken over the conveyor belt carrying the agglomerates. As a very rough guideline, Chamberlin (1986) suggested that no particle smaller than 104 μm (150#) should remain unattached. Let us not forget that what is an acceptable agglomerate size distribution for a given ore may not be for another.

Lipiec and Bautista (1998) emphasized that agglomeration should eliminate free fines and produce uniformly sized agglomerates. A disc agglomerator is well suited to produce uniformly sized agglomerates from tailings. However, the heap leaching industry prefers drums to discs for agglomerating crushed ore. Therefore, to produce more uniformly sized agglomerates from crushed ore without the use of a disc agglomerator, mining operations may choose one or more of the following options:

- Crush run-of-mine ore to eliminate boulders
- Screen ore to eliminate fines
- Add more transfer points with belt conveyors
- Use a longer drum operated at lower speed and setup at a lower angle
- Tightly control the moisture content of the agglomerates
- Recycle the undersize agglomerates
- Add coarse particles to shift the apparent particle size distribution, as tested by Serrano (2003) on clay-containing copper ores

The agglomerate moisture content can be determined by drying a representative sample. Although very accurate, this method requires at least a few hours until dryness. In a few hours, the moisture content of the ore fed continuously to the agglomerator may have changed from the sample previously collected. Chamberlin (1986) estimated the moisture content of the agglomerates as 1–3% less than the moisture content of a dewatered filter cake. Both methods (drying and filter cake) lack the immediateness sought after of a continuous operation. Observing that no free moisture glistens on the surface of the agglomerate also is too subjective of a method.

One of the greatest contributions to the field of agglomeration since the early work of the U.S. Bureau of Mines was the recent use of electrical conductivity at Phelps Dodge Cerro Verde operation in Peru (Fernández 2003). The electrical conductivity increases exponentially, with the largest signal detected when a liquid film forms around the agglomerates. Agglomerates prepared with three types of ores (no clay, medium clay, and high clay content), all registered 150 mA when the proportion of agglomerates smaller than 4.8 mm (4#) remained constant or was nil. Fernández (2003) related this conductivity reading to the optimum moisture content of 4% (no clay), 6.5% (medium clay), and 10% (high clay).

In addition to measuring the conductivity, each type of agglomerates was prepared at different moisture content and submitted to compaction tests. At the optimum moisture content, the slump was 10% for the no-clay ore and 25% for the high-clay ore. The extent of compaction for each ore type also increased with increasing moisture content up until the optimum moisture, and then remained fairly constant.

Fernández (2003) did not describe the setup employed nor mentioned the initial height of the bed prior to irrigation. This data could help explain why nonoptimum agglomerates produced a more permeable bed of lesser bulk density than optimum agglomerates. Expressing the slump as a percentage can be misleading if the bed does not contain the same amount of material and if the initial heights are different between tests.

Subjective tests for measuring the agglomerate strength include squeezing agglomerates into someone's hands and looking for clumping from good agglomerates. Others suggest that a clump of good agglomerates should fall apart if poked. Others look for the rolling of agglomerates on the ground after they have been thrown up into the air. Herkenhoff (1987) proposed tumbling dry agglomerates and measuring the proportion of abraded fines. Others measure the height of the drop that leads to complete agglomerate disintegration. The latter two techniques offer some quantitative basis for comparison, but can be applied only to pellets that contain particles of uniform size. No technique obtains a direct measure of the strength of agglomerates made up of particles of various sizes, such as rim agglomerates.

Other tests quantify the disintegration of agglomerates upon contact with water. For instance, Chamberlin (1986) suggested that good agglomerates submerged in water should not disintegrate for many hours. Milligan and Engelhardt (1983) measured the amount of fines produced when dipping pellets in water 10 times. The pellets must be previously cured for 6 h at 90°C and cooled before dipping. Such dip tests do not simulate the unsaturated conditions prevailing in a heap and apply primarily to pellets because of their homogeneous structure. Dip tests are more qualitative for agglomerates.

Rather than dipping pellets, Chamberlin (1986) placed them in a burette and covered the top with glass wool. He then applied water at increasing flow rates and measured the fines content in the discharge solution. This technique does not measure the disintegration of the pellets throughout the burette. The U.S. Bureau of Mines (Heinen et al. 1979; McClelland et al. 1985; Eisele and Pool 1987) rated the agglomerate strength using a more severe method that consisted of flooding a column of agglomerates and measuring the rate of drainage. A column containing good agglomerates drain rapidly.

Additional quantitative parameters can be obtained about the agglomerate quality by loading and irrigating a column. For instance, one can measure the pore space in a column before and after irrigation. The pore space, ε_b, is related to the bulk density of the column, ρ_b, and of the agglomerates, ρ_a, through the following relationship:

$$\varepsilon_b = 1 - \frac{\rho_b}{\rho_a} \tag{12}$$

where

ε_b: pore space between agglomerates (does not include pores within the agglomerates)

ρ_b: bulk density of a bed (mass of dry ore agglomerated/total volume of the bed)

ρ_a: bulk density of an agglomerate (mass of dry ore agglomerated/volume of the agglomerates)

To overcome the challenge of measuring the bulk density of an agglomerate, the pore space in (12) can be approximated as

$$\varepsilon_b \approx 1 - \frac{\rho_b}{\rho_s} \tag{13}$$

where ρ_s is the ore true density (mass of dry ore/volume of dry ore).

This equation combines the pore space in and between agglomerates in a single variable. It becomes straightforward to calculate the pore space by measuring the mass of dry ore loaded in a column, the height of the bed after slumping, the ore true density, and the volume of water necessary to flood the column.

Scaling down this concept at the agglomerate scale, one defines the agglomerate pore space, ε_a, by the following relationship:

$$\varepsilon_a = 1 - \frac{\rho_s}{\rho_a} \tag{14}$$

where ε_a is the pore space in the agglomerates and ρ_a is the bulk density of an agglomerate (mass of dry ore agglomerated/volume of the agglomerate).

Videau and Roche (1990) found that a column containing large and wet agglomerates and another column containing smaller and drier agglomerates slumped by different extent, but ultimately had the same final bulk density. The slump is, thus, a poor indicator of the agglomerate quality and does not in fact inform about the final pore space in the heap.

An even better parameter than the pore space value is the pore space value diminished by the proportion of pores filled with solution. This indicator is calculated using the ore bulk density, calculated from the tonnage stacked and the slump, and the ore moisture content during irrigation. The air-filled pore space is critical for mutiphase reactions. However, the corrected pore space still does not define the size and interconnectivity of pores, two important aspects for heap aeration.

STACKING AND IRRIGATION

The stacking equipment, the heap height, and the irrigation equipment are as important as the agglomeration itself to maintain adequate permeability.

With regard to the stacking equipment, 15 operations stacked with conveyors and 9 stacked with trucks (Kappes 2002). 100% of precious metal heap leaching operations that agglomerated in drums stacked with conveyors (Kappes 2002). Likely four of five operations that agglomerated onto belt conveyors also stacked with conveyors. The fifth operation that belt-agglomerated, and likely 100% of operations that did not agglomerate, stacked with haul trucks.

Truck stacking is suitable for run-of-mine ores. Trucks compress the surface of the newly stacked lift if stacked from the top down or the bottom lift if stacked from the bottom up. At the Alligator Ridge Mine, haul truck traffic was ultimately responsible for the low permeability of an agglomerated lift. This resulted in lateral flow of solution that broke out through the slopes of the heap and around the access ramps (DeMull and Womakc 1984). Limiting the traffic to a central road did not improve the permeability. The ground pressure and vibration from the dozer was believed to cause as severe compaction as haul trucks. The leach performance improved significantly when the haul trucks dumped ore at the toe of the new lift, leaving it to the dozer to push up against the slopes.

There are two types of conveyor stacking systems: 1) mobile conveyor unit (or grasshopper) combined with radial stacker and 2) spreader conveyor employed primarily for dynamic pads of constant width and height. Because spreader conveyors travel across the entire width of the pile, there is less segregation across the length of the pile. Radial stackers tend to create discontinuities at the intersection between ridges and fingers (Scheffel 2002).

According to published data on 17 former and current copper heap leaching operations, the maximum and average heap height reach 10 and 5.5 m (32.5 and 17.9 ft), respectively (Figure 13). Static heap designs, such as practiced as Monto Verde and Escondida, stack lifts of 5–10 m (16.5–33 ft), but install liners and drainage pipes between lifts (Scheffel 2002).

According to a 1987 survey of North American precious metal heap leaching operations, 22% of operations stacked lifts of 0.9–1.8 m (3–6 ft) tall and 30% from 2.1 to 3.7 m (7–12 ft) (Worstell 1987). These statistics agree well with those gathered by this author from publications dating back to the 1980s and are presented in Figure 13. Kappes' more recent survey (2002) revealed that the average heap height of 32 gold heap leaching operations has more than doubled to 8.9 m (29 ft). In 2002, the highest heap had 10 lifts that rose to 120 m (394 ft) (Kappes 2002).

Figure 14 shows the proportional relationship between the heap height and its bulk density. The taller the heap, the greater the bulk density. The probability of retaining more solution and of reducing the air-filled pore space increases with increasing bulk densities.

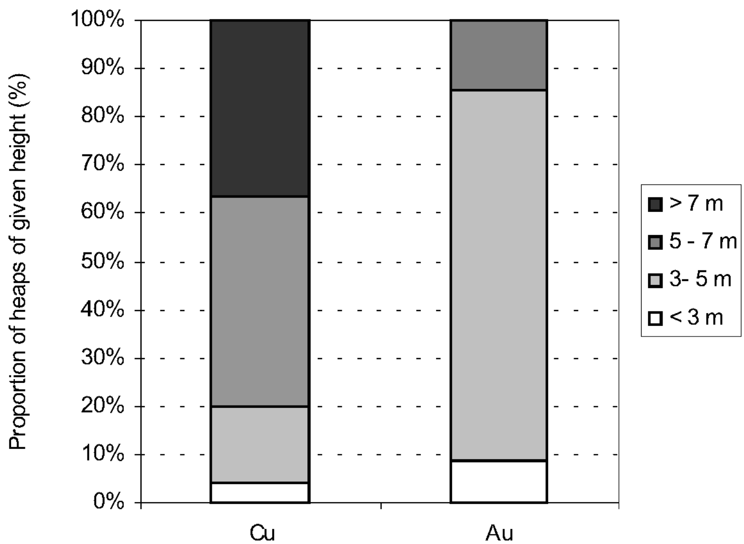

Figure 13. Survey of the heap height of 17 copper heap leaching operations and 11 gold heap leaching operations (former and current operations included).

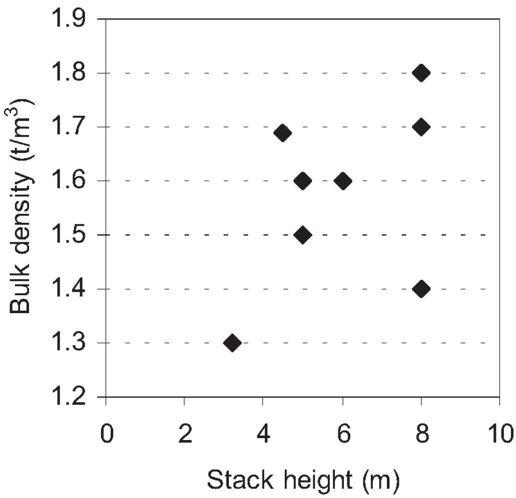

Figure 14. Influence of the heap height on the heap bulk density. Data obtained from Miller (2003).

The mode of solution application and the irrigation rate may also affect the permeability of the heap. There are three common methods of irrigation:

- drip emitters manufactured originally for agricultural purposes
- wobbler sprinkler, such as Senninger™ wobblers
- reciprocating sprinkers, often referred to as Rainbird™ sprinklers

Drip emitters consist of perforated plastic tubes or soft pipes laid parallel to each other on top of the heap. Some tubes and pipes consist only of holes evenly spaced along the length. Others contain a small labyrinth inside the tube or pipe that creates an increased pressure drop, thus producing more uniform flow over long distances. Drops emitted from these systems have the gentlest and the most local impact on the heap surface. In addition, drip emitters reduce evaporation, especially if buried. At Phelps Dodge's Morenci Mine-for-Leach heap irrigated with drip emitters, the evaporation was measured to be 12% (O'Brien et al. 2003). The main disadvantage of drip emitters is hole plugging, thus requiring the use of antiscalants and inline filters.

Wobblers are vibrating and rotating devices mounted on the header line producing multiple, small jets of coarse water droplets across a radius of 3 m (10 ft). Droplets hit the entire heap surface, producing more damage to the agglomerates than drip emitters. The better surface coverage with wobblers than drip emitters yields a greater extraction of the ore at the top of the heap.

Rainbird™ sprinklers emit a single water stream 5–8 m (16–26 ft) long that rotates 360°. They are ideal for irrigating slopes, but have high evaporative losses. Rainbird™ sprinklers are not suitable for agglomerated heaps. The strong jet disintegrates agglomerates, unless the agglomerated heap surface is covered by coarse particles to dampen the impact, as practiced by the Cerro Rico operation of Compania Mineral del Sur S.A., Bolivia.

In the late 1980s, Rainbird™ sprinklers were the most popular irrigation method of precious metal heap leaching operations (41%) (Worstell 1987). Nine percent used Bagdad wigglers, 9% used Senninger™ wobblers, 14% drilled holes in the header lines, and 5% actually built walls on top of the heap for the solution to pond on the surface. Fifteen years later, the popularity of drip emitters has grown markedly at the expense of Rainbird™ sprinklers. According to the Kappes (2002) survey of 37 operations reporting, 35% used drip emitters, 14%— all based in a tropical climate with heavy rainfall—used only wobblers, and 51% used both drip emitters and wobblers. No operation reporting used Rainbird™ sprinklers. One third of operations using drip emitters buried the tube.

With regard to the irrigation rate, precious metal heap operations that crush ore irrigate, on average, at larger flows than run-of-mine heap operations (11 vs 8.3 $L/m^2/h$) (0.0049 vs 0.0037 GPM/ft^2) (Kappes 2002). Table 8 compares the variability of the irrigation rates among run-of-mine and crushed ore heap leaching operations.

Table 8. Comparison of irrigation rates of precious metal heap leaching operations of run-of-mine and crushed ore (Kappes 2002)

Irrigation rate Number of operations reported	Run-of-mine ore 17	Crushed ore 19
$< 8\ L/m^2/h$ ($< 0.0036\ GPM/ft^2$)	N/A	21%
8–10 $L/m^2/h$ (0.0036–0.0045 GPM/ft^2)	88%	63%
$>10\ L/m^2/h$ ($> 0.0045\ GPM/ft^2$)	12%	16%

BENEFITS OF AGGLOMERATION

The benefits of agglomeration can be classified into three categories: heap physical structure, leach chemistry, and environmental impact.

Agglomeration improves the heap physical structure by minimizing or avoiding ponding, slope failure, or solution channeling. Channeling occurs in zones containing coarse particles. Because agglomeration reduces the spread of the material size distribution, it minimizes segregation whereby coarse particles roll down to the toe of the heap, leaving smaller particles at the top. Segregation is nonetheless ideal for upward gas flow, but unsuitable for parallel downward solution flow due to increased chances of ponding. The following example is an exception to this acknowledged benefit of agglomeration. Despite the fact that a very clayey ore (60%) containing kaolinite and montmorillonite had produced uniformly sized agglomerates of 0.3 to 1 cm in diameter using 5 kg/t (10 lb/ton) of lime and 4 kg/t (8 lb/ton) of cement, trenches digged out in a 9 m (30 ft) tall heap showed coarse, well-graded, and fine gradations after leaching (Kinard and Schweizer 1987). The segregation was attributed to the radial stacker. Agglomeration does not prevent the expansion of swelling clays upon contact with water, but will avoid the formation of zones impermeable to flow by distributing the clayey particles more evenly into the heap.

Agglomeration is also thought to lessen fines migration—a phenomenon apparently observed by many but poorly quantified. In this author's opinion, low solution rates applied to cyanide and copper sulfide heaps carry enough momentum to transport fines at the most 0.3 m (1 ft) below the surface, but not to the bottom of the heap. On the other hand, heavy rainfall, particularly in tropical climates, was shown to cause fines migration (Phifer 1988). The gradation observed in heaps that some have attributed to fine migration may have been the result of segregation caused by changes in ore properties and method of stacking.

The cross-section of a leached heap at the Alligator Ridge mine confirmed that surface fines do not migrate too far, only 8–10 cm (3–4 in) below the surface (Strachan and van Zyl 1987). The gradation was uniform everywhere else (Strachan and van Zyl 1987). Rainfall and sprinklers are usually to blame for damages (agglomerate disintegration, ore decrepitation) caused to the heap surface by the impact of water droplets. The impact of a stalled emitter on the heap surface should not be ignored.

A stalled wobbler still sprays solution, while a sprinkler emits a single stream that hits in a single spot.

Agglomeration helps create a more porous heap with better air and solution distribution. The bulk density is a reasonable indicator of porosity. Miller (2003) showed that the bulk density of a heap containing nonagglomerated clayey ore increased from 1.15–1.30 t/m^3 (71–81 lb/ft^3) at the surface to 2.0–2.1 t/m^3 (125–130 lb/ft^3) 4 m (13 ft) below. This was equivalent to a porosity of 50–55% at the surface to less than 30% 1 m (3.3 ft) below, approaching only 15–20% 6 m (20 ft) below the surface. At Candelaria, the bulk density decreased from 1.59 to 1.49 t/m^3 (99 to 93 lb/ft^3) after agglomerating an ore containing as little as 2.5% of -147 μm (100#) fines (Chamberlin 1980). The bulk density of a heap agglomerated with both 125 g/t (0.25 lb/ton) of Nalco 9760 and 2.5 kg/t (5 lb/ton) of cement was 3% lower than a heap agglomerated with cement alone (Pautler et al. 1990). There are no general guidelines for the optimum heap bulk density after agglomeration. The bulk density can be as low as 0.88 t/m^3 (55 lb/ft^3) at the Gooseberry Mine in Nevada (Butwell 1990) and as high as 1.67 t/m^3 (104 lb/ft^3) at the Masbate operation in the Philippines (Pizzaro et al. 1986). Agglomeration minimizes but not does eliminate slumping. At Little Bald Mountain, cement agglomeration reduced slumping from 24 to 8% (Tibbals 1987). Agglomerated copper sulfide heaps 4–8 m (13–26 ft) tall still slumped rapidly by about 12% (James and Lancaster 1998).

If neither agglomeration or desliming improve the heap permeability, ripping the heap surface or remining the entire heap (ore turnover by backhoe) are methods commonly employed for underperforming heaps. Blasting the leach pile has been suggested.

Remining has been pioneered by Girilambone Copper Company in Australia and is performed in several Chilean operations using hydraulic excavators. Remining may increase copper recovery by 2–10% after the same leach cycle.

Scheffel (2002) recommends ripping the heap surface two to four times in criss-cross direction and to rip the lower lift before stacking the next lift. Uhrie and Koons (2001) have shown that truck traffic areas should be ripped to 2.4 m (7.9 ft) before irrigation. This supports the recommendations of the Alligator Ridge Mine to use 3 m (10 ft) long shanks (DeMull and Womakc 1984). The severe consolidation of some heaps at the Alligator Ridge Mine even stalled a dozer equipped with shanks. At this mine, ripping temporarily increased the gold extraction rate by establishing new flow paths through the impermeable zones. However, only the zones actually disturbed by the dozer leached gold thoroughly, leaving higher-grade ore below. The mine finally chose to also rip immediately after stacking, rather than only after consolidation.

Ripping the surface of a run-of-mine heap, initially 3 m (10 ft) tall and only 2.1 m (7 ft) tall after irrigation and heavy rainfall, was not successful (Phifer 1988). The $-74\,\mu m$ (200#) fines, which accounted for 30–40% of the run-of-mine ore, had already migrated to the bottom of the heap. Ripping the surface was, thus, disturbing a couple of feet of the remaining coarse particles.

It ultimately is the proportion of connected pores filled with air that determines the efficiency of oxygen-based heap leaching systems, whether cyanide, thiosulfate, thiourea, or sulfidic heaps. Therefore, by producing a material of more constant size with agglomeration, there should be fewer contact points between the wetted surfaces, which, in turn, would reduce the stagnant moisture held up between agglomerates and increase the gas/liquid surfaces. At Little Bald Mountain, cement agglomeration did in fact reduce the moisture content of the heap (Tibbals 1987). In addition, the initial presence of moisture everywhere in the heap may contribute to the even spreading of the solution during wetting.

A more porous heap could sustain larger irrigation rates, which may decrease the leach cycle of certain heap leaching applications. Agglomeration may also lead to a more structurally stable heap, capable of bearing greater loads. An increase of the heap height directly translates into increased metal production. If the heap can support more than its own weight, one may opt for multiple lift stacking rather than dynamic (on/off) pads. Besides the criterion of structural stability, other factors, such as stackers and heat control in sulfide heaps, determine the heap height.

Used in conjunction with a binder, agglomeration can also increase the overall metal extraction of a heap stacked with material of smaller size than could have been stacked without agglomeration.

From the perspective of the leach chemistry, the greatest benefits of agglomeration are to reduce the travel time of reagents and increase the initial recovery rate. These benefits arise due to the faster contact between the mineral grains and the reagents introduced by the agglomeration solution. The Alligator Ridge Mine has observed a faster initial recovery (DeMull and Womakc 1984). Compared to nonagglomerated heaps, agglomerating gold ores with cyanide also reduced the overall cyanide consumption. However, adding more than 50 g NaCN/t of ore did not improve the gold extraction and increased the overall cyanide consumption (DeMull and Womakc 1984). However, in copper heap leaching, the better and faster contact of the reagents with the mineral surfaces also lead to undesirable chemical reactions between gangue minerals and sulfuric acid. To avoid such reactions, the rest period between stacking and irrigation should be minimized.

According to the sulfide bioheap model developed by the University of British Columbia, sulfide heap bioleaching environments should also benefit significantly from agglomerating the ore with the leaching solution (Bouffard 2003). Model simulations suggest that mixing an inoculum of mesophilic microorganisms (cell viability between 15 and 45°C) with the ore could eliminate the downward microbial colonization wave that would otherwise advance through the heap at a slower rate than the barren solution. With microbial preinoculation of the ore, the simultaneous growth of microorganisms everywhere throughout the heap could increase the leaching rate. According to the model, the more rapid colonization of the heap could occur in spite of the relatively few initial number of microorganisms added, a condition imposed by the appropriate moisture content of the ore (typically 5–15%) and the number of microorganisms in the inoculum (at most 10^9 cells/mL). Pre-inoculating the ore with a variety of temperature-sensitive microorganisms (mesophiles, moderate thermophiles, and extreme thermophiles) would further accelerate the oxidation of certain sulfide minerals, such as pyrite and sphalerite. In copper heap leaching, the effects on the microbial viability of simultaneously mixing the inoculum with the 5–45 kg concentrated sulfuric acid/t of ore are not well understood.

Agglomeration was a technical and economical breakthrough technology for heap leaching of clayey ore and ore containing high fines content. Figures 15 and 16 illustrate the most important performance indicator of the success of agglomeration. Up to 80% of the metal value of tailings could be extracted in 20–70 d. Gold or silver extraction from crushed ore was as high as 90% in sometimes as little as 10 d. Agglomeration at an Arizonian silver heap leach operation yielded incredible results. The extraction increased from 37% to 90%, while the leaching time dropped from 90 to only 7 d. Significant improvements were also obtained at a gold heap leaching operation in Nevada, where the gold extraction increased by 60% and the leaching time was reduced by half

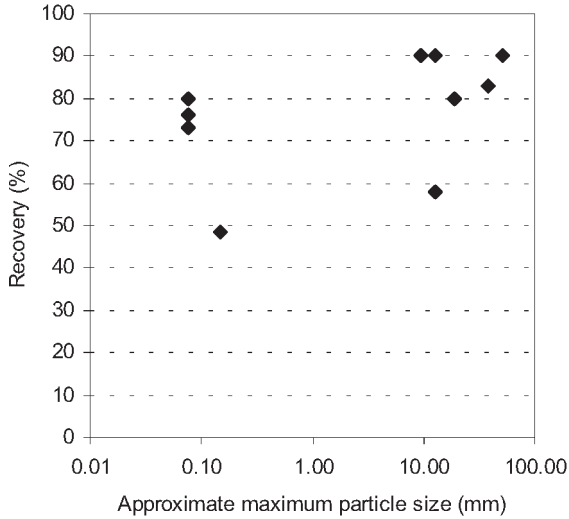

Figure 15. Influence of agglomeration of tailings (on the left of the X-axis) and crushed ore (on the right of the X-axis) on the precious metal recovery.

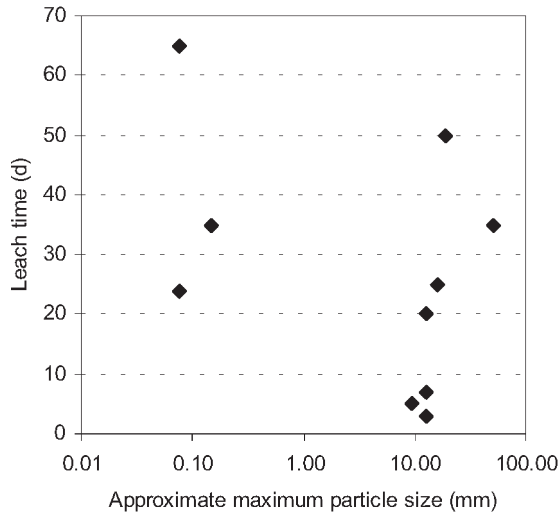

Figure 16. Influence of agglomeration of tailings (on the left of the X-axis) and crushed ore (on the right of the X-axis) on the leach time.

from 50 to 20–30 days. The Alligator Ridge Mine in Northeastern Nevada has well documented the implementation of agglomeration of crushed ore at its site (DeMull and Womakc 1984; Strachan and van Zyl 1987). Ten pads were tested for cement dosage, lime dosage, and stacking method. The combination of agglomeration at 50 g NaCN/t of solution and 1.5–5 kg lime/t of ore and a different stacking method (pushing the agglomerates up the slopes) increased the gold recovery to 70% and reduced the leach time from 60–90 d to 30–40 d. The heaps contained a reasonable amount of moisture (5–11%), no saturated zone, and leached uniformly.

From an environmental perspective, the better solution distribution in an agglomerated heap should also increase the recovery of the remainder of the soluble metal value during rinsing. It may also reduce the duration of the rinse cycle or the volume of wash water applied. Producing pellets that will remain intact long after leaching and rinsing may also reduce dust emissions.

CONCLUSIONS

This review article reported on advances in agglomeration in the heap leaching industry. To achieve the greatest benefits, the agglomeration process depends on the proper characterization of the material and on the appropriate selection and design of the agglomeration equipment. Such decisions affect, in turn, the design and operating conditions of the heap. However, as demonstrated at the Alligator Ridge Mine, the proper combination of crushing, agglomeration, stacking, and irrigation guarantees the success of heap leaching operations.

Agglomeration was a breakthrough technology for heap leach producers faced with ore of high fines or clay content. Achieving up to 80–90% precious metal recovery from ores at first thought to be heap-unleachable attests to the success of agglomeration.

The benefits of increased recovery and shorter leach cycles must be weighed, though, against the 5–10% extra capital costs incurred with agglomeration. With regard to the agglomeration operating costs, labor and energy costs evaluated at $US 0.10–0.30 per tonne of ore pale in comparison to the cost of the binder alone at roughly $US 1.00 per tonne of ore. The total agglomeration operating costs account for 10–20% of the total.

All operations reviewed that practiced agglomeration used a binder. Precious metal heap leaching operations prefer cement in an amount from 2.5 to 10 kg cement/t of ore (5–20 lb/tonne) added to a cyanide solution containing typically less than 300 ppm NaCN. Copper heap leaching operations agglomerate with typically 20 kg sulfuric acid/t of ore (80 lb/tonne) and 80 kg water/t of ore.

To reduce the binder costs, a combination of two or more binders could be envisaged. The performance of polymeric binders should also become more predictable, as they currently are one of two binders suitable for copper ores. A greater selection of inexpensive binders, tolerant of low pH's and resistant to microbial attacks, should be developed for the copper heap leaching industry.

Most heap leaching operations, including some of the largest heap leach producers, have chosen mobile or spreader conveyors for stacking and drums for agglomeration. The residence time in the drum and the amount of moisture added determine the agglomerate size and size distribution, but not necessarily the agglomerate strength. A residence time of less than 60 s in industrial drums does not live up to recommendations from laboratory trials. An empirical equation that includes the drum diameter, drum length, and rotation speed was proposed to predict drum throughputs of less than 5,000 tonne per day.

A nonrecognized disadvantage of using drum agglomerators is the production of large agglomerates or not sufficiently porous agglomerates. The influence of the pore length and tortuosity inside an agglomerate on the diffusion rate has been examined up until now from a theoretical standpoint only (Bouffard 2003). Future trials should attempt to define the optimum agglomerate structure.

Although not representative of the unsaturated conditions in a heap, laboratory dipping, flooding, or compaction tests can help compare the performance of the binders on a quantitative basis and optimize the binder requirements. In this author's opinion, none of these methods, including even the full height columns or silos can accurately predict the expected performance of agglomerates in an industrial-scale setting. Test pads are undeniably necessary for scaling-up production. Online methods for controlling the moisture and size of agglomerates are now available industrially. Nevertheless, the criteria of size and moisture content, as well as an increased in metal recovery, do not directly relate to the true benefits of agglomeration, i.e., increased porosity and uniform flows. What are needed are methods for measuring the strength of agglomerates. Such methods exist for agglomerates made up of fines, i.e., particles of the same size, but none is available for agglomerates comprised of particles of very different sizes. What are also needed are methods for visualizing the movement and possible disintegration of agglomerates in a heap and for measuring the moisture retention, air pore space, and pore connectivity in the heap. Existing methods for measuring the moisture content, such as electrical resistivity tomography, are not properly calibrated in the field and current methods for measuring pore space lack scalability. Hence, other than the financial benefits of agglomeration on the bottom line, opportunities still exist to quantify the real physical impact of agglomeration to ultimately make this process more controllable, predictable, and robust.

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