Physicochem. Probl. Miner. Process. 53(1), 2017, 443−464

www.minproc.pwr.wroc.pl/journal

Physicochemical Problems of Mineral Processing

ISSN 1643-1049 (print)
ISSN 2084-4735 (online)

Received January 21, 2016; reviewed; accepted October 20, 2016

FULL SCALE SYSTEMATIC OPTIMIZATION STEPS FOR A HEAP LEACH CRUSHING PLANT

E. Tugcan TUZCU

DAMA Engineering, PLC Ankaralilar Cad. Azatbey Sit.498 Sok. No:17 Cayyolu, Ankara, TURKEY, tugcan.tuzcu@gmail.com, t.tuzcu@dama-engineering.com

Abstract: This paper represents a plant optimization (capacity improvement) study for a gold heap leach plant to obtain the product size of 100% of -6 mm crushed ore at a capacity of 180 Mg/h. The former plant flowsheet was re-designed to obtain the target product size and capacity by modeling and simulation. The plant measurements and ore test work were carried out as the first part of the design study. During the simulation phase, the model predictions (estimations) were calibrated based on the measurements and ore test work while developing the flowsheet. A flexible layout design and site construction were done after the flowsheet re-design by using available plant footprint as much as possible and providing production infrastructure by adding intermediate stocks. Finally, after 8 months of intensive work, the plant achieved 100% of -6 mm crushing product with the target size of 80% of -4.55 mm at a capacity of 200 (±10%) Mg/h.

Keywords: *four-stage crushing, mechanical improvement, modeling-simulation, optimization, flexible layout design, heap leach*

Introduction

Gold extraction technologies of our era have the roots for centuries. Well known methods such as gravity concentration, amalgamation, cyanide leaching, zinc precipitation, and carbon adsorption are the base for todays' technology and still used (Marsden and House, 2009). The recovery method (process selection) for gold truly depends on many factors such as geological, mineralogical, metallurgical, and economic. Comminution, classification, and solid-liquid separation are the unit operations those are utilized in any of the mentioned recovery methods. Heap leach is the well-known method for generally low grade gold ore treatment due to its relatively low capital and operating costs. Crushing the ore to an allowable upper limit is the only size reduction step. A good rule of thumb is to avoid crushing if at all possible. If necessary to select the crushing system with a proven capability (Kappes, 1979). However, crushing the ore to an experimentally determined size and agglomerating

http://dx.doi.org/10.5277/ppmp170135

help better recovery. Poor percolation may lead to low recovery for the heap leach operation (Kappes, 2005). Size reduction cost of an ore is the major cost item as operating costs. Annual based, comminution of gold ore requires more energy now than it used to be regarding the increased gold production in the last decades. Comminuting Au and Cu ores consume 0.2% of electricity globally (Ballantyne and Powell, 2014). A portion of the energy belongs to heap leach operations. Crushing energy and consequently crushing operating cost constitutes 18% of total operating cost of heap leaching (Dhawan et al., 2013). The Gold Deposit is located in the Azerbaijan. The plant processes an oxide gold ore. Gold mineralization at the area is predominantly concentrated in a silicified polymictic breccia body representing a typical high-sulfidation system characterized by pervasive, massive silicification, alunitisation, kaolinisation, and brecciation. The production rate of the plant is expected to be 980000 Mg/y with 4 stage of crushing. An American Engineering company carried out a proper design (KCA, 2011) for the plant, however plant could not follow what was suggested in the design. The design specifications for the production are given below.

Table 1. Summary of design production specifications

Production Schedule		
Daily working hours	16.7	hours/day
Weekly working days	7	days/week
Annual working weeks	52	weeks/year
Annual working days	327	days/year
Shift Availability	70%	
Production Rate	3000	Mg/d
Nominal	180	Mg/h
Design	200	Mg/h
Target Crush Size	100% - 6	mm
Target Crush Size	80% - 4.65	mm
Leach size for the best recovery	100% - 3.35	mm

The clay content of ore body and high crushing index of ore were the main problems. The blinding problem of the screens due to clay content of ore body was used to reduce effective area of the screens. Hence, it used to cause to the increase circulating load to the crushers as the time proceeds starting from the first run. Thus, the system could not reach the steady state flow pattern. Capacities of the equipment (crushers, bins, screens etc.) were exceeded. In addition to the problem mentioned, the current configuration of the plant flowsheet was not suitable for the desired throughput that compensates high crushability index of the ore. Due to several reasons such as ore conditions, blinding of the screens, and overflowing of the silos due to unsteady state of operating conditions, the plant was used to work at: 30-40 Mg/h or 700 Mg/d (under wet-though conditions) and 80-100 Mg/h or 1500 Mg/d (under dry-easy conditions). Therefore, the capacity has not been reached.

The aim of the study was to investigate the measures those ramp-up the capacity to the design level and apply these measures on site. This paper is explaining the road map for the plant optimization after the proper design and starting from procurement period. First and second steps in the study are shown in Figs. 1 and 2.

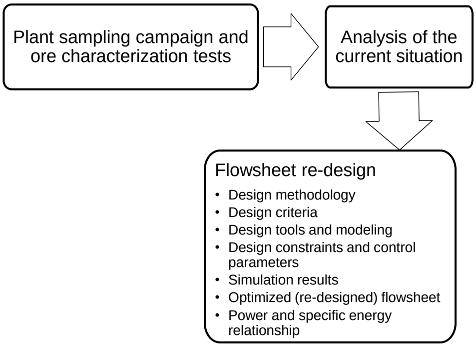

Fig. 1. Heap leach plant experimental and theoretical part road map

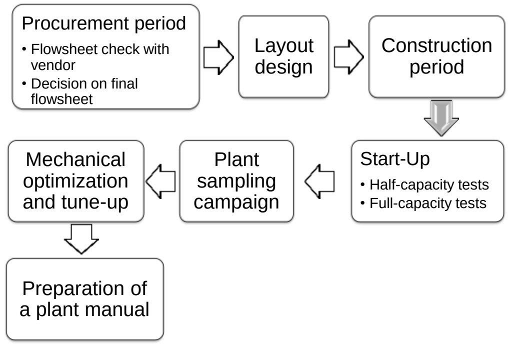

Fig. 2. Heap leach plant construction, start-up, process optimization part road map

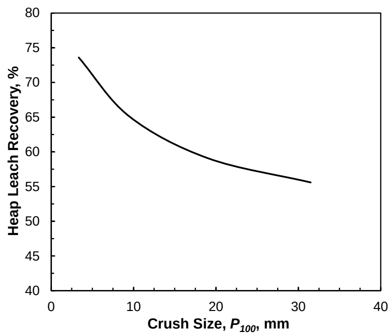

Fig. 3. Heap leach recovery vs. crush size

At the end of the simulations, 80% of -4.65 mm crushing product target with 55.4% -3.35 mm was achieved with the capacity around 200 (±10%) Mg/h. Particles having 3.35 mm was the leach size for the best recovery. Figure 3 shows the heap leach recovery versus crush size for the process. However, obtaining 100% -3.35 mm by conventional crushing is not practical. Reducing the size down to 3.35 mm was too fine for the crushing and some type of grinding was required for this aim. There was a twilight zone in which the product was 3.4 mm, 2.0 mm or 1.4 mm limiting size which was either the crushing or grinding (Myers and Lewis, 1946). The study was aimed to adapt four-stage crushing to obtain the product. As the size goes finer for crushing the flowsheet design gets more challenging. Since there are more options in terms of equipment, and their installation and maintenance requirements (Boyd, 2002) for fine crushing than coarse crushing, the design must be balanced economically and technically. The study represents a plant ramp-up and optimization study that has been carried out by the Engineering Company for an Azerbaijani client. The plant was aimed in 8 months study to bring into the design production rate of 3000 Mg/d by eliminating major construction application related process problems.

Flexible and optimum layout design for an existing plant has major drawbacks which are meant to be constraints for an optimization problem. Some researchers targeted to achieve optimal plant layout designs by minimizing the cost associated to potential domino effects (Lira-Flores et al., 2014). This is the formulation to achieve optimal facility layouts taking into account the main variables affecting an index called Domino Hazard Index. Layout footprint is affected by flowsheet developed for the particular plant. If a flowsheet was designed with non-representative design criteria and assumptions, the mass flows would be unrealistic therefore is the circulating loads. Minimizing circulating load and taking this into its account for the layout was given in a study (Lotter et al., 2013). Therefore, the realistic flowsheet and layout design is possible by giving realistic design criteria (specification) to the calculator (simulator). An iterative methodology was developed for treatment of design specifications within the simulation of complex solid processes (Reimers et al., 2009). The flowsheet design is the first and most important step for layout design. It may require a multidisciplinary approach of sampling, geology, quantitative and qualitative mineralogy, applied statistics and mineral processing sometimes (Lotter, 2011). This may aid for better layout design. The processing plant layout design is also a function of early ore test works. A diagnostic approach which use standard tests and optimization procedures for the particular ore is explained in a gold plant design study (Torres, 1999). As explained so far, the flexible layout design and optimization is achieved via optimization of flowsheet, test works, and mass balance calculations. All of these methods either use pure technical efficiency as a criterion for optimum setting of a mineral separation (Jowett and Sutherland, 1985) or also the economic criteria (Yingling, 1990) in addition to the technical criteria. In this study both were considered.

Experimental

The purpose of the campaign was to know more about the ore coming to the plant and the response of the installed equipment to the ore. There are several technological and metallurgical tests carried out for the ore. However, the crushing properties of the ore were investigated in the study. A plant sampling campaign was done. The initial sampling campaign was aimed at the ore crushing characterization, plant feed, and plant product sizes analyses. Several points in the plant were selected and sampled. These initial points were jaw crusher feed, jaw crusher product, screen feeds and products, and the rest of the crushers' feed and product streams. The summary of initial sampling work for the plant that was ramp-up: plant feed rate was around 100 Mg/h at best (never runs continuously for 1 hour during bad weather conditions and cannot reach steady state conditions), jaw crusher F_{100} (plant feed top size) was around 750 mm (based on hand measurements), jaw crusher F_{80} (corresponding particle size value at 80% passing size distribution; 80% of the plant feed) was around 550 mm (based on hand measurements), final product was around 7 mm at 100% passing.

The hand measurements were carried out by a rope by determining the circumferences of the sampled rock particles by making the assumption that the volume was similar to that of a sphere. Then, the equivalent spherical volume gave the diameter. This was rough assumption but for primary size distribution there were not too many options including some software.

One of the most important starting stages of the modeling study was to choose a proper feed size distribution to the circuit. Because of the process nature, jaw feed could not be measured by screening but the product. Once feed was observed, the hand measured and the product of the jaw crusher was measured, a plant feed size distribution was re-created that obeys Gates-Gaudin-Schuhmann (*GGS*) truncated distribution. By creating the size distribution it was aimed to obtain F_{100} 750 mm and F_{80} around 560-600 mm. Equation 1 gives the *GGS* size distribution created with a distribution modulus of 0.68, where xi is the particle size class, and FSD is the feed size distribution.

$$FSD = (\frac{xi}{750})^{0.68} \tag{1}$$

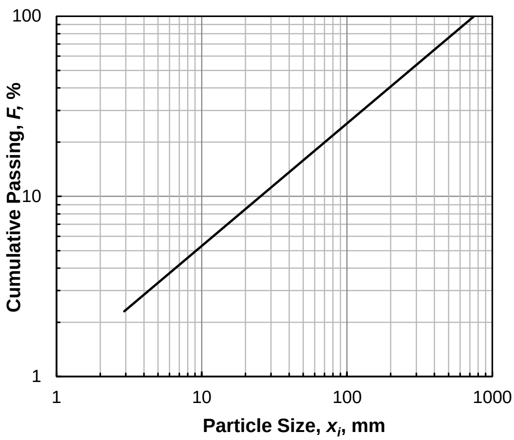

Fig. 4. Reconstructed *GGS* feed size distribution

In the same way, all streams were sampled and size measured. Some streams could be sampled only twice, some three to five times. Due to the nature of the plant feed material and equipment instantaneous response to the loading conditions, there are measurements having different shapes even though they are taken from the same stream. Figure 5 shows some of the plant feed size distributions.

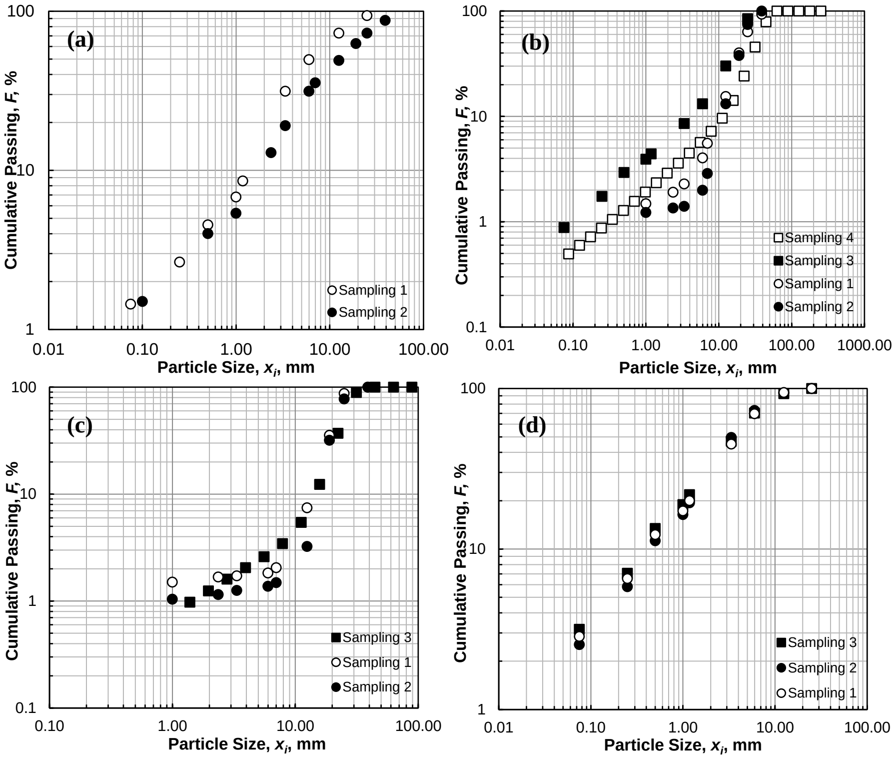

Fig. 5. Sampling points size distributions: (a) grizzly undersize, (b) secondary crusher product, (c) tertiary crusher product, (d) quaternary crusher product

Table 2 summarizes ore physical properties based on the ore test work. As seen from the crushing work index W_{ic} and abrasion index A_i, the ore was very though to crush and not liner friendly.

Table 2. Ore crushing properties

Item	Unit	Value
W_{ic}	kWh/Mg	18
A_i		0.9
Density	g/cm^3	2.6
Ore moisture	% of ore tonnage	3
Clay content	%	3-4

Methods

Analysis of the previous plant flowsheet

The previous plant used to run around 100 Mg/h capacity at maximum. That was far below the desired (designed) throughput. A jaw crusher discharges to an open circuit secondary crusher which feeds a closed circuit tertiary crusher. Output of this circuit (undersize of screen) goes to the final banana screen whose over and mid-size feed the quaternary crushers and undersize is taken as the final product of the crushing circuit. An American engineering company first offered a 4-stage crushing to solve the problems in their scoping study (KCA, 2011). The company however used the tertiary crusher instead of secondary crusher. Even if the plant used to seem as 4-stage crushing plant, the equipment configuration was not suitable for the purpose. The flowsheet of the previous plant is given in Fig. 6.

The main equipment list for the plant is given Table 3.

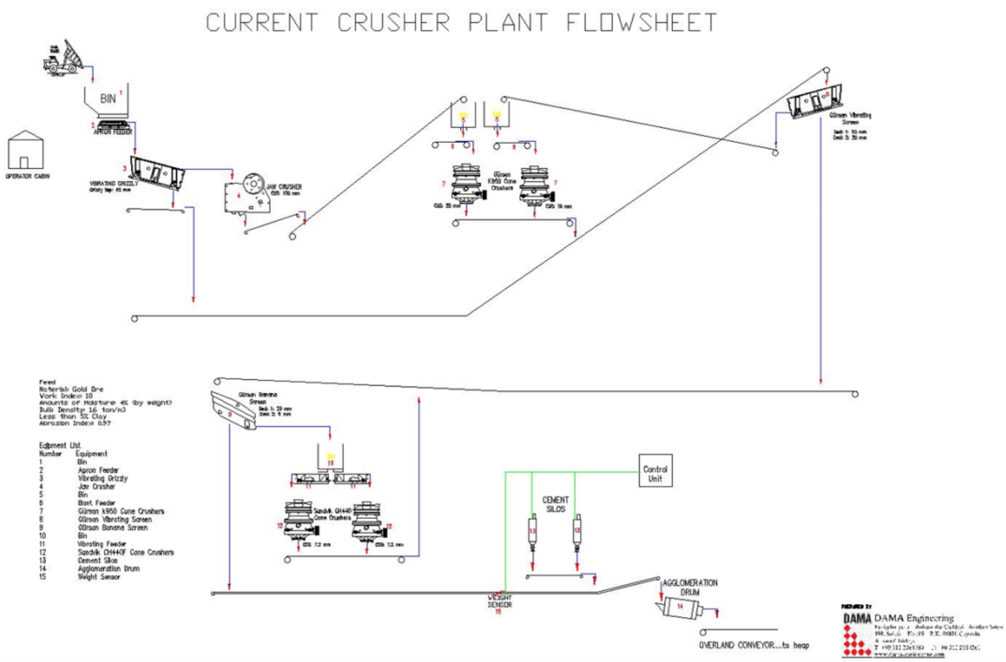

Fig. 6. Plant flowsheet before our work

Flowsheet re-design

The flowsheet of the previous plant was re-designed and optimized via modeling. The data collected during the plant sampling and the ore crushing characterization tests supported design work via modeling.

Table 3. Main equipment list of the previous plant (2012)

Equipment	Quantity	Size (m×m)	Power (kW)	Aperture size or close side setting (*CSS*)
Grizzly	1			65
Jaw crusher	1		132	100
Coarse cone crusher	1		160	25
Coarse cone crusher	1		160	19
Fine cone crusher	2		220	6
Banana screen	1	2.4×7		50, 20, 8 (top, middle, bottom decks)
Vibrating screen	1	2.4×6		20, 6 (top, bottom decks)
Agglomerator	1	2.5×8	110	

Design methodology

The re-design methodology of the process is given in Fig. 7 (Tuzcu, 2013). The plant data was collected, and the lab scale experiments for ore characterization were completed. The data obtained from the plant, and lab test were analyzed and used in modelling and simulation tools in a loop such that until the desired value was obtained. The calculations and estimations were continued obeying the constraints (flowrate, power, reductions ratio etc.). The most plausible scenario was decided and tried in the plant environment to see the response later on. The main aim of the study was to create a cost effective scenario that used the current equipment park as much as possible and current layout in a most efficient way. This reduces the *CAPEX* (capital expenditure) of the work.

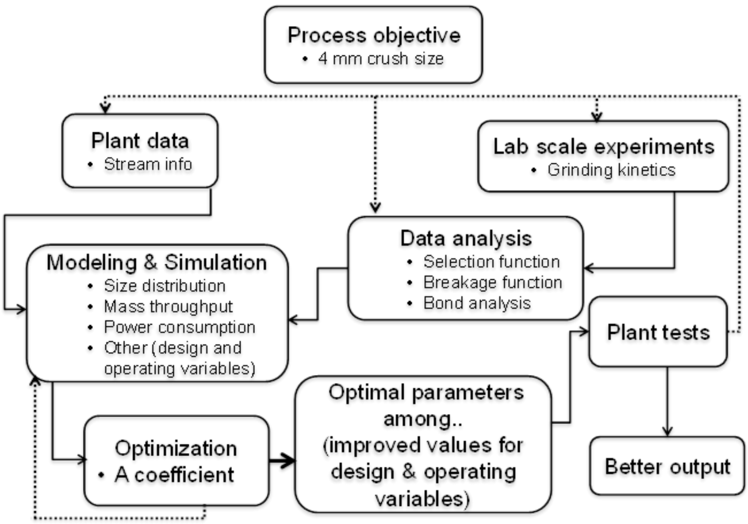

Fig. 7. Process of re-design of the flowsheet (Tuzcu, 2013)

Design criteria

Deciding on the design criteria is the most important step in any flowsheet design. Some of these parameters come from customer, some from experiments, some from engineering company, and some from the location where the plant is located. Table 4 summarizes the design criteria for this study.

Table 4. Design criteria

Item	Unit	Value
W_{ic}	kWh/Mg	18
A_i		0.98
Feed rate	Mg/h	200
Density	Mg/m^3	2.62
$F_{100\text{-}plant}$ (plant feed top size)	mm	750
$P_{100\text{-}plant}$ (plant product top size)	mm	6
Working time	hours per day	16.7
	days per year	327
Production	Mg/h	180
	Mg/d	3000
	Mg/y	981002
Design capacity	Mg/h	200

Design tools and modeling

Different simulators were used against hand calculations for the comparison purposes. Estimating circulating load itself for such though ore with 4 stage crushing is an important part of design in determining correct capacities of the main equipment and as well as the belt conveyors. The simulators used in the study were: Bruno by METSO, MODSIM PRO by MTI (King, 1990; King, 2001), and USIMPAC by Caspeo.

There are several models that could be used in the modeling work. Table 5 indicates models used in our study.

Table 5. Equipment models used in the flowsheet re-design

	BRUNO	USIMPAC	MODSIM
Jaw crusher	-	model 131 - crusher from database	JAW 1
Cone crusher	GP100S, GP11F	model 106 (1) - cone crusher (2)	SHHD
Cone crusher	HP200sh	model 131 - crusher from database	CRSH
Single deck screen	CVP1845	-	SCRN, DCS1
Double deck screen	A132L	model 148 (1) - screen double deck (1B)	CSCR
Triple deck screen	-	model 149 (1) - screen triple deck (1B)	-

There is no access to the embedded BRUNO models which is given as the equipment names in Table 5.

The models from USIMPAC are categorized either level 1 or level 2. Level 1 model requires very few details and experimental data. However, the models of level 2 or higher are much more accurate than the level 1 models as they require special laboratory tests for the sample. In this study both level 1 and 2 were used as the data in hand allowed doing so. The crusher models given in the Table 5 use Whiten's (1973) classification function (Eq. 2):

$$C(X) = 1 - \left[\frac{x-k_2}{k_1-k_2}\right]^{k_3} \tag{2}$$

where x is the particle size, k_1, k_2 and k_3 are the model parameters some of which are related to the closed side setting. The model used in lieu of breakage function is standard breakage function with Whiten's interpretation (Eq. 3):

$$P(x,y) = (1 + Ky^{m-q})\left(\frac{x}{y}\right)^n + (Ky^{m-q})\left(\frac{x}{y}\right)^m \tag{3}$$

where $P(x,y)$ is the fraction of the material smaller than size x produced from a particle of size y, K is the constant, n, m and q are the model parameters. Bond's or Magdalinovic's formulas are used to predict power for the crushing events in USIMPAC models given in the Table 5. The crusher models used in MODSIM also utilize the same formulas in lieu of classification, breakage function and power prediction.

Design constraints and control parameters

Simulation is a tool that you can create a number of cases arbitrarily. Depending on the equipment or processing environment there are physical constraints which limit the estimated values. These parameters may be roping point, cyclone diameter or apex nozzle in a classification simulation (Delgadillo et al., 2008), whereas they may be bubble diameter, cell volume or floatability component parameter in a flotation simulation (Runge et al., 1998). The simulation constraints in our study were: total specific energy for crushing (power demand for crushing only, excluding no-load power) is allowed to be maximum around 2 kwh/Mg, installed power values of current crushers those need to be intended to re-use, screen surface areas (dimensions) of current screens those need to be intended to re-use, maximum circulating load is allowed to be 300%, layout constraints effecting belt angles, chute volumes.

Based on the above parameters, the number and the type of the simulations were decided (restricted). Finally, there were 13 cases run for the design purposes (Table 6).

Table 6. Final cases run for simulations (*SD*= single deck, *DD*= double deck, *TD*= triple deck)

	Unit	Case1 (base)	Case 2	Case 3	Case 4	Case 5	Case 6	Case 7	Case 8	Case 9	Case 10	Case 11	Case 12	Case 13
# Jaw crushers		1	1	1	1	1	1	1	1	1	1	1	1	1
# Secondary crushers		1	1	1	1	1	1	1	1	1	1	1	1	1
# Tertiary crushers		1	1	1	1	1	2	2	1	2	2	2	2	2
# Quaternary crushers		2	2	2	2	2	2	2	3	3	3	3	2	2
W_{ic}	kwh/Mg	18	18	18	18	18	18	18	18	18	18	18	18	18
A_i		0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98	0.98
Feed rate	Mg/h	180	200	200	200	200	200	200	200	200	200	200	200	200
$F_{100\text{-}plant}$	mm	750	750	750	750	750	750	750	750	750	750	750	750	750
Jaw *CSS*	mm	150	150	150	150	150	150	120	125	150	150	150	150	130
Sec. cone *CSS*	mm	40	40	40	40	40	40	30	25	40	35	35	30	30
Ter. cone *CSS*	mm	19	15	8	13	13	8	8	10	8	12	12	12	12
Qua. cone *CSS*	mm	6	6	6	6	6	6	6	6	6	5	5	6.3	6
Grizzly	mm	40	40	40	40	40	40	40	40	40	40	40	40	30
SD Screen 1		-	-	-	-	-	-	-	-	-	-	12	-	-
SD Screen 2	mm		6	6	6	6	6	6	6	6	6	6	6	6
**DD* Screen 1	mm	50,20*	50,15	40,12	25,14	25,14	40,12	40,8	40,10	40,12	40,12	35,6	60,30	30,10
***DD* Screen 2		-	-	-	-	-	-	-	-	-	-		18,10	-
TD Screen	mm	50,20,6**	-	-	-	-	-	-	-	-	-		-	-

**top, bottom decks,* ***top, middle, bottom decks*

There are parameters which need to be followed to track the outcome or performance of the simulation. However, there are a few systematic works published specifically for crushing modeling (Lynch, 1977; King, 1990; McKee and Napier-Munn, 1990) which explains how to follow simulation performance by tracking control parameters and obeying the constraints. Considering the complex nature of the process, one cannot track all design and operating variables at each simulation to come up with a conclusion. Instead of observing too many parameters, tracking a single coefficient (parameter) which is the function of process variables and tells about the performance of the simulation is manageable (Tuzcu, 2013). If the study doesn't allow tracking just one variable, it is meaningful to track representative parameters for the particular modeling. In this crushing design study, 5 parameters were tracked for the final decision on the simulations: total crushing power ($\sum_{i=1}^{4} Power_{CR_i}$, kW), plant final product ($P_{80\text{- }plant}$), $P_{3.35mm}$ in plant final product, total reduction ratio ($F_{80\text{- }plant}$ / $P_{80\text{- }plant}$), specific energy of crushers (($\sum_{i=1}^{4} \frac{Power_{CR_i}}{Tonnage_{CR_i}}$), kWh/Mg).

Results

Table 7 shows the parameter values tracked through the simulations. One of the most important parameters is obviously the (specific) energy consumption together with the desired reduction ratio or the amount of desired fines in the product. From this standpoint, Cases 12 and 13 were favorable over the rest. Case 12, however, exceeded the attached power constraint (constraint 2) of the current secondary crusher that was intended to re-use. Case 13 was the remaining as the last alternative or the case optimized before going to vendor.

Table 7. Control parameters in simulations

		Cases												
	Unit	1	2	3	4	5	6	7	8	9	10	11	12	13
Total power only for crushing	kW	300	369	366	369	370	345	365	355	367	445	336	351	352
Final product (P_{80})	mm	5.99	4.63	4.66	4.64	4.64	4.65	4.68	4.66	4.65	4.48	4.50	4.65	4.65
Percent passing @ 3.35 mm	%	50.0	55.7	55.3	55.6	55.6	55.3	55.0	55.3	55.3	57.9	57.7	55.4	55.4
Total reduction ratio		95	122	122	122	122	122	121	122	122	127	126	122	121
*Specific energy	kWh/Mg	2.05	1.74	2.03	1.80	1.81	1.91	1.71	1.77	2.03	2.14	1.67	1.39	1.55

*Specific energies are based on the tonnage processed through the particular equipment

Optimized (re-designed) flowsheet

Four stages crushing was proposed (Fig. 8) to reduce the size from 750 mm to 100% passing of 6 mm. A jaw crusher with 130 mm *CSS* (close side setting) was used as primary, open circuit cone with 30 mm *CSS* was used as secondary, closed circuit cones with 12 mm *CSS* were used as tertiary, and closed circuit cones with 6 mm *CSS* were used as quaternary crushers. Underflow of secondary screening was stocked in 800 m^3 fine ore silo. A double deck screen with 30 mm top deck and 10 mm bottom deck worked in closed circuit with tertiary crushers and a single deck screen with 6.5 mm deck worked in closed circuit with 6 mm quaternary crushers were used. Finally, the material stocked in the silo was fed to the agglomeration drum. The main equipment settings' comparison for the plant in 2012 (previous), 2013 (starting) and after modeling is given in Table 8.

The flowsheet was optimized with the settings (Table 8) for the equipment. After this stage, the project went to vendor for selecting the industrially available and suitable equipment. Then, the crusher supplier was asked to evaluate the circuit and propose the equipment among the industrially available ones (Tuzcu and Vural, 2014). After mutual simulations and agreement, particular setting values were adjusted to vendor's particular machines. These settings better fitted to the equipment's working principles. Our and vendor settings' comparison are listed in Table 9.

Table 8. Previous and optimized case settings

	Unit	*Previous Plant 2012	**Base Case 2013 Case 1	Optimized Case Case 13
F_{100}-R.O.M. Ore	mm	600	750	750
Jaw Crusher *CSS*	mm	100	150	130
Secondary Cone Crusher *CSS*	mm	25	40	30
Tertiary Cone Crusher *CSS*	mm	19	19	12
Quaternary Cone Crusher *CSS*	mm	6	6	6.3
Grizzly	mm	65	40	30
Screen Double Deck 1 (top, bottom decks)	mm	50,20	50,20	30,10
Screen Single Deck 1		-	-	6
Screen Triple Deck (top, middle, bottom decks)	mm	50,20,6	50,20,6	-

**Previous Plant and Base Case 2013 include 1 secondary, 1 tertiary, 2 quaternary*
***Optimized Case includes 1 secondary, 2 tertiary, 2 quaternary*

Table 9. Vendor configuration based on our design

	Unit	Our Engineering Company	Vendor
Jaw Crusher *CSS*	mm	110-130	130
Secondary Cone Crusher *CSS*	mm	30	37
Tertiary Cone Crusher *CSS*	mm	12	12
Quaternary Cone Crusher *CSS*	mm	6.3	6.9
Grizzly	mm	30	30
Double Deck Screens Aperture (top, bottom decks)	mm	30,10	20,10
Single Deck Screens Aperture	mm	6	6

The optimized flowsheet was finalized by us with the input from vendor based on what is available industrially. The final main equipment list with the main specifications is given in Table 10.

Table 10. Main equipment list for the optimized flowsheet

Equipment	Specification
Old Jaw Crusher	132 kW
Old Secondary Crusher	132 kW
Old Screen I	14.4 m^2
Old Screen II	14.4 m^2
New Tertiary Crusher 1	220 kw, Fine Concave
New Tertiary Crusher 2	220 kw, Fine Concave
Old Quaternary Crusher I	200 kw, Extra Fine Concave
Old Quaternary Crusher II	200 kw, Extra Fine Concave
New screen 1	12.6 m^2
New screen 2	12.6 m^2

Layout design

According to the optimized flowsheet, equipment, and structures were placed in the layout plan. The drawings of the new crushers were obtained from the manufacturer to use in the layout design. In the same way, conveyors belts were re-arranged and placed on the layout. We had difficulty in the re-arrangement of the plant layout design due to restricted plant area between primary crusher and agglomerator. The main goals of the layout design were to design a compact plant as possible and increase the plant availability. Minimizing the footprint for the new and modified equipment and structures reduces the project CAPEX. An increasing the plant availability by putting a fine ore silo or intermediate silos increases the production return. One of the most important constraints in such a restricted layout for a crushing plant is the slope of the belt conveyors. Belt slopes shouldn't exceed belt angle of repose of particular size material for certain rheology. Most of the time for regular belt conveyors this angle shouldn't exceed 15-16^o (Fruchtbaum, 1988). In addition to this, if belt slope goes up the steel and structural requirement will be high. So is the CAPEX. Optimum and flexible layout design is obviously one of the most important parts of the project. Figure 9 shows the previous and modified layout.

Power and specific energy relationships

As seen from Table 11 and Fig. 10, the crushing power demand was three-fold less than installed power for primary crusher, 1.**5-**fold for secondary, three-fold for tertiary and five-fold for quaternary crusher. This is process inefficiency due to crushing size and throughput. We would have chosen the similar capacity primary crusher if the primary crushing power demand would be 60 kW. In the same way, we would have chosen the similar capacity tertiary crusher if tertiary crushing power demand would be 150 kW.

Table 11. Three types of power designation

Crushing stage	Crushing Power Demand kW	No-load power kW	*Power Draw kW	**Installed Power kW
1	44	40	84	132
2	95	50	145	160
3 (×2)	137	150	287	440
4 (×2)	67	170	237	400
Total power, kW	342	410	752	1132
Specific energy, kWh/Mg	1.5		3.41	5.13

**estimated based on the manufacturers data*

***power of proposed (industrially available) equipment. However at this stage, we did not give any vendor or brand*

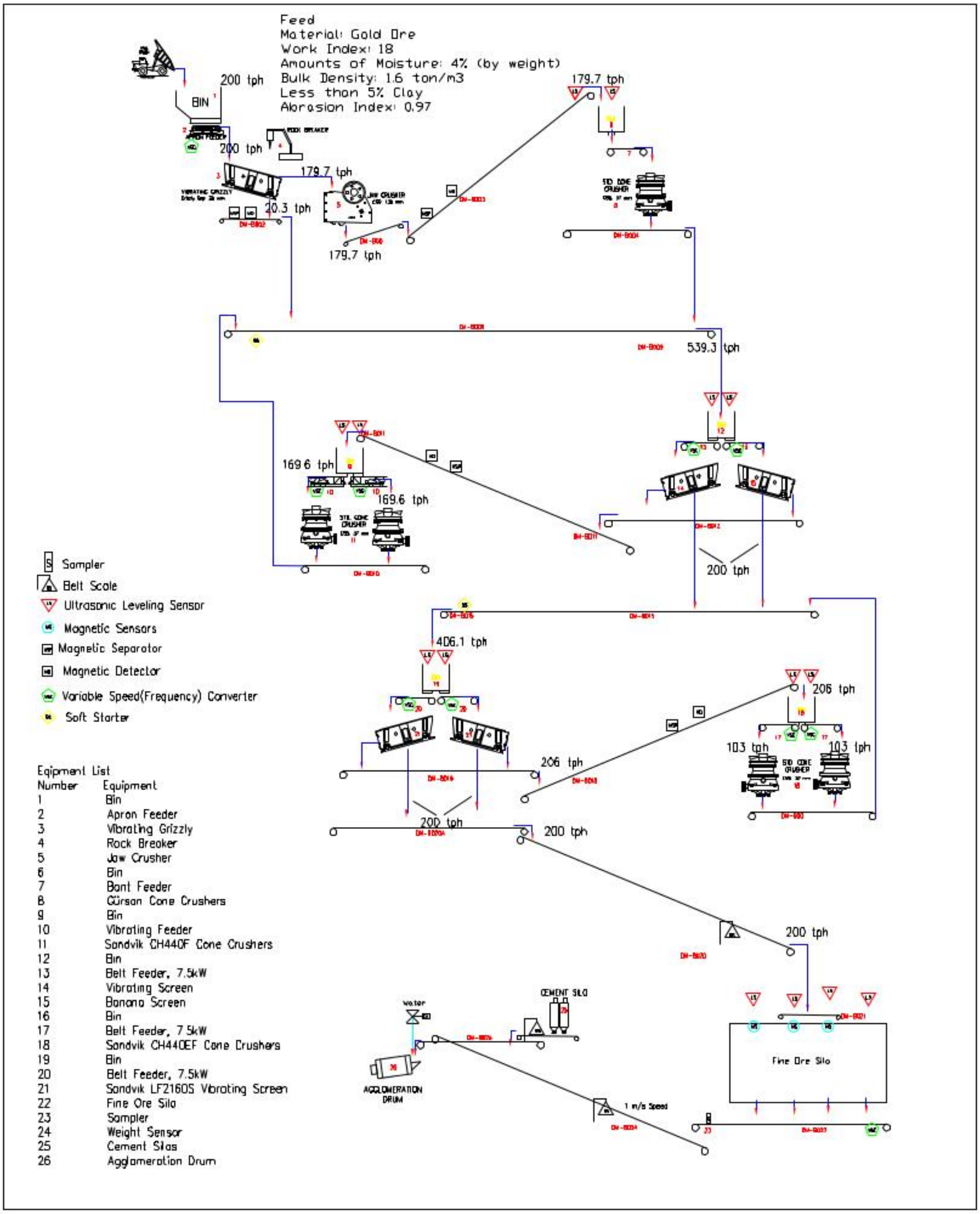

Fig. 8. Flowsheet developed after re-design

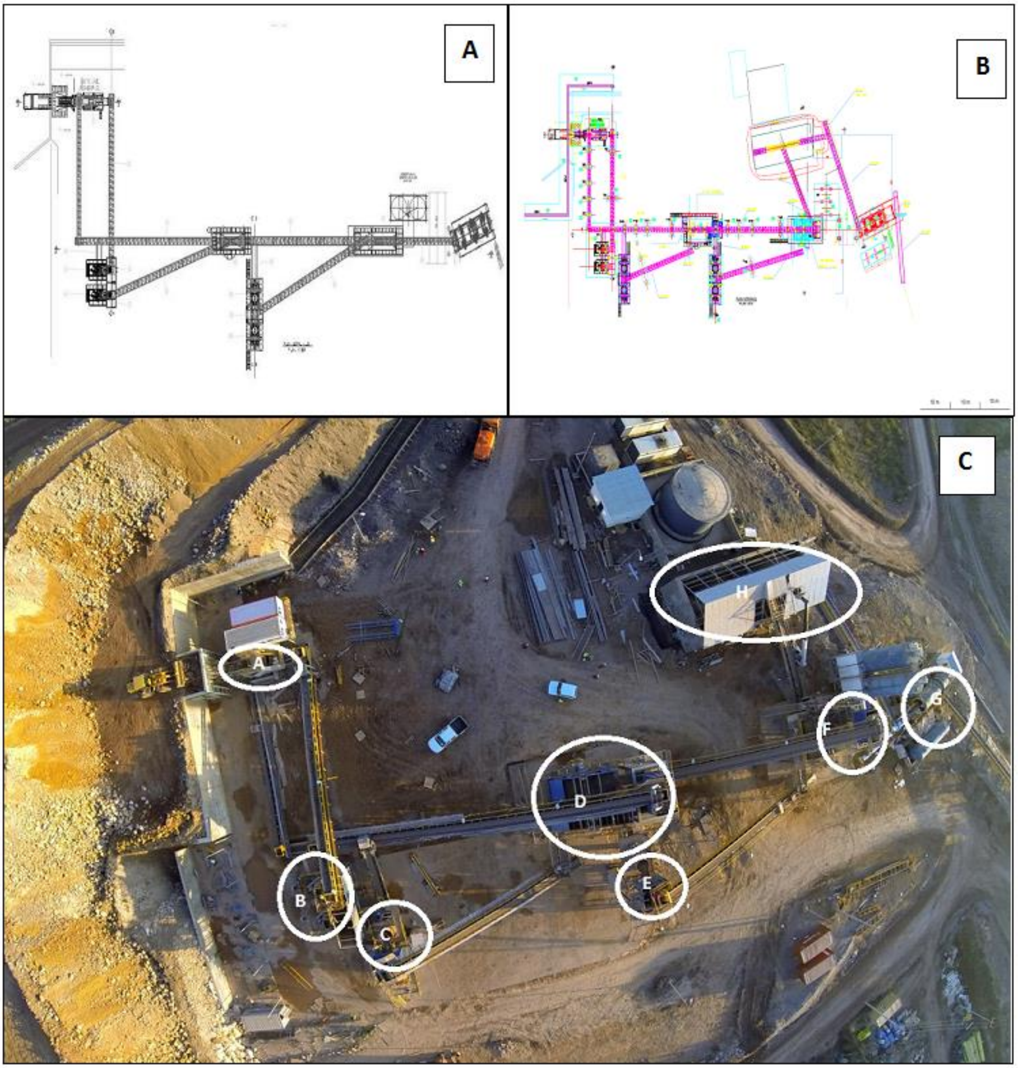

Fig. 9. Plant layout (A – before capacity increase and optimization work, B – after capacity increase and optimization work, C – (A) renewed primary crushing; (B) secondary screening and crushing; (C) added/renewed tertiary crushing; (D) renewed tertiary screening; (E) added/renewed quaternary crushing; (F) added/renewed quaternary screening; (G) renewed agglomeration section; (C) added fine ore silo)

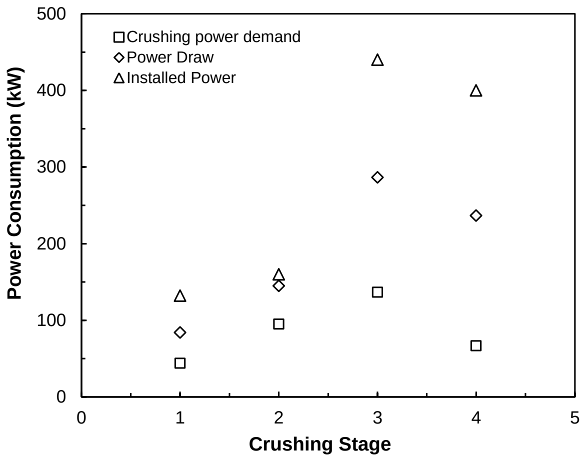

Fig. 10. Three types of power designation

However, the situation was little different for the quaternary crushing. Fourth stage crushing could achieve very low reduction ratios (here 1.3) (Table 12). The feed size was around 9 mm, *CSS* of fourth stage crushing was around 6 mm, and the product size was around 7 mm. Even though the power demand for the quaternary crushing was 34 kW/crusher, one crusher could handle only around 100 Mg/h capacity (for 6 *CSS*), which meant to select two quaternary cone crushers to achieve this duty. Including no-load power of the quaternary crusher for particular throw setting, around 85 kW/crusher, the power draw of the 4th crushing was around 120 kW/each. However, the available industrial size is around 200 kW/each which could handle up to 100 kW crushing power demand for the suitable feed rate and *CSS* combination.

Table 12. Power, specific energy, and reduction ratio relationship for crushing levels

Crushing stage	Crushing Power kW	Throughput of crusher Mg/h	Specific Energy kWh/Mg	Reduction Ratio kWh/Mg
1	44	180	0.24	4
2	95	180	0.53	3
3 (×2)	137	340 (170 each)	0.40	1.8
4 (×2)	67	220 (110 each)	0.30	1.3
Total	342		1.5	

Start-Up

After mechanical improvement of the plant was done, the plant was commissioned. Each vendor tested their machine separately. Finally, our team started-up the plant. Plant start-up tests were carried out at two steps gradually.

Half-capacity tests

1500 Mg/d or 90 Mg/h, half capacity, tests were done using one parallel line of circuit given in Fig. 8 including 1 secondary, tertiary and quaternary crushers with the related screens.

Full-capacity tests

After half capacity tests were successful, 3000 Mg/d or 180 Mg/h, full capacity, tests were done using all circuit given in Fig. 8 including 2 tertiary and quaternary crushers with the related screens. These tests were done in December. The plant used to work around 30–40 Mg/h in seasons like December before the optimization work due to the plant and weather conditions. However, after extensive effort for months the goal was achieved, Table 13.

Table 13. December 2013 – crushing data from control room

Days	Crushed Ore (Mg)	Apron Working Hour (h)	Av. Capacity (Mg/h)
1	1485	8.8	169
*2	787	4.8	165
*3	0	0.0	0
*4	522	3.2	166
5	1370	9.1	150
6	1645	6.8	241
7	1606	7.7	209
8	1001	5.7	177
**9	42	0.3	158
**10	620	6.2	100
11	978	6.5	152
TOTAL (including plant stop)	10056	59	171
TOTAL (not-including plant stop)	8085	45	182

**old crusher shaft breakdown*
***snow and freezing*

The plant was tested in the freezing times for 11 days after the start-up and the capacity was reached. The solid data was based on total ore processed versus apron working hours. It was shown that that the design target was met for the capacity wise.

Plant sampling campaign

The plant was operated for another 5 weeks. Both single line and full line tests were continued in this period. A comprehensive sampling campaign was aimed after the plant properly worked. The sampling points (total 23 points) for the plant were detected and 2 to 4 samples were obtained at different times. The equipment settings were set to the following values in Table 14 during the sampling and for the rest of the plant operations, which were subjected to the change time to time.

Table 14. Setting during the tests for the rest of the operations

	Value (mm)	*CSS* / Aperture
Jaw	110	*CSS*
Secondary crusher	37	*CSS*
Secondary screen-double deck	25/12	aperture
Secondary screen-banana	25/12	aperture
Tertiary crusher 1	12	*CSS*
Tertiary crusher 2	12	*CSS*
Quaternary crusher 1	6	*CSS*
Quaternary crusher 2	6	*CSS*
Quaternary screen 1	6	aperture
Quaternary screen 2	6	aperture

Final size distribution

After taking the samples, the size distribution and efficiency analyses were carried out. Conclusions were drawn from this study. Final settings and mechanical optimization (mechanical optimization and tune-up) of the study were done based on the conclusions from sampling work. Figure 11 shows the sample screen analyses, which were taken from the stream of 4^{th} stage crushing final product (quaternary screen underflow) during final sampling campaign. It was shown that that the design target was met for the final product size wise.

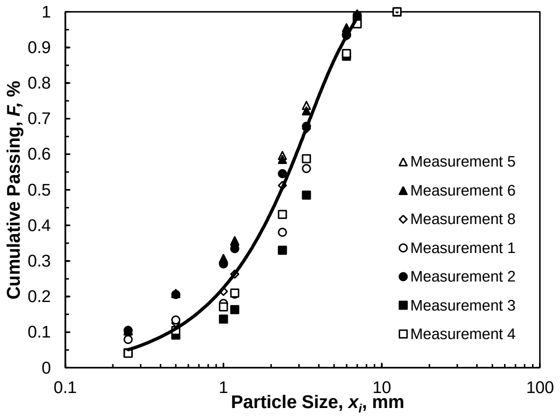

Fig. 11. Plant final product (quaternary screen underflow) sent to fine ore silo before agglomeration

Reduction ratio at 180 Mg/h

Reduction ratio analysis of the plant is meaningful to carry out based on the known (measured) value rather than a guess (primary crusher F_{80}). Since the experimental measurement of plant feed was not possible, the reduction ratio was given as the ratio between jaw P_{80} and quaternary screen underflow P_{80}. Table 15 presents that the design target was met also for reduction ratio.

Table 15. Reduction ratio (*RR*) (design vs. measurement)

		Design	Average of Measurements
1	Jaw F_{80}, mm	562	–
2	Jaw P_{80}, mm	139	135
3	P_{80} 4^{th} screen underflow, mm	4.65	4.55
2/3	*RR* (based on Jaw P_{80})	29.9	29.7
1/3	*RR* (based on Jaw F_{80})	121	–

After 8 months of intensive work the plant could reach the target even better than what was estimated in the simulations (P_{80} is 4.55 against 4.65, percent passing at 3.35 mm was 68 against 55.4) at 200 (±10%) Mg/h. By doing this minimum amount of money was spent and plant became process friendly. The flexible layout design with stockpile added has provided continuous gold production capability.

Conclusions

There are several conclusions drawn from the study. The design specifications were met (Table 1). The plant achieved 100% of -6 mm crushing product with the target size of 80% of -4.55 mm at a capacity of 200 (±10%) Mg/h. The product size for the best recovery, 3.35 mm, was measured between 63% and 68%. Secondary crushers achieved reduction ratio of 2–2.5 in average. Secondary screens' overall screen efficiency at 12 mm cut size was around 83–84%. Tertiary crushers achieved reduction ratio of 3-4 in average. Quaternary crushers achieved reduction ratio of 1.2–1.8 in average. Tertiary screens' overall screen efficiency at 6 mm cut size was around 73–78%. Maximum circulating load was obtained in the feed stream of double deck screen. Plant was ramped-up and optimized with the minimum investment using all available equipment in the system. Consumables and wear materials consumption were reduced.

Overall, this study showed a successful combination of the plant measurements, lab scale experiments, and modeling work to increase the capacity of a gold heap leach plant. Considering the importance of creating plant conditions in a simulation environment, in this study, the appropriate model selections, model calibrations, creating plant base case, and moving forward from the base to the scenarios were successfully applied. The modeling aided proposed design was implemented by changing current layout to reach at the desired capacity. This study also indicated a complete and a successful combination of the theory and practice for the optimization and capacity increase of a gold crushing plant.

Acknowledgement

Author would like to thank Mr. Sabri Karahan (General Manager of DAMA), Okan Vural (Mining Engineer) who spent several months on site to contribute to the study, Askın Karahan of DAMA for taking picture given in Fig.9., Murat Us (Metso Crushing Section Manager of Turkey) for providing Bruno for simulations, Sandvik people Orkun Oksuz, Bahadir Erden, Arkin Dik, Aimroc Management, Superintendent Tevfik Herdurak, ADR Foreman Ahmet Bey, lab technician Shamil Omerov, all the staff for plant sampling campaign and Elgam Qurbanov who directed the mechanical construction works.

References

BALLANTYNE G.R., POWELL M.S., 2014, *Benchmarking comminution energy consumption for the processing of copper and gold ores*, Minerals Engineering, 65, 109-114.

BOYD K., 2002, *Crushing Plant Design and Layout Considerations*, Mineral Processing Plant Design, Practice, and Control Proceedings, Vol. 1, pp. 669-697.

DELGADILLO J.A., LOPEZ-VALDIVIESO A., TELLO A., 2008, *Optimization of a grinding and classification circuit of a magnetite ore processing plant through computer simulation*, Minerals & Metallurgical Processing, 25(4), 223-228.

DHAWAN N., SAFARZADEH S., MILLER J., MOATS M., RAJAMANI R., 2013, *Crushed ore agglomeration and its control for heap leach operations*, Minerals Engineering, 41, 53–70.

FRUCHTBAUM J., 1988, *Bulk Material Handling Handbook*, Springer Science, New York.

JOWETT A., SUTHERLAND D.N., 1985, *Some theoretical aspects of optimizing complex mineral separation systems*, International Journal of Mineral Processing, 14(2), 85–109.

KAPPES D.W., 1979, *Precious Metal Heap Leach Simple – Why Not Successful?*, December, Northwest Mining Association.

KAPPES D.W., 2005, *Heap Leaching of Gold and Silver Ores*, Advances in Gold Ore Processing, 15, 456–478.

KCA, *Chovdar Scoping Study*, June 2011.

KING R.P., 1990, *Simulation—the modern cost-effective way to solve crusher circuit processing problems*, International Journal of Mineral Processing, 29, 249-265.

KING R.P., 2001, *Modeling and Simulation of Mineral Processing Systems*, Butterworth-Heinemann, Oxford.

LIRA-FLORES J., VAZQUEZ-ROMAN R., LOPEZ-MOLINA A., MANNAN M.S., 2014, *A MINLP approach for layout designs based on the domino hazard index*, Journal of Loss Prevention in the Process Industries, 30, 219-227.

LOTTER N.O., 2011, *Modern Process Mineralogy: An integrated multi-disciplined approach to flowsheeting*, Minerals Engineering, 24, 1229–1237.

LOTTER N.O., OLIVEIRA J.F., HANNAFORD A.L., AMOS S.R., 2013, *Flowsheet development for the Kamoa project – A case study*, Minerals Engineering, 52, 8–20.

LYNCH A.J., 1977, *Mineral crushing and grinding circuits, Their simulation, optimization, design and control*, Vol. 1, Developments in Mineral Processing, DW Fuerstenau, adv. ed., Elsevier Scientific, Amsterdam.

MARSDEN J. and HOUSE I., 2009, *The Chemistry of Gold Extraction*, SME, Second Edition, Colorado, USA.

MCKEE D.J., NAPIER-MUNN T.J., 1990, *The status of comminution simulation in Australia*, Minerals Engineering, 3(1-2), 7-21.

MYERS J.F., LEWIS F.M., 1946, *Fine Crushing with a Rod Mill at the Tennessee Copper Company*, AIME Chicago Meeting, February.

REIMERS C., WERTHER J., GRUHN G., 2009, *Design specifications in the flowsheet simulation of complex solids processes*, Powder Technology, 191, 260–271.

RUNGE K.C., ALEXANDER D.J., FRANZIDIS J.P., MORRISON R.D. and MANLAPIG E.V., 1998, *JKSimFloat – A Tool for Flotation Modelling*, AusIMM'98- The Mining Cycle, Mount Isa 19-23 April.

TORRES V.M., CHAVES A.P. and MEECH J.A., 1999, *Process Design for Gold Ores: A Diagnostic Approach*, Minerals Engineering, 12(3), 245-254.

TUZCU E.T. and VURAL O., 2014, *Mechanical Improvement and Simulation Assisted Optimization of a Gold Heap Leach Crushing Plant for Capacity Increase*, IMEC-2014, September 21-24, San Luis Potosi, Mexico.

TUZCU E.T., 2013, *Iron Plant Mill Optimization through the Optimization of a Single Coefficient via Simulation*, 142nd SME Annual Meeting & Exhibit, Colorado Convention Center, Colorado, Denver, February 24 -27.

WHITEN W.J., 1973, *The simulation of crushing plants with models developed using multiple spline regression*, Application of Computer methods in the Mineral industry. Eds. M.D.G. Salamon and Lancaster. S. Afr. Inst. Min. Metall. Johannesburg, 317-323.

WHITEN W.J, WALTER G.W, and WHITE M.E, 1973, A breakage function suitable for crusher *models*, 4^{th} Tewksbury Symposium, Melbourne, Feb P19.1-19.32

YINGLING J.C., 1990, *Circuit analysis: optimizing mineral processing flowsheet layouts and steady state control specifications*, International Journal of Mineral Processing, 29(3–4), 149-174.