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Proposal to expand the pumping system for treatment and recirculation of underground water in a mining operation

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Abstract*– This work presents a proposal for the expansion of a water treatment system inside an underground mine, and thus optimize the water resource present in the mining operation. For this purpose, the design parameters of an existing two-stage pumping system were determined and the hydrogeological conditions of the mine were analyzed. It was determined that a 500 HP pump was necessary that can deliver a dynamic head of 75 meters. Finally, an evaluation of the economic feasibility of the expansion design towards a three-stage pumping system with sedimentary pumps was carried out for its implementation. An approximate CAPEX of the pumping system of US$ 13,571,442 was determined; and an OPEX of 0.299 US$/m^3. Likewise, based on the estimated VPT of the mineral, the reserve report, the CAPEX of the pumping system and other projects, a NPV of the mine of US$ 65,562,057 and an IRR of 18.10% was estimated.

***Keywords*-- Pumping system, mining operation, underground mining, water recirculation, water treatment.**

I. INTRODUCTION

The role of water in mining operations plays a very important role [1]. This resource can serve as an alternative supply for domestic and industrial use; however, it can also become a problem from a geomechanically and environmental point of view [2] due the production of acid mine drainage (AMD) [3-5]. If water is not managed in the best way, a large volume of AMD can be produced, which can affect the environment, affecting the performance of mining operations and impacting production, productivity and operating costs [6,7].

For an underground mining operation to be carried out, it is essential to have adequate water management and management [8,9]. As operations deepen, the chemical composition of the groundwater varies in content of metals and suspended solids [10]. On the other hand, since water is a very valuable resource, it is important to treat and recirculate it within the mining operation [11]. Due to the depth of the mine, the use of gravity from the lower levels of the underground exploitation to the surface is not possible. In this sense, the use of a system for collecting, pumping and treating water inside the mine is an alternative solution to the handling and management of underground water [12,13].

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The diagram in Figure 1 shows us the main components of a mining pumping system, where hydraulic pumps are the main axis of its operation. An alternative solution to increasing the volume of groundwater is to expand an existing pumping system to control, treat and dispose of a greater amount of groundwater [14].

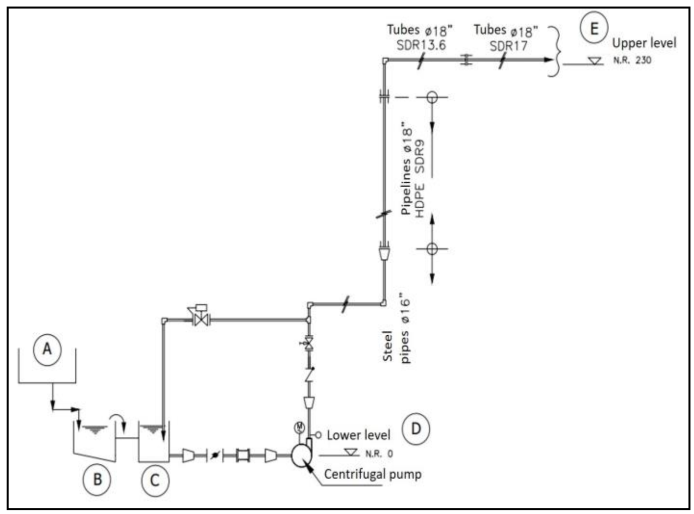

Fig. 1 Diagram of a pumping system [15].

II. METHODOLOGY

A. *Case study*

For the case study, an underground mining operation located in the central high Andean area of Peru was chosen (Figure 2). The study area is located at the head of the Mantaro River basin, at 4600 meters above sea level (m.a.s.l.), in the district of Huayllay, department and province of Pasco, within the mining concession from Empresa Administradora Chungar S.A.C (EACH), at an approximate distance of 325 km from the city of Lima. The type of operation of this mining unit is Over Cut and Fill (OCF).

The existing pumping system in the mine was designed for a certain level and flow of water. The discharge calculations indicate that for Level 4000 there is a discharge between 700 to 800 l/s, and for level 3900 there will be flows of 900 to 1000 l/s, calculated from the deepening of two ramps and the construction of the horizontal workings of the mine.

The sustainability of mining operations will require greater efforts through the implementation of optimization processes and efficient use of resources. The current pumping system does not guarantee a correct mining cycle, when necessary, nor does it guarantee compliance with the projected progress. To meet the annual mineral tonnage projection of approximately 1,540,000 DMT, the auxiliary services area must guarantee a reliable pumping system, therefore, a project to expand the current two-stage pumping system is necessary. In this sense, the expansion of the pumping system to three stages was proposed, so that the future mineral extraction process is carried out in an environment free of water accumulation that would harm the transit of heavy equipment and personnel.

Fig. 2 Location of underground mining operation.

B. *Meteorological conditions*

The project area belongs to the central zone of the central mountain range of the Andes, between the limits of the regions of Lima and Cerro de Pasco. It is characterized by its high altitude (>4,000 m.a.s.l.) and the existence of a system of lagoons generated by the natural melting of the mountain chain. It is necessary to have information about climatological characteristics for the study area, and verify their influence on the recharge of groundwater in the mine area; as well as the hydraulic behavior of the lagoons and their impact on the parameters of the hydrological cycle.

1) *Precipitation cycle*

Precipitation in the area usually occurs after noon; where the season of greatest rainfall occurs from December to March, with February being the month with the greatest intensity of rain. Furthermore, the dry season from May to November has an absence of precipitation. For precipitation, the values from four climatological stations (Upamayo, Huayllay, Ayaracra and Cerro de Pasco) were taken into account. The average precipitation of the study area considering an average altitude of 4600 m.a.s.l. was determined. Table 1 shows us average monthly precipitation for the study area.

TABLE I
AVERAGE MONTHLY PRECIPITATION (MM)

Monthly average total precipitation (mm)					
Jan.	Feb.	Mar.	Apr.	May.	Jun.
154.9	185.6	166.2	90.2	43.3	18.2
Jul.	Aug.	Sep.	Oct.	Nov.	Dec.
16.7	31.1	53.8	129.9	135.4	141.2
Annual = 1166.5					

2) *Evaporation.*

Evaporation was analyzed based on data from the Upamayo station, resulting as an annual average 775 mm/year. Also values of up to 1101 mm/year in years humid and 481.2 mm/year in dry years were registered (see Table II).

TABLE II
AVERAGE MONTHLY EVAPORATION (MM)

Monthly average total evaporation (mm)					
Jan.	Feb.	Mar.	Apr.	May.	Jun.
74.3	65.7	69.6	57.6	56.2	53.1
Jul.	Aug.	Sep.	Oct.	Nov.	Dec.
57.0	63.0	65.6	69.5	70.8	72.5
Total = 775					

3) *Temperature*

The main station presents regular temperatures without sudden variation, having an average of 4.6 °C (Table III).

TABLE III
MONTHLY AVERAGE TEMPERATURE (°C)

Monthly average temperature (ºC)					
Jan.	Feb.	Mar.	Apr.	May.	Jun.
5.5	5.5	5.5	5.4	5.0	3.8
Jul.	Aug.	Sep.	Oct.	Nov.	Dec.
3.1	2.8	3.4	4.3	5.1	5.1
Annual = 4.6					

4) *Relative humidity (RH)*

The relative humidity of the area can be variable. The humidity relative of the area, as well as the minimum and maximum values is show in Table IV.

TABLE IV
RELATIVE HUMIDITY

Average relative humidity	62.2
Minimum relative humidity	60.4
Maximum relative humidity	84.5

5) *Atmospheric pressure.*

The atmospheric pressure was 59.9 kPa (equivalent to 0.59 atm).

C. Physicochemical parameters of surface and mine interior waters.

Knowledge of the physicochemical properties of surface and interior waters mine aims to establish the relationships between the composition of the underground waters with the geology and mineralogy of the deposit ore. The results were obtained from representative sampling points. Selected point based on the lithology of the rock box, vein zone, presence of thermal waters, leaks in faults and fractures were chosen.

1) Electric conductivity (E.C.)

Electrical conductivity (E.C.) data were recorded during sampling in surface, such as springs (MQ-01), lagoons (LNN-01, LNS-01, LH-03) and piezometer (P 05); as well as water inside the mine at the discharge fronts, seepages and drillings (Figure 3).

In the wetlands area, values from 96 to 355 µS/cm were obtained; in springs values of 165 and 259 µS/cm; and in water bodies of lagoons values between 137 and 1310 µS/cm being the North Naticocha Lagoon has the highest salinity due to the discharge of mine waters. The waters mineheads register values between 776 and 2226 µS/cm; while the effluent waters mine vary between 1088 and 3250 µS/cm.

Finally, the waters inside the mine are between 950 to 2560 µS/cm. The recent groundwater being less mineralized with values between 950 and 970 µS/cm. The waters of intermediate depths, the product of the mixture of water and sulfide oxidation, present values between 1080 and 1470 µS/cm. Meanwhile, the waters linked to deep thermal springs, mainly in the María Rosa, Principal and Lorena veins (Level 150 and 175), register values between 1500 and 2560 µS/cm.

2) Variation of hydrogen potential (pH).

The waters recorded on the surface show pH values between 7.39 and 8.97. The values recorded in lagoons show more alkaline, due to the presence of organic matter and aquatic vegetables. The waters of the mine entrance and lagoon (LL-02), affected by the acid rock drainage, have pH values between 2.70 to 5.07 (Figure 3).

The water samples inside the mine presented variable pH values between 5.9 to 7.3, some have greater acidity due to the sulfuric acid produced from the leaching of pyrite. The GEOH-08 drilling presents greater alkalinity related to waters containing carbonates. Meanwhile, mine leaks, drainage, pools and pumping chambers register a pH between 5.9 and 6.8; those that would be linked to deep and ancient waters.

3) Temperature variation.

The temperature values of surface waters, streams and lagoons register values between 7.5 and 16.8 °C. The water in the tailings dams and pools presents values between 8 and 11.2 °C, while the channels register values between 14.1 and 24.5 °C, influenced by environmental variation and underground circulation related to the depth of the water table. Mine waters related to leaks presented values between 15.3 and 41.2 °C; while the drillholes GEOH-01 and GEOH-02 presented values between 17.9 and 43.6 °C, respectively. Being the highest spots of temperature as they were related to the hot springs of the main vein (Figure 3).

4) Total dissolved solids.

The values of the concentration of total dissolved solids (TDS) in surface waters ranges from 49 to 565 ppm. Being the P-05 (268 ppm) and LNN-01 (565 ppm) of the Naticocha Centro lagoon the highest values registered (Figure 3). Both sampling pints were influenced by the discharge of mine waters. The water inside the mine from seeps, drains, pools and pumping chambers recorded values between 460 and 1692 ppm. Meanwhile, the drillings showed values of 210 and 1320 ppm. In the mine mouth waters, deep underground flow waters high TDS values between 388 to 1128 ppm were observed. In addition, the trends to increase the temperature according to the depth of the operations is observed. The pH of the solution remains constant regardless of the level.

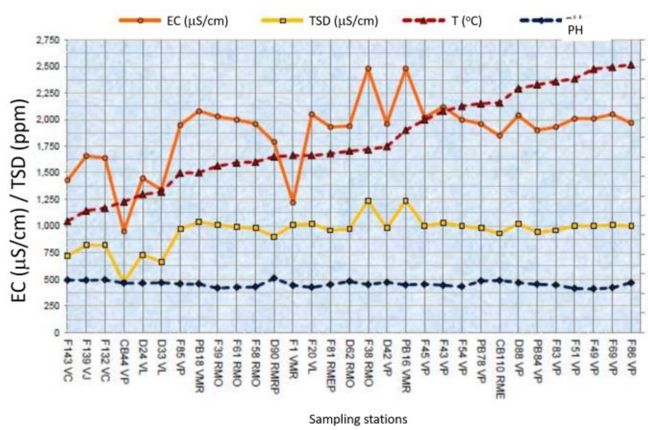

Fig. 3 Variation of physicochemical parameters in internal mine waters

D. Historical behavior of mine drainage.

The mine water discharges recorded at the mine's E-2 Monitoring Station, located at the exit of the Treatment Plant, indicate that the pumping flows were increasing according to the deepening levels. Between 2003 and 2008, drainage flows varied according to the deepening work between 90 and 150 l/s. During the years 2009 and 2010, the greatest jump was recorded, with flows of 150 and 450 l/s, having tripled. This coincides with the deepening operations of the mine. The greatest jump in flow occurred with the deepening work at Level 4250, which extended to Level 4175. As the water table had been suddenly depressed, the discharge fluctuated during 2010, and subsequently a trend towards reduction was recorded due to stabilization of the hydraulic gradient. For 2013, a total flow of 530 l/s was reported from the entire inventory of sources by levels inside the mine, with maximum values of 720 l/s for Level 4115. Currently, the mine demands a flow of 900 L/s. The Figure 4 shows us the historical flow demand of the mine according to its depth.

E. Geology and Mineralization.

1) Local Geology

Lithological, the site is made up of sedimentary rocks that reflect a period of emersion and intense denudation. The "Red Layers" of the Casapalca Group present two sedimentation cycles. The oldest cycle is the most powerful at 1400 to 1500 meters thick and the youngest cycle is 800 to 900 meters thick. Each cycle in its lower part is characterized by the abundance of conglomerates and sandstones, in its upper part they contain conglomerate and volcano-clastic horizons. The gradation of the clasts and their orientation indicate that the materials have come from the East, probably from the area currently occupied by the Eastern Cordillera of the Andes.

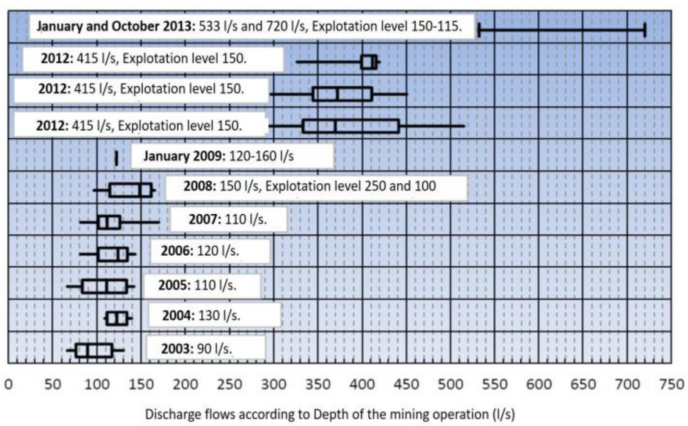

Fig. 4 Historical pump flow diagram.

2) Mineralization

The mineralization of the mine is not clearly related to an outcropping intrusive, so it is presumed that the mineralizing focus would be located at depth within the classic arrangement of base metal vein-type hydrothermal systems. Until now, the most important polymetallic mineralization is housed in a system of tensional fractures developed in the Eastern sector where the Main vein is located, and secondly the Andalucía vein as well as other minor ones that locally form sigmoidal arrangements mostly with the first.

The most important vein is the Main vein, trending E-W and dipping 65 to 75 degrees to the north, with almost 2km long, 600 m recognized current depth and power ranging from 0.50 meters to 12 meters wide. In addition, this vein presents subsequent nails of excellent mineral filling that also varies in its length, presenting in the central and deep part the most developed sector of massive sphalerite mineral and in the sector, this domain of mineralization in carbonates with interesting values of Pb-Ag.

The second most important vein is the Janeth vein, heading E-W and dipping 65 degrees to the south, with almost 900 meters long, and 400 meters of current depth and power ranging from 0.50 meters to 7 meters wide, it presents mineral filling of sphalerite, galena, pyrite carbonate gangue and some quartz. Currently four levels have been developed in the intermediate zone of the mine. Due to its intermediate position, its contribution to production is important in terms of lead and silver.

F. Hydrogeology.

The hydrogeological study area of the mine is located within the scope of the Naticocha microbasin, and Quimacocha, both of which are part of the Huaroncocha microbasin. The study area has high rainfall corresponding to a mountain range area, with typically extreme Andean climates.

The area is located on stratified sedimentary rock blocks, composed of marl and shale in the lower parts of the plateau, while in the high mountains there are sandstones and limestones, and intrusions of igneous and subvolcanic dikes.

G. Production.

The production process consists of the extraction of polymetallic mineral (Ag-Pb-Zn Cu). The mine has 5 main exploitation veins, which are the following: Ramal Piso 3 Principal, Andalucía 120, Janeth, Principal, Karina 1 and Ofelia. The extraction of the mineral occurs through the methods: over cut and fill (OCF), Sublevel stopping (SLS) and Sub Level according Breasting criteria (SN) (Table V). The ore is transported by dump trucks and dumpers and sent to the concentrator plant at a rate of 4,200 TMH/day (wet metric tons per day) for treatment.

TABLE V

ANNUAL PRODUCTION SCHEDULE BY TYPE OF METHOD

Method	TMS	% Zn	% Pb	% Cu	Oz Ag
SN	445.581	4.62	1.24	0.14	1.38
OCF	959.999	5.75	1.24	0.14	1.46
SLS	127.420	7.27	2.04	0.21	2.20
Total	1,533,000	5.55	1.31	0.14	1.50

H. Reserves and resources.

Mineral reserves and resources have been estimated for 2018 in accordance with international standards established by the Joint Ore Reserves Committee (JORC7). The reserve calculation for 2018 was made using long-term metal price projections: 2,450 USD/MT for zinc, 2,100 USD/MT for lead, 6,400 USD/MT for copper and 18.5 USD/oz for silver. Table IV shows the reserves by category.

TABLE VI

ORE RESERVES BY CATEGORY

Category	MT	AV(m)	AM (m)	Zn (%)	Pb (%)	Cu (%)	Ag (oz/T)
Proven	3,999,604.13	3.19	4.16	6.14	1.85	0.17	2.24
Likely	5,336,762.51	3.06	3.99	5.65	1.74	0.15	1.89
Total	9,336,336.64	3.12	4.06	5.86	1.79	0.16	2.04

III. RESULTS

A. Technical evaluation

1) Two stage pumping system

The mine's current water pumping system considers pumping in two stages: from Level 4100 to Level 4355 and from Level 4355 to Level 4610 (surface) (Figure 5). The pumping station Level 4100 consists of the use of 12 bombs distributed in three trains of 4 bombs each (two trains running and one in stand-by) with 500 HP engines and a nominal capacity 300 l/s for each train (Figure 5). The pumping station at Level 4355 contemplates the use of 9 pumps distributed in 3 trains of 3 pumps each (two running trains and one stand-by) with 550 motors HP and a nominal pumping capacity of 300 l/s for each train.

In the same way, there is a Level surface water conduction system 4610. This level contemplates the use of a water receiving pond inside the mine and gravity conveyance to the existing sedimentation pond. The hydraulic values for the two current stations are presented in Table VII.

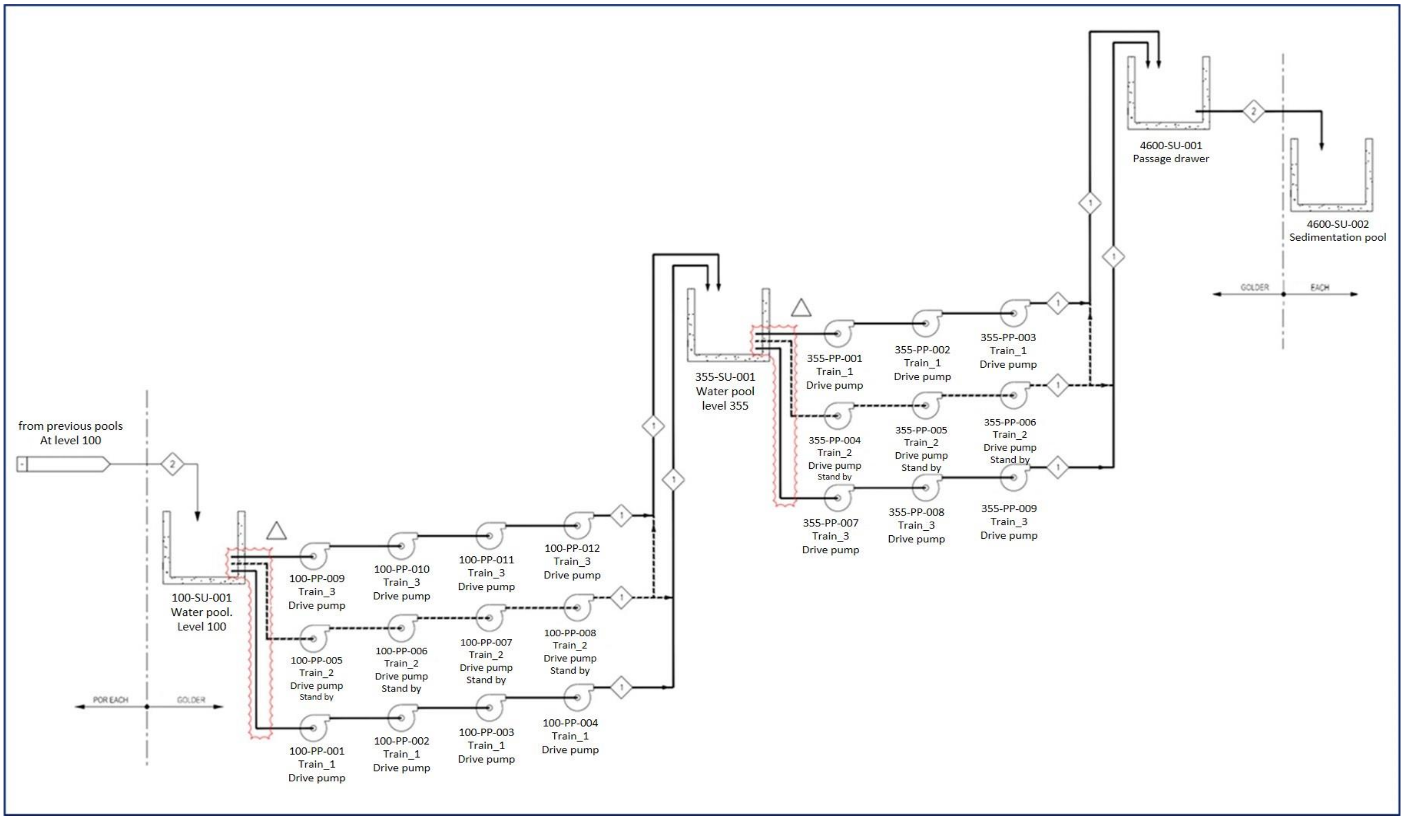

Fig. 5 Current pumping system.

TABLE VII
HYDRAULIC PUMPING CALCULATION FOR THE PROPOSED SYSTEM

Description.	UND.	Station design level 4100	Station design level 4355
Nominal flow per train of bombs	l/s (m^3/h)	270 (972)	210 (756)
Total nominal pumping flow	l/s (m^3/h)	540 (1944)	420 (1512)
Solids content (maximum)	% w/w	< 5%	< 5%
Fluid temperature	°C	30-35	30-35
Elevation of the water mirror in the water pool	m.a.s.l.	4089.19	4357.29
Pumping station elevation	m.a.s.l.	4079.00	4343.36
Elevation of the highest point of the discharge pipe	m.a.s.l.	4357.29	4612.36
Drive pipe run length	m	856.70	452
Static charge	m.w.c.*	255	269

*m.w.c. = meter of water column.

2) Design proposal

The new pumping system proposed will be a three-stage system of settled waters designed to evacuate a flow rate of 1,200 l/s, an estimated flow rate according to the historical drainage of the mine for level 3850. For this, multistage centrifugal pumps will be used, distributed in 4 trains in each stage (Figure 6).

The proposed pumping system for the mine considers pumping in three stages: from Level 3845 to Level 4100, from Level 4100 to Level 4355 and from Level 4355 to Level 4610 (surface). For the pumping station at Level 3845, the use of 20 pumps distributed in five trains of 4 pumps each (four trains running and one in stand-by) with 500 HP motors and a nominal capacity of 300 l/s per pump is proposed. every train. Likewise, the installation of a 2300 m^3 feeding pond.

For the existing station at Level 4100, it is proposed to install two more trains, with 4 pumps each with 500 HP motors (four trains running and one in stand-by), maintaining the nominal capacity of 300 l/s for each train. For the Level 4355 station, it is proposed to expand the number of pumps for each existing train, from three to four pumps, and the installation of two trains of 4 pumps each with 500 HP motors, with the same nominal capacity of 300 l /s (a total of four trains running and one on standby). The number of pumps per train is increased due to the loss that each train has when draining groundwater (210 l/s is sucked out of a nominal capacity of 300 l/s). Before pumping, a laminar sedimentation system is being planned for each station inside the mine.

The clear waters will be received in a chamber, and will be pumped to the next stage that will begin at Level 355 of the Montenegro shaft, to be finally pumped to the surface. On the surface, the clear waters will be deposited in a collection tank, and then diverted to the northern Naticocha lagoon.

The sludge generated in the sedimentation system will be transferred with a submersible pump to sludge ponds (3), which will be stored preliminarily. Later, it will be evaluated to transfer the sludge to abandoned works, as well as to use the sludge with detrital fill in the process. hydraulic filling. Figure 5 shows a sketch of the design proposal for the 3-stage pumping system.

3) Hydraulic calculation.

The hydraulic calculation for the three stations is presented in Table VIII.

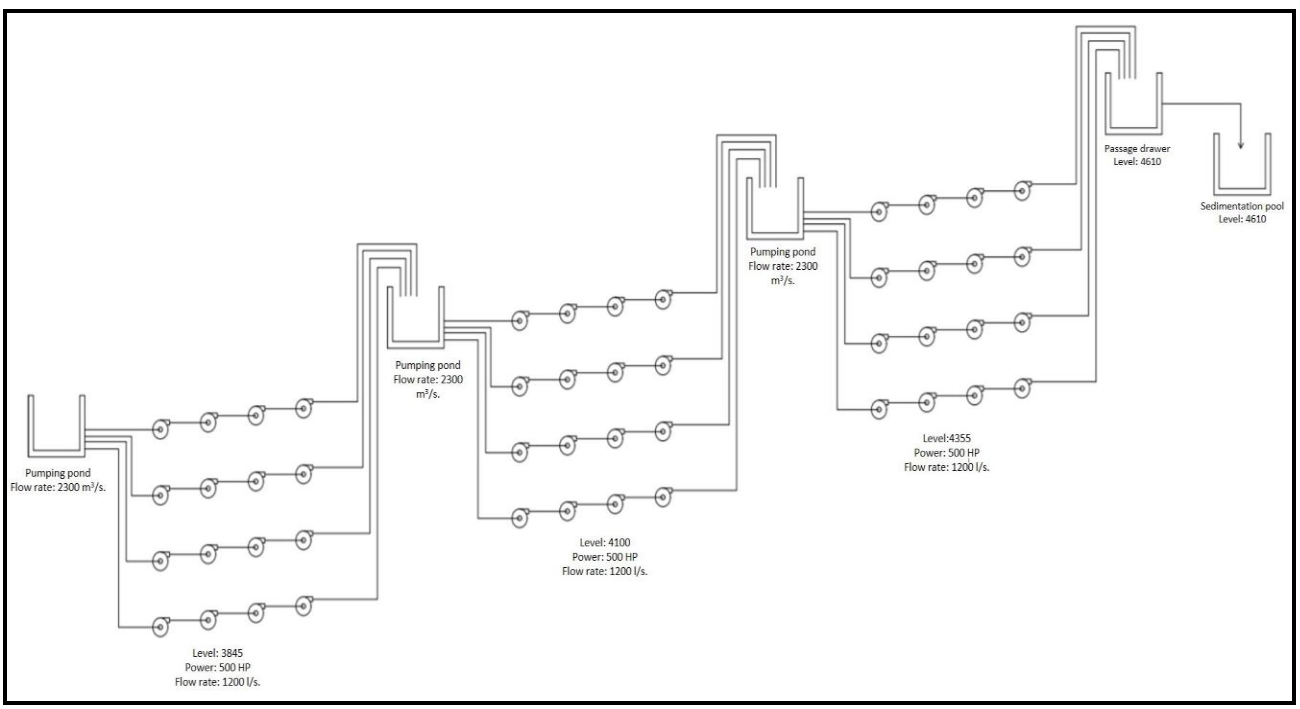

Fig. 6 Proposal for a three-stage pumping system.

To calculate friction losses, it was considered to use the Manning formula [16]:

$$hf = 10.3 \frac{n^2}{D^{5.33}} \, x \, Q^2 x \, L \qquad (1)$$

Where hf = loss of charge or energy (m); Q = flow rate (m^3/s); n = roughness coefficient; D = internal diameter of pipe (m) and L = pipe length (m). The value of n depends on the roughness of the pipe, whose value depends on the type of material.

According to the calculation carried out, considering the friction losses depending on the type of pipe, there will be a dynamic load between approximately 270 and 300 meters for the three pumping stations. For this, 500 HP pumps will be needed to supply the necessary power pumping for the estimated mine flow for the deepening levels (1200 l/s).

TABLE VIII
HYDRAULIC PUMPING CALCULATION FOR THE PROPOSED SYSTEM

Description.	UND.	Station design level 3845	Station design level 4100	Station design level 4355
Nominal flow per train of bombs	l/s (m^3/h)	300 (1080)	300 (1080)	300 (1080)
Total nominal pumping flow	l/s (m^3/h)	1200 (4320)	1200 (4320)	1200 (4320)
Number of pumps in series (train)	---	4	4	4
Water pH	---	6.7	6.7	3.7
Solids content (maximum)	% w/w	< 3%	< 3%	< 3%
Fluid temperature	°C	30-35	30-35	30-35
Pumping station elevation	m.a.s.l.	3845	4079	4343.36
Pipe internal diameter	m	0.409	0.409	0.409
Length	m	650	608	694
Friction losses	m.w.c.	0.11	0.10	0.12
Pipe internal diameter	m	0.41	0.41	0.41
Length	m	1100	1105	210
Friction losses	m.w.c.	0.04	0.04	0.01
Fittings (elbows, tee, valves)	m.w.c.	35	30.98	36.48
Static charge	m.w.c.	234	264.36	266.64
Total dynamic head	m.w.c.	269.15	295.48	303.25
Total dynamic head for each bomb	m.w.c.	67.29	73.87	75.81
Power required for each pump	HP	414.08	454.59	466.53
	KW	308.78	338.99	347.89

*m.w.c. = meter of water column.

4) Pump selection.

For the selection of the pump, we will take as reference the hydraulic calculation carried out in the previous section. We will perform calculations to determine the total dynamic load per pump for different flow levels to determine the H-Q curve of the pumping.

Table IX summarizes the different measurement units for the flow rate and the total dynamic head per pump, which allows us to create the H-Q characteristic curve of the pumping system.

TABLE IX
H-Q VALUES OF THE PUMPING SYSTEM

Q (l/s) per pump	Q (l/min)	Q (m^3/h)	Q (gpm)	Dynamic head per pump (m)	Dynamic head per pump (feet)
6.25	375	22.5	99.06	75.81	248.66
12.5	750	45	198.13	75.91	248.97
25	1500	90	396.25	76.28	250.21
50	300	180	792.50	77.79	255.15
75	4500	270	1188.75	80.30	263.39
100	6000	360	1585.00	83.82	274.93
125	7500	450	1981.25	88.34	289.77
150	9000	540	2377.50	94.53	310.07

The Goulds Pumps 3316 pump was chosen for our study. This pump is a phase double and its horizontal casing is designed for boiler feeding, mine drainage and other services that require high moderate loads [17]. The Figure 7 shows us the characteristic curve of the pump. In red plot, we have the H-Q curve of our proposed pumping system.

It is observed that the intersection of the pump performance curve and the H-Q curve of the pumping system for 1188.75 gpm is given in pump size 6x8 – 17, n = 1750 rpm. This pump family is ideal for performing a dynamic head of 75 meters required with a power of 500 HP.

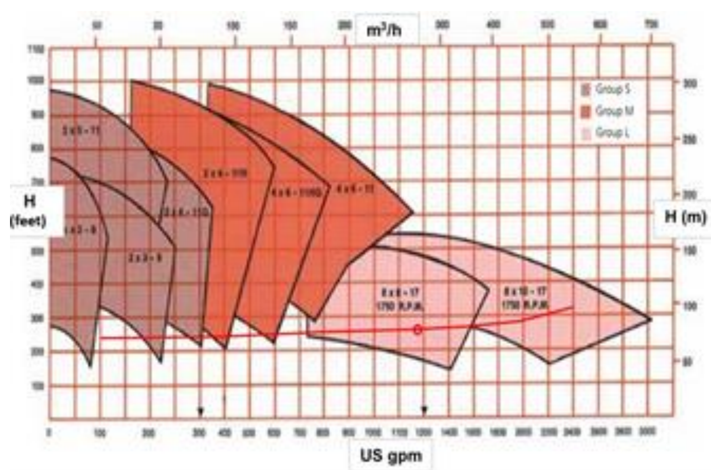

Fig. 7 Gould Pump 3316 pump performance curve [17].

B. Economic evaluation

1) Calculation of NSR and VPT

Based on the data provided by the marketing area, the payable and deductible parameters to include in the estimate of the Net Smelter Return (NSR) are determined. The prices of the metals with which the estimate will be made are Zn (2400 US$/t), Pb (2100 US$/t), Cu (6400 US$/t) and Ag (18.50 US$/oz).

The metallurgical recoveries of each metal are calculated from the reports monthly metallurgical measurements of the concentrator plant, taking into consideration the treated tonnage, concentrated tonnage, average head grades and average concentrate grades.

- Zn Concentrate: Zn Recovery 89.34% and Ag Recovery 7.60%
- Pb Concentrate: Pb Recovery 87.81% and Ag Recovery 63.08%.

The NSR method determines the sale of the minerals produced by the property less the associated production costs. With the following formulas the point value of each element is determined:

$$NSR_{Zn} = (PrZn * ZnL * ZnRZn * ZnPZn + PrAg * AgL * AgRZn * \frac{AgPZn}{31.10348}) * (1 - m\%) - \frac{ZnCT}{RtZn} - \frac{ZnCF}{RtZn} \quad (2)$$

Where $PrZn$ = zinc price, ZnL = zinc grade, $ZnRZn$ = recovery of zinc in zinc concentrate, $ZnPZn$ = payable for zinc in zinc concentrate, $PrAg$ = silver price, AgL = silver grade, $AgRZn$ = recovery of silver in zinc concentrate, $AgPZn$ = payable for silver in zinc concentrate, $m\%$ = loss percentage (contractual), $ZnCT$ = transportation costs of zinc concentrate, $ZnCF$ = processing costs zinc concentrate and $RtZn$ = concentration ratio in zinc concentrate.

$$NSR_{Pb} = (PrPb * PbL * PbRPb * PbPPb + PrAg * AgL * AgRPb * \frac{AgPPb}{31.10348}) * (1 - m\%) - \frac{PbCT}{RtPb} - \frac{PbCF}{RtPb} \quad (3)$$

Where $PrPb$ = lead price, PbL = lead grade, $PbRPb$ = recovery of lead in lead concentrate, $PbPPb$ = payable for lead in lead concentrate, $PrAg$ = silver price, AgL = silver grade, $AgRPb$ = recovery of silver in lead concentrate, $AgPPb$ = payable for silver in lead concentrate, $m\%$ = loss percentage (contractual), $PbCT$ = transportation costs of lead concentrate, $PbCF$ = processing costs lead concentrate and $RtPb$ = concentration ratio in zinc concentrate.

$$NSR_{Cu} = (PrCu * CuL * CuRCu * CuPCu + PrAg * AgL * AgRCu * \frac{AgPCu}{31.10348}) * (1 - m\%) - \frac{CuCT}{RtCu} - \frac{CuCF}{RtCu} \quad (4)$$

Where $PrCu$ = copper price, CuL = copper grade, $CuRCu$ = recovery of copper in copper concentrate, $CuPCu$ = payable for copper in copper concentrate, $PrAg$ = silver price, AgL = silver grade, $AgRCu$ = recovery of silver in copper concentrate, $AgPCu$ = payable for silver in copper concentrate, $m\%$ = loss percentage (contractual), $CuCT$ = transportation costs of copper concentrate, $CuCF$ = processing costs copper concentrate and $RtCu$ = concentration ratio in copper concentrate.

$$NSR_{Total} = NSR_{Zn} + NSR_{Pb} + NSR_{Cu} \quad (5)$$

The point values for each mineral is presented in Table X. The point value of copper is not taken into consideration for the VPT estimation due to its low grade.

TABLE X
POINT VALUE OF EACH MINERAL

Mineral	Unit	Point Value
Zn	US$/t	14.28
Pb	US$/t	15.58
Ag	US$/oz	9.64

Based on these values and the mineral reserves, we determine the VPT (US$/t) of the produced ore (Table XI).

TABLE XI
RESERVES WITH VPT VALUE

Category	MT	Zn (%)	Pb (%)	Cu (%)	Ag (oz/t)	VPT (US$/t)
proven	3,999,604.13	6.14	1.85	0.17	2.24	138.10
probable	5,336,762.51	5.65	1.74	0.15	1.89	126.01
total	9,336,336.64	5.86	1.79	0.16	2.04	131.23

Taking into account only the proven reserves and the mine's production of 4200 tpd, we have that the life of the mine will be 6.17 years.

2) Pumping system quote

For the quote of the pumping system, both CAPEX and OPEX were considered on the project. CAPEX includes mechanical equipment, electrical equipment, pipes, instrumentation, civil structures and mining works. And within the OPEX it was considered energy consumption, consumables, tanker transportation cost, labor cost inside the mine, equipment maintenance and possible eventualities.

A CAPEX of US$ 13,571,442 was determined. For the operating costs of the pump installation, we have a total annual cost of US$8,492,638 and a unit cost of US$0.299/$m^3$.

3) Operating costs

The cost to cover all costs assumed in mineral production is known as the total operating cost, expressed in dollars per ton (US$/t). This includes costs of mine, plant, maintenance, general mine services, operating costs of the pumping system, hauling, mine administration costs, which represents the total costs (fixed costs and variable costs).

Table XII summarizes the total operating costs of the mine according to the mining method.

TABLE XII
TOTAL OPERATING COSTS BY MINING METHOD

Mining method	Fixed costs (US$/t)	Variable costs (US$/t)	Total costs (US$/t)
OCF	28.3	35.9	64.2
SLS	25.3	32.0	57.3

4) Calculation of NPV and IRR

A total reserve of 9,336,366.64 tons have been determined, which will be considered to realize our cash flow. Within the CAPEX we consider investment in the pumping system, exploration, development, mine, plant, tailings dam, maintenance of heavy equipment, energy, administration, EACH, among others (Table XIII).

Considering the CAPEX and OPEX of the pumping system project, the VPT calculated for the ore produced, the life of the mine estimated based on the reported reserves and the total mine operating costs, the following cash flow was made. Consider that the CAPEX of the pumping system was only considered for year 0 (Table XIV).

Considering an interest rate of 12%, the NPV and IRR values of US$ 65,562,057 and 18.10% were estimated respectively.

TABLE XIII
ANNUAL CAPEX

CAPEX	Investment (US$)
Pumping system	13,571,441.78
Exploration	5,900,000.00
Development	26,400,000.00
Mine	13,400,000.00
Plant	2,400,000.00
Tailings	3,700,000.00
Maintenance	300,000.00
Energy	3,000,000.00
Administration	1,300,000.00
HSEC	100,000.00
Another	1,100,000.00
TOTAL	71,171,441.78

TABLE XIV
CASH FLOW

Period	Tonnage produced by year	Income (US$)	CAPEX (US$)	Expenses (US$)	Final balance (US$)
0	---	---	13,571,442	---	---
1	1,512,000.00	198,426,715	57,6000,00	97,070,400	101,356,315
2	1,512,000.00	198,426,715	57,6000,00	97,070,400	101,356,315
3	1,512,000.00	198,426,715	57,6000,00	97,070,400	101,356,315
4	1,512,000.00	198,426,715	57,6000,00	97,070,400	101,356,315
5	1,512,000.00	198,426,715	57,6000,00	97,070,400	101,356,315
6	1,512,000.00	198,426,715	57,6000,00	97,070,400	101,356,315
7	264,366.64	34,694,050	0	16,972,338	17,721,712

5) Sensitivity analysis.

There are two parameters that can vary when calculating the NPV of the project: the price of zinc and the interest rate.

The Figure 8 and Figure 9 show the variation of the NPV for different values of zinc, specifically between 1500 US$/t and 3500 US$. We take as reference the minimum historical value of zinc of 1478.50 US$/t on January 2016, and the maximum historical value of zinc of 3432.50 US$/t on February 2018 of the last 10 years (Exchange, London Metal, 2019). We see that the relationship between the price of zinc and the NPV is directly proportional.

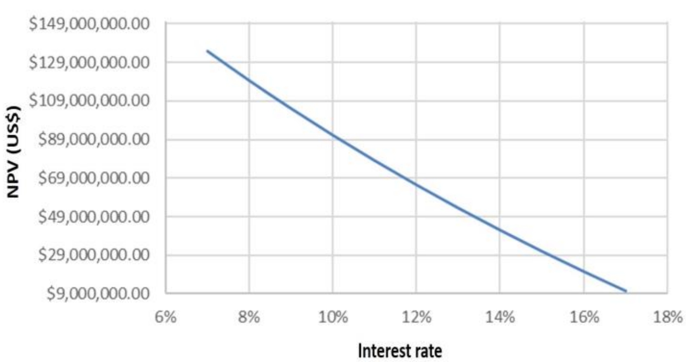

Fig. 8 Variation in NPV due to the price of zinc.

We see that the project remains viable even in the worst zinc price scenario. Similarly, we carry out the analysis of the variation of the NPV for different values of the interest rate, specifically between 7% and 17%. It is noted that the relationship between the interest rate and the NPV is indirectly proportional. It is observed that the expansion project remains viable despite the variations in the interest rate.

Fig. 9 Variation of the NPV due to the interest rate.

IV. CONCLUSIONS

The technical evaluation demonstrates that it is possible to propose a water pumping design in three stages, from Level 3845 to Level 4100, from Level 4100 to Level 4355 and from Level 4355 to Level 4610, to transfer water from the interior of the mine to the surface.

The pumping system consisting of multistage centrifugal pumps distributed in 4 trains in each stage, will allow the evacuation of a flow rate of 1200 l/s, an estimated flow rate according to the estimates of the mine's historical drainage for level 3850.

The economic evaluation shows that in the period of 6.17 years, estimated life time of the mine, the implementation of the pumping system is viable given that the NPV is US$ 65,562,057 and the IRR is US$ 18.10% for a CAPEX of US$ 13,571,442 and an OPEX of 0.299 US$/m^3.

The sensitivity analysis shows that the economic design proposal is still viable when the price of zinc decreases to a value of 1500 US$/t and the interest rate increases to a value of 17%.

It is possible to propose technical solutions to recirculate water in order not to impact this important resource, while respecting the environment.

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