KOLWEZI COPPER PROCESSING PLANT

SEPTEMBER 2022

GOLDEN MOUNTAIN TECHNOLOGY | 510, AVENUE KAMBOVE, LUBUMBASHI

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COPPER PROCESSING PLANT IN KOLWEZI

Feasibility Study

September 2022

Feasibility Study

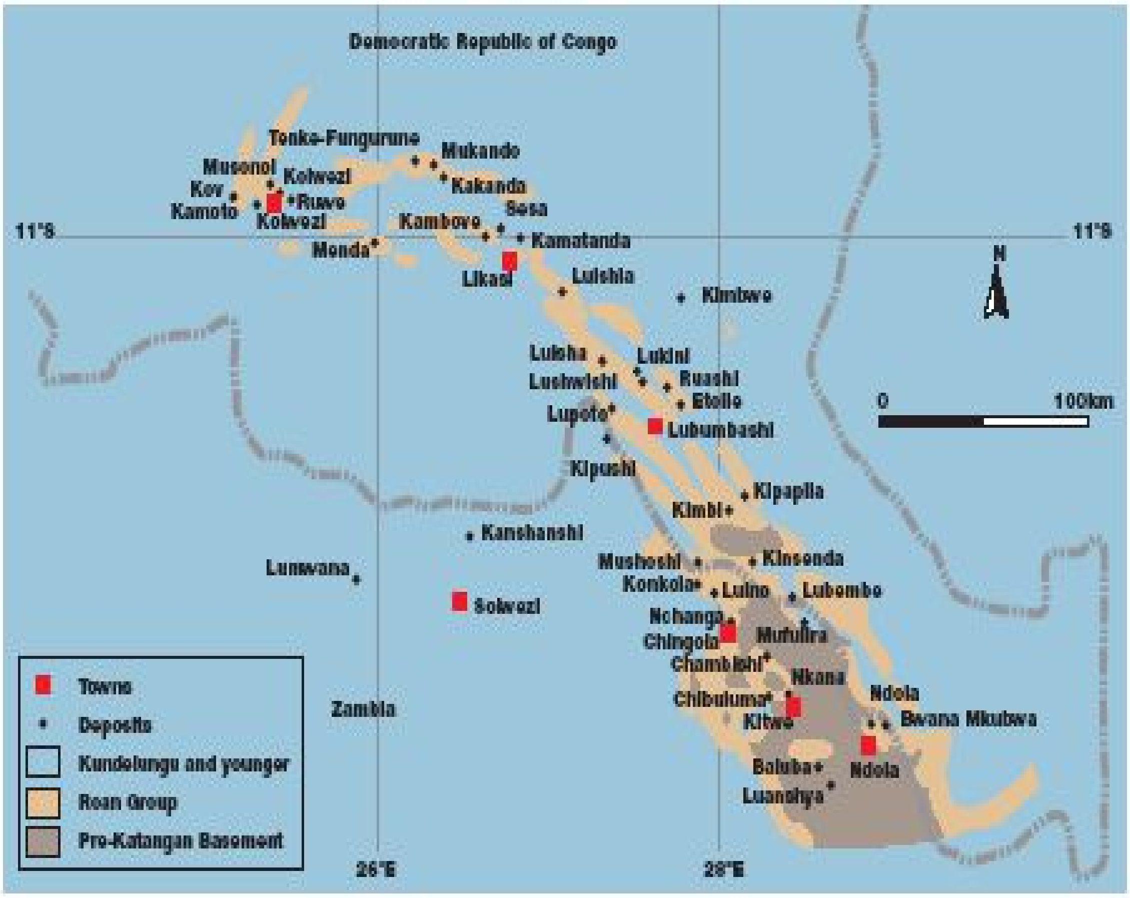
PROJECT TO ESTABLISH A COPPER METALLURGICAL PROCESSING PLANT

September 2022

1. LOCATION OF THE MINE SITE

The **GOLDEN PLANT** project is located 10 km from the mining town of Kolwezi, in the Democratic Republic of Congo.

Table 1



MINING TITLE

The GOLDEN PLANT deposit is a portion of the operating licence owned by GOLDEN. The site has an area of about 170 ha, which represents 2 cadastral squares.

RESOURCES AND RESERVES

The extent of the concession which includes the deposit is 2000 x 500 m, or 10 ha. This deposit is located in meta-sedimentary rocks of the Lower Roan unit of the Katanga System. It is one of many high-grade copper-cobalt deposits found in the Kolwezi flap where Lower Roan rocks are present in the form of discrete tectonic rafts and in sheared contact with the younger Kundelungu Series. The structure is complex, with long folds of East-West, North/West-South/East orientation, whose spine has been faulted, with sediments from the Roan that have overlapped younger rocks (Kundelungu).

Prospecting

The prospecting drilling of the GOLDEN PLANT deposit is planned for a total of 191,000m in the formations of the mining group.
However, GOLDEN has identified and inventoried mineral sources in the vicinity of the plant site in the Kolwezi mining area with world-class deposits.

Geological potential

The potential to run the plant at its highest capacity is estimated at 5,600,000 tonnes of copper and 560,000 tonnes of cobalt.
However, a potential attack has been identified from the artisanal exploitation of the deposits to be explored for a potential estimated at 16 million tons of ores grading 3.5% Cu and 0.3% Co or 560000 tCu content and 48000 tCo content.

Category	TONNAGE	%Cu	Urc	%Co	tCo
Measured					
Indicated					
Inferred	16,000,000	3,5	560,000	0,3	48,000
Total	16,000,000	3,5	560,000	0,3	48,000

2. GEOLOGY

2.1 REGIONAL GEOLOGY

The GOLDEN PLANT deposit is at the western end of the Katanga copper belt. This deposit is located in meta-sedimentary rocks of the Lower Roan unit of the Katanga System. It is one of many high-grade copper-cobalt deposits found in the Kolwezi flap where Lower Roan rocks are present in the form of discrete tectonic "rafts" and in sheared contact with the younger Kundelungu Series. The structure is complex, with long folds of East-West, North/West-South/East orientation, whose "spines" have been faulted, with sediments from the Roan that have overlapped younger rocks (Kundelungu).

The Lower Roan stratigraphy of the Kolwezi Zone differs from that of Zambia, in that the sediments are essentially clayey dolomites while the Zambian Lower Roan is more arena in nature. The base of the Lower Roan in the Kolwezi region has never been observed and no bedrock has been observed as a rock outcrop or in mining operations and boreholes.

The surface topography is generally flat, with thick or laterite soils limiting outcropping. Deep alteration has led to the oxidation of sulphides at depths of more than 200 metres, so mining operations to date have been carried out mainly by means of open-pit mining to exploit oxidized ores.

Metallogeny

The table below shows a stratigraphic column of the Katanga supergroup.

Table 2 Stratigraphic column of the Katanga supergroup

Age		Supergroup	Group	Formation and / or Lithologies
Pleistocene				
	Karro and Kalahari			
Upper Carboniferous				
+/- 650Ma				
		Upper Kundelungu	Plateaux	Arkoses with sandstones and shales
		(Ks)	(Ks-3)	
Proterozoic	Katangan		Kiubo	ks-2.2 / Ks2.1 :carbonated shales,sandy and
			(Ks-2)	argillaceous shales, pink limestones.
			Kalule	Ks-1.3/Ks-1.2: carbonated siltstones, sandy and
			(Ks-1)	argillaceous shales, pink limestones.
				Ks-1.1:diamictite 'petit congolmerat'
		Lower Kundelungu	Monwezi	arcosic sandstones with carbonated shales and
		(Ki)	(Ki-2)	siltstones.
			Likasi	Ki-1.3:carbonated shales and siltstones
			(Ki-1)	Ki-1.2:dolomites and limestones with shales
				Ki-1.1:diamictite 'grand conglomerat'
		Roan	Mwashya	R-4.2: dolomitic shales, with sandstones or
		®	(R-4)	carbonaceousshales at top; 'conglomorat de
				Mwashya' at the base.
				R-4.1:dolostones, dolomitic shales, jasper, oolites,
				pyroclasric units, iron.
			Dipeta	R-3.4/R-3.3:dolostones, limestones, shales,
			(R-3)	sandstones and arkoses
				R-3.2/R-3.1:dolomitic and argillaceous siltstones
				(R.G.S), dolostones.
			Mines	R-2.3:Kambove Dolomite (upper and lower)
			(R-2)	R-2.2:dolomitic shales (SDB, Bomz)
				R-2.1:kamoto dolomite (RSC, RSF, D.Strat,
				grey RAT
			RAT	R-1.3, R-1.2, R-1.1:argillaceous dolomitic siltstones
			(R-1)	and sandstones, base unknown.
+/- 1050Ma				Basal conglomerate
+/- 2050 Ma	Kibaran and Pre-Kibaran			

2.2 LOCAL GEOLOGY

In the past, exploration of the GOLDEN PLANT deposit unit was restricted to surface mapping and the drilling of 6 boreholes by the Union Minière du Haut Katanga, between 1938 and 1950.

These boreholes were drilled west of the GOLDEN PLANT deposit and are therefore not considered relevant.

Exploration drilling has identified a large deposit of copper and cobalt, immediately north of Kolwezi (GOLDEN PLANT deposit). This deposit outcrops and is bordered by faults with a pronounced south dip (70°). The GOLDEN PLANT deposit represents a distinct, well-mineralized "raft" or "megabreccia" fragment that has entered tectonically in juxtaposition with the mapped "domain" of MANGA West which is little mineralized.

General

The GOLDEN PLANT deposit is a scale of the Groupe des Mines oriented substantially W-E located in the heart of the Kolwezi aquifer. The deposit constitutes the eastern part of the MANGA deposit which extends on the surface over a length of 7 km.

The structure of the deposit

The deposit consists of two heart-shaped synclines-oriented W-E to SW-NE in the eastern part, separated by a faulted anticline axis which presents overlapping structures in places.

The structure is well defined in the center of the first cadastral square after the Musonoie River (figures in appendix: surface plans and section). At this level, the depth of the scales reaches 700 meters. The northern flank of the North Syncline (N.R.) is continuous at the surface along the entire length of the deposit. The southern flank of the southern syncline is buried under a cover of 50 to 100 m depth and also has outcropping fragments on the surface (T.17 on the plan).

Mineralization

Cuprocobalt mineralization exists in the Lower Ribs (grey rats, D strats, RSFs), CSRs and SDBs. It sometimes goes back to the BOMZ.
The oxidized ore is composed mainly of malachite and heterogenite which come from the alteration of sulphides (chalcosine, carrolite and bornite). This oxidized mineralization sometimes manifests itself to the bottom of the scales. Indications of sulphide ore intersected by boreholes start from 200 m (details will be provided when more holes are completed).

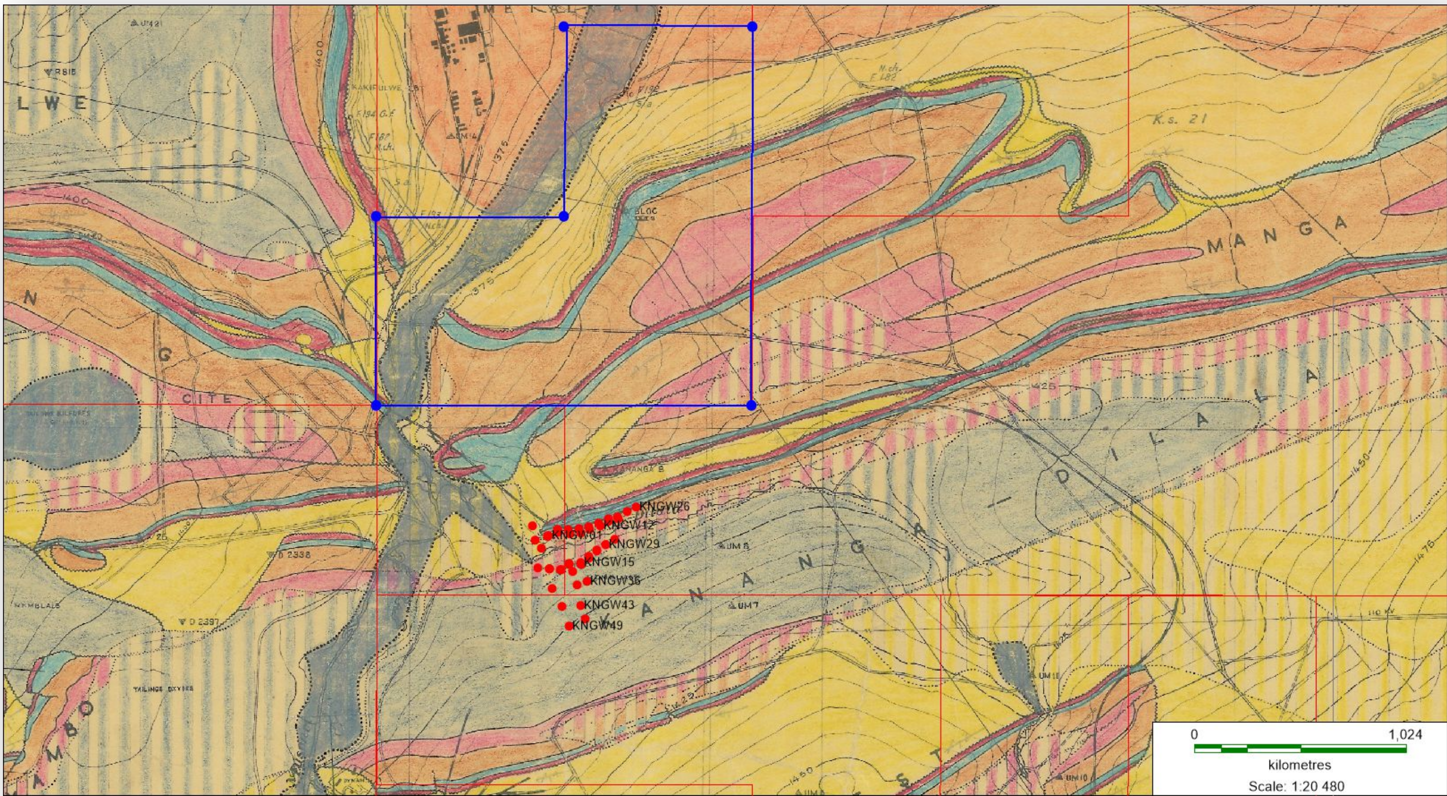
The table below illustrates the grades observed in the historical boreholes, grouped by cuts.

SECTION	N° SOND	PASSES		PASSE	Cu	Co	FORMATIONS GEOL0GIQUES
		DE	A				
X-500	U12	173,70	180,50	6,80	5,50		SD
		230,70	255,00	24,30	4,50		RSC
	U9	116,00	123,50	7,50	6,00		SD
		145,00	151,00	6,00	4,40		RSF
		167,50	176,00	8,50	3,90		RSF
		196,50	204,00	7,50	4,70		SD
X - 650	U20	395,80	420,50	24,70	4,60	0,10	SDB/RSC/RSF
		675,80	688,70	12,90	5,00	0,60	SD
		688,70	691,50	2,80	1,30	1,90	RSC
	MU 265	574,40	583,40	9,00	2,60	0,15	SDB/RSC
		590,70	611,00	20,30	1,60	0,50	RSC/RSF/D ST
	MU 288	167,40	171,00	3,60	4,20	0,20	SDB
		187,05	206,30	19,25	5,02	0,42	INF
		211,62	223,05	11,43	3,60	0,30	RSC/SDB
		229,50	232,60	3,10	1,40	0,30	SDB
	MU 267	636,60	700,10	63,50	6,10	0,40	SDS
		721,95	735,00	13,05	3,30	0,90	INF
	MU 264	103,00	115,00	12,00		0,50	RSC
		131,00	145,20	14,20		1,40	INF
X - 1050	MU 287	213,00	221,30	8,30	1,80	0,20	SDB
		221,30	233,30	12,00	5,80	0,60	SDB
		262,10	264,00	1,90	8,50	0,30	RSF
		269,00	272,00	3,00	1,00	0,30	RSF
X - 1250	MU 329	172,70	175,70	3,00	3,00	0,20	Br de RAT
X - 1450	MU 337	99,40	105,50	6,10	5,90	0,10	Br de RAT

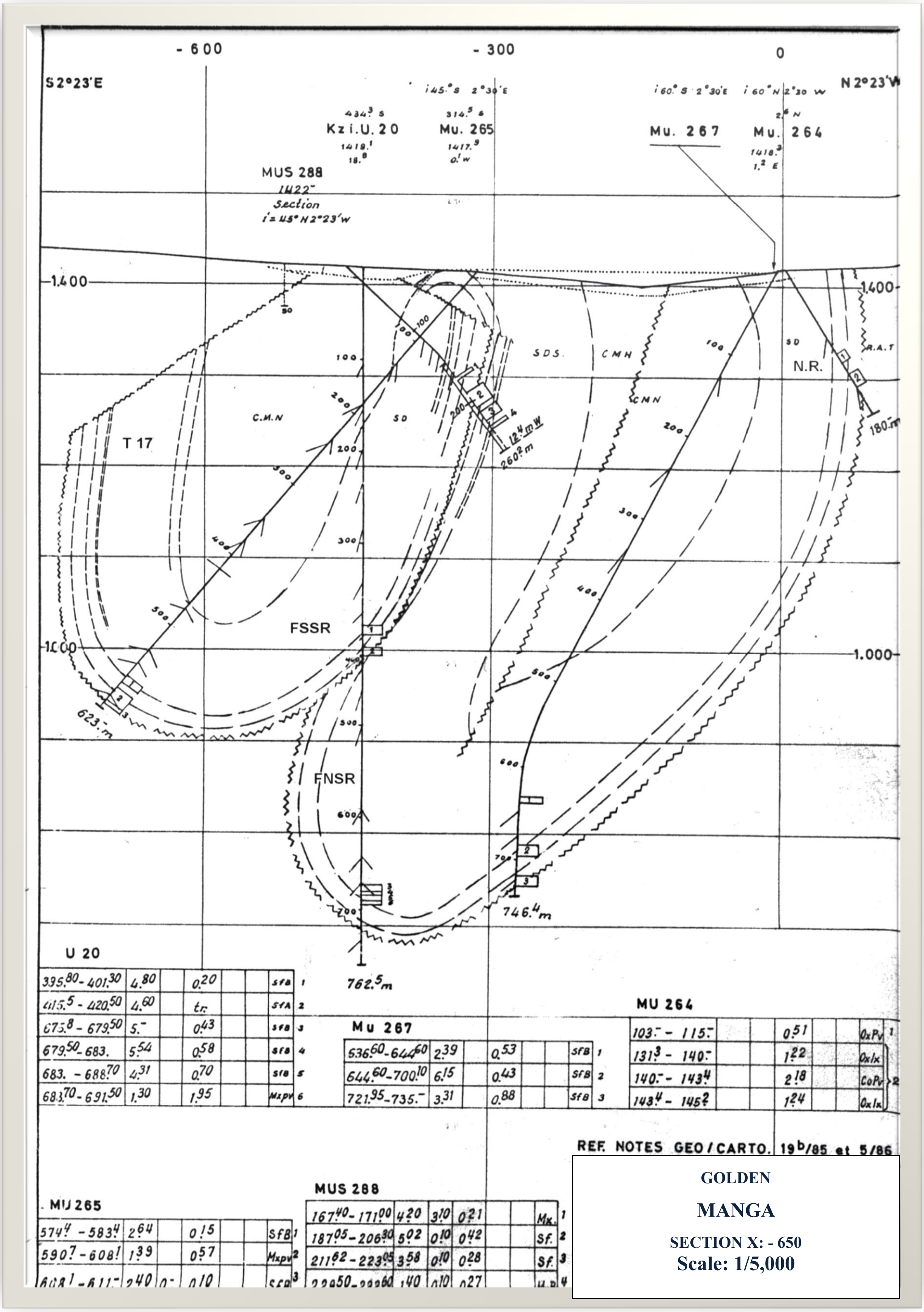
1. POTENTIAL

The surface plan below delimits 4 squares with a potential of **5,676,480 TCu**

PLAN VIEW OF THE COPPER and COBALT DEPOSIT (4 CARRES)



X-650 CUT CUPRO – COBALTIFERE DEPOSIT IN THE SURROUNDING AREA



The mineralization is typical of the Kamoto and KOV mines, with two high-grade zones (the RSF unit: 14 m thick and the SDB unit: 12 m thick), separated by the less mineralized RSC unit: 20 m thick.

The mineralized zones consist of talcous shales, breccias and silstones that are part of the metasedimentary sequence. Copper and cobalt mineralization generally occurs as malachite and heterogenite/kolwezite in the oxidized portion of the deposit and chalcosite, bornite and carollite in the deeper sulphide zone. The oxidation depth is about 200 m from the surface.

The levels of arsenic, bismuth, selenium and uranium are considered very low enough not to pose environmental or metallurgical problems.

2.3 MINERAL RESOURCES ESTIMATION

Drilling in the GOLDEN PLANT deposit was planned along grid lines 100 m apart and perpendicular to the direction of the deposit. Drill holes are provided at intervals of 100 x 100 m along the grid grids.

To start this project, mineral resources have been identified and inventoried in the project area and will come from the two main sources:

- Artisanal production
- The exploitation of the MANGA deposit

The target geological potential. is 5,600,000 tonnes of copper and 560,000 tonnes of cobalt. However, the inventory has been made in such a way as to guarantee a production of artisanal source of the magnitude of 560000tCu with 48000tCo.
The life of the mine will be increased with drilling to be carried out in the MANGA deposit to make a potential of 5.6 million tons of copper metal.

Category	TONNAGE	%Cu	Urc	%Co	tCo
Measured					
Indicated					
Inferred	16,000,000	3,5	560,000	0,3	48,000
Total	16,000,000	3,5	560,000	0,3	48,000

3. MINING

3.1 GENERAL

The study consists of providing field results, laboratory tests and mining parameters determined by the Golden Mining Design Office (EMI) to allow the exploitation of the GOLDEN PLANT deposit in compliance with technical standards. The ores extracted will feed the new metallurgical processing plants that will be located not far from the mine, at a distance of about 6 km, for the production of cathode copper and a cobalt salt. The metallurgical processing capacity is progressive: from 100,000 tCu/year to a maximum of 200,000 tCu/year.

The study provides for preparatory work (access, work platforms,), free discovery and the extraction of minerals. The essential steps are as follows:

- Geotechnical study;
- Choice of operating method;
- Mine optimization using SURPAC software;
- Determination of mineral reserves;
- Definition of mining operations;
- The determination of the fleet of mining machines;
- Planning of mining excavations;
- Determination of staff numbers;
- Estimated mining investment.

3.2 GEOTECHNICAL STUDIES

The aim is to determine the geomechanical properties of the rocks on theGOLDEN PLANT deposit site, in order to identify the parameters of the mining structure essential to its safety and stability.

The results of the tests carried out will inform the development of the mine design, by determining the angles of the liquidation embankments of the open pit mine, the dimensions of the inclined ramps, benches and operating bleachers. The essential data of this work are: the resistance of the rocks to compression, tensile strength, internal friction angle, cohesion, degree of cracking and weathering, etc.

3.2.1 Sample preparation

The samples submitted for testing are cylindrical cores obtained during core drilling campaigns. In this campaign, there was a succession of geological surveys and geotechnical surveys (12 in total) whose samples were sent to the Laboratory of Rock Mechanics of the Golden.

The samples to be tested are cylindrical cores generally obtained during core drilling campaigns. It has become commonplace, to reduce costs, to combine the geological and geotechnical survey to provide samples to the Soil and Rock Mechanics laboratory.

3.2.2 Laboratory equipment

a) Drilling

The samples collected on site in the form of multiform blocks are sent to the laboratory where a drill equipped with a corer and a vice is used which, following the combination of the rotational movement of the corer and the vertical translational movement of the vice, makes it possible to obtain cylindrical cores. A circuit facilitating the circulation of water is mounted to evacuate the cuttings (rejects) and cool the bit. There is the triple tube corer of respective diameters: , and .56 mm42 mm36 mm

b) Core cutting Saw

Cores brought to the laboratory have an equal maximum length for a diameter D = 42 mm. They shall be reduced to the dimensions required for testing. Cores should be cut by means of an electric saw. In the case of cylindrical samples, the IBG (Internatinale Buro Fur Gerbirgsmechanik) Standards recommend using samples with diameter D = 42mm and height H = D or H = 2D, depending on the type of test. But diameters of 40 to 50mm are tolerated.153 mm

c) Polisher

The polishing machine makes it possible to achieve the flatness of the faces, their parallelism and the perpendicularity with respect to the axis of the cylinder. These are the conditions to be respected absolutely before each test for fear of negatively influencing the results with friction between carrots and press trays.

d) Hydraulic Press

Hydraulic presses are required for compression and tensile testing on rock samples. There are three types of presses: 10, 200 and 300 tons.

3.2.3 Test works

a) Simplecompression ssai (uniaxial)

This is the most common test in Rock Mechanics. It provides an interesting amount of information about the rock. These are:

- compressive tensile strength
- the static modulus of elasticity Es (YOUNG modulus)
- the coefficient of POISSON.

The machine intended to carry out this test is a Press. There are currently three presses at the EMI Laboratory: a 10-ton press, another 200-ton press and the largest, the 300-ton press.

b) Compressive tensile strength

The charge P for which appears the fracture of the rock relative to the surface S of the cross-section of the sample, measured in the plane perpendicular to the direction of the load, is equivalent to the compressive strength of the rock, it is noted:

$$\sigma_c = w/s \quad \text{in kg/cm}^2$$

N.B: There are some corrections to be made to this result according to the value of the diameter (cylindrical sample) or the D/H ratio).

c) Young's elasticity module : Es

During the simple compression test, there is a recorder (Hewlett Packard) that allows the sample deformation process to be measured and recorded in a load (Y in Kg/cm²) – strain (X in mm/m) diagram.

The modulus of elasticity is obtained at 50% of the breaking load or at the tangency point.

$$\mathbf{tg\ \alpha = Es = \sigma c / \varepsilon}$$

With εdeformation.

d) Coefficient of Poisson

By applying the load P that compresses the sample, it results in transverse expansion and longitudinal shortening which are both proportional to the load applied.

An XY potentiometric recorder connected to the strain gauges of the compressive rock allows us to obtain the diagram of lateral (εt) and axial (εl) deformations.

The Poisson coefficient **ν** is the ratio of lateral deformation to axial deformation, it is a characteristic of the rock.

$$\mathbf{\nu = - \varepsilon t / \varepsilon l}$$

e) Tensile test

Tensile strength ranks among the most important mechanical properties in mining work.
Indeed, most breakages are caused by insufficient tensile strength. According to the literature, it is five to six times lower than compressive strength.

The rock is very resistant to tensile forces. Tensile strength was determined in the laboratory by an indirect method called Brazilian Test. This test is carried out on cylindrical samples packed parallel to the generator between the press trays which are compressed until the rock breaks. The tensile strength is given by the formula:

$$\mathbf{\sigma t = - 2P / DH\pi} \qquad \text{in Kg/cm}^2$$

With: - P: the breaking load
- D: diameter of the sample
- H: sample height
- σt: tensile strength
- π : 3,14
- The samples used for this test have the dimensions: D = H.

f) Internal friction angle

Knowing the tensile strength and the single uniaxial compressive strength, it is necessary to determine approximately the internal friction angle of the rock.

The internal angle of friction is that made by the stress axis with the tangent to the intrinsic curve for the region near the origin of the axes.

g) Cohesion

Cohesion reflects a force that binds together the particles (grains) of a material. On the graph, it is read on the y-axis from the origin to the intersection with the tangent to the intrinsic curve.

$$\tau = c + \sigma \, tg \, \varphi$$

h) Density measurement

Density measurements were made using the Mettler P3N precision balance for weighing samples using Archimedes' principle: "Any body immersed in a fluid undergoes vertical thrust directed from bottom to top equal to the weight of the fluid being moved."

The density is obtained by the ratio of the weight of the sample (We) to the volume of the fluid corresponding to the immersion weight (Wi).

3.2.4 Results

Table 1: Densities

Rock	No. sample	D Cm	L Cm	We g	Wi g	Density
D. STRAT Mineralized	1/1	4,20	4,18	149,0	92,0	2,61
	1 /2	4,20	8,45	302,0	186,0	2 ,60
	1 /3	4,20	8,40	301,0	185,0	2,59
	1/4	4,20	8,45	301,5	185,0	2,59
SDB sterile	2/1	4,15	8,44	238,5	126,0	2,12
	2/2	4,15	4,22	115,5	61,0	2,12
	2/3	4,15	8,33	235,5	124,0	2,11
	2/4	4,15	4,16	117,0	61,5	2,11
	2/5	4,15	4,13	117,0	62,0	2,13
SDB	3/1	4,20	8,39	276	162,0	2,42
	3/2	4,20	4,24	143,5	86,0	2,5
	3/3	4,19	411	137,0	81,0	2,45
RSC Mineralized	4/1	4,20	8,49	285,0	170,0	2,48
	4/2	4,20	4,27	137,0	51,0	2,45
D.STRAT Mineralized	5/1	4,20	4,20	123,0	67,5	2,22
	5/2	4,20	4,26	125,0	69,0	2,23
	5/3	4,20	4,28	124,0	70,0	2,30
	5/4	4,20	4,23	104,0	54,5	2,10
D.STRAT Mineralized	7/1	4,20	8,47	259,0	150,0	2,38
	7/2	4,20	4,29	138,0	81,0	2,42
	7/3	4,20	4,23	125	71,5	2,34
	7/4	4,20	8,46	263,0	151,0	2,35
	7/5	4,20	4,27	112,0	62,0	2,24

Table 2: Brazilian Trials

Rock	No. sample	D Cm	L Cm	SL Cm²	W Kg	σt Kg / Cm²
D.STRAT min	1/1	4,20	4,18	55,15	5375,0	195
SDB sterile	2/2	4,20	4,22	55,68	675,0	24
	2/4	4,15	4,16	54,24	720,0	27
	2/5	4,15	4,13	53,85	86,0	32
SDB	3/2	4,20	4,24	55,95	2090,0	75
	3/3	4,19	4,11	54,10	1075,0	40
Ore CSR	4/2	4,20	4,27	56,34	2225,0	79
D. STRAT ore	5/1	4,20	4,20	55,42	2200,0	79
	5/2	4,20	4,26	56,21	3450,0	123
	5/3	4,20	4,28	56,47	2250,0	80
	5/4	4,20	4,23	55,81	650,0	23
D. STRAT ore	7/2	4,20	4,29	56,61	2300,0	41
	7/3	4,20	4,23	55,81	1475,0	53
	7/5	4,20	4,27	56,34	800,0	28

Table 3: Simple Compression

Rock	No. sample	D Cm	S Cm²	W Kg	σc Kg / Cm²	Are T / Cm²	ν
D. STRAT ore	1/2	4,20	13,85	14500	1047	711	0,12
	1/3	4,20	13,85	10250	740	484	-
	1/4	4,20	13,85	14250	1029	546	0,15
SDB ore	2/1	4,15	13,53	3900	288	41	0,24
	2/3	4,15	13,53	2525	187	45	0,18
SDB	3/1	4,20	13,85	3200	231	83	-
RSC	4/1	4,20	13,85	11200	808	-	-
D. STRAT m	7/1	4,20	13,85	6300	455	461	0,20
	7/4	4,20	13,85	6200	448	784	0,32

N.B. : The shear plane having passed through the gauge train contacts, there are no readings of the deformations (Es and ν)

Table 4 :Statistical services

Rock	Density	σt Kg / Cm²	σc Kg / Cm²	Are T / Cm²	ν
D. STRAT mineralized	2,60	195	939	580	0,14
SDB Non-mineralized	2,12	28	238	43	0,21
SDB	2,46	58	231	83	-
mineralized RSC	2,47	79	808	-	_
D. STRAT Mineralized	2,21	76	-	-	_
D. STRAT mineralized	2,35	41	452	623	0,26

Table 5: Average Values

Rock	σt Kg / Cm²	σc Kg / Cm²	C Kg / Cm²	φ Degree	τ Kg / Cm²
D. STRAT mineralized	195	939	110	41	350
SDB(sterile)	28	238	41	52	73
SDB	58	231	58	36	175
mineralized RSC	79	808	124	55	226
D. STRAT mineralized	41	452	68	58	124

3.3 MINE OPTIMIZATION

The project is planned to be implemented using the interlocking pit open-pit mining method. The optimization was carried out with the SURPAC 6.3 software, using the parameters below to achieve an optimized design.

Parameters	Securities
Liquidation slope angle	37°
Bleacher height	10m or 2 x 5m
Inclined width	17- 18m
Slope of the inclined	8%
Mine depth	200 m
Maximum expansion North – South East – West	 400 m 7000 m
Temperament	2,5

3.4 MINERAL RESERVES (Phase 1)

RESOURCE	TONNAGE	%Cu	tCu	%Co	tCo
Measured	111 000 000	3,15	3 500 000	0,27	299 700

3.5 MINE PLANNING

See table in the economic model

3.6 DETERMINATION OF THE MINING MACHINERY FLEET

a) Standard distance

Given the shape of the mine, the overall distance calculated is 2.5Kmst including 2.6 Kmst for waste rock embankments and 2.4 Kmst for ore backfills. This standard short distance will reduce the cycle time of transport units, thus improving the hourly efficiency of these machines.

b) Mining machinery fleet

The fleet of mining machines is determined for control of their respective category, capacity and work yields. Mining operations, from excavation to backfilling, are to be subcontracted by a contractor. This fleet includes the workshop of primary production machinery:

Production/month x 10^3 (m^3)	Capacity	400	500	700	800	915	1.500
Sounders	4" Diam	3	3	4	4	4	5
Shovels	4.6 m^3	3	3	4	4	5	6
Skips	40 T	13	15	16	20	23	40
Loaders	4.6 m^3	2	2	3	4	4	6
Dozers	D9R	2	2	2	2	3	4
Graders	140K	2	2	2	2	3	3
Sprinklers	30 m^3	2	2	2	2	2	3
Compactor	530D	1	1	1	1	1	1

3.7 EXHAURE

The average annual rainfall in the vicinity of GOLDEN PLANT is of the order of 1,240 mm with extremes of 720 and 1,770 mm. The height of water likely to fall on a day of heavy rain can reach 60mm or even 80mm. As for the topography of the site, it is relatively flat in nature, the contribution of water outside the mine will be less, being easily evacuated by a drain surrounding the mine. Water falling directly into the mine can be discharged either naturally by infiltration or by means of a few pumps placed in a sump of capacity calculated as follows:

Area of the drainage basin: 900 x 1300 = 1.170.000 m^2
Critical rush of a day:50 mm
Infiltration rate: 50% (derived from GOLDEN/SKM mine statistics in weathering areas).
Sump capacity: 0.05 x 1.170.000 x 0.5= 29 250 m^3

The maximum volume of water to drain to the sump would be of the order of $\pm$ 30,000m^3. We propose the digging of a sump with a capacity of about 40,000m^3 which is sufficient to shelter the levels of work.
Four Flyght 2400 pumps of 200 m^3/h or equivalent will be sufficient to ensure the secondary dewatering.

3.8 LABOUR REQUIREMENT

N°	LIBELLE	CATEGORIE MO	EFFECTIF PAR POSTE	NOMBRE DE POSTES DE TRAVAIL	NOMBRE D'ENGINS	COEFF. D' ABSENTEISME	EFFECTIF TOTAL
	SUPERVISION						
1	ingénieur chef de sce	MOCA 2	2	1		1	2
2	conducteur EXPL	MOCA 1	2	1		1	2
3	conducteur DEM	MOCA 1	2	1		1	2
4	chef de poste EXPL	MOCA 1	2	3		1	6
5	chef de poste DEM	MOCA 1	2	3		1	6
6	chefs chantiers	MOE CL4	4	3		1	12
			SOUS/TOTAL SUPERVISION				30
	OPERATEURS ENGINS						
1	pelles	MOE CL5-8	1	3	7	1,2	25
2	bennes	MOE CL5-8	1	3	40	1,2	144
3	dozers	MOE CL5-8	1	3	5	1,2	18
4	graders	MOE CL5-8	1	3	5	1,2	18
5	sondeuses	MOE CL5-8	1	3	6	1,2	22
6	chargeuse	MOE CL5-8	1	3	6	1,2	22
7	arroseuses	MOE CL5-8	1	3	6	1,2	22
			SOUS/TOTAL OPERATEURS				270
	DISPATCHING						
8	dispatchers	MOE CL5-8	1	3	1	1,2	4
9	camion trans personne	MOE CL5-8	2	3	1	1,2	7
10	camionnette de surv.	MOE CL5-8	2	3	1	1,2	7
11	employé de bureau	MOE CL5-8	1	3	1	1	3
12	amion Bennes (Reprise	MOE CL5-8	1	2	14	1	28
			SOUS/TOTAL DISPATCHING				49
	MAINTENANCE						
12	mécaniciens dépannage	MOE CL5-8	8	3	1	1,2	29
13	mécaniciens réparation	MOE CL5-8	12	1	1	1,2	14
14	électriciens dépannage	MOE CL5-8	8	3	1	1,2	29
15	électriciens réparation	MOE CL5-8	12	1	1	1,2	14
16	agent planning DEM	MOE CL5-8	4	1	1	1,2	5
			SOUS/TOTAL MAINTENANCE				91
	MINAGE						
17	boute-feu	MOE CL5-8	5	1	1	1,2	6
18	aide boute-feu	MOE CL5-8	8	1	1	1,2	10
19	camion explosif	MOE CL5-8	2	1	1	1,2	2
			SOUS/TOTAL MAINTENANCE				18
	STATISTIQUES						
20	employés statistiques	MOE CL5-8	4	1	1	1	4
			SOUS/TOTAL STATISTIQUES				4
	EFFECTIFS TOTAUX						462

4. METALLURGICAL TREATMENT

Table 6.1: Plant feed and expected metal production

Main plant parameters

Annual feed in tonne/year 4 450 000
Available days days/year 365
Annual hours of availability hours/year 8,760
Hours of plant operation hours per year 8,322
Provision of the plant 95%
Hourly plant supply (dry tons) tonnes/hour 535
Leaching efficiency Copper (Cu soluble) 94%
Cobalt leaching yield (total Co) 72%
Overall copper yield (Cu soluble) 92%
Cobalt leaching yield (total Co 68%
Nominal copper production (tonnes) /year (five years) 100 000
Nominal production of cobalt tonnes contained)/year (five years) 9,000
Sulphide ore supplement in tonne/year 3,412,800
Efficiency Cu concentrator 91%
Yield Cu plantallurgical 90%
Efficiency Co concentrator 75%
Yield Co factory 75%
Production Cu tonne/year after five years 200 000
Cobalt production tonne/year after five years 14,800

4.1 PROJECT DESCRIPTION AND TREATMENT PLANT

The first phase of the GOLDEN PLANT project will treat the oxide part of the ores to produce Copper and Cobalt and will consist of the following processing circuits.

- Crushing of minerals all coming
- Grinding
- thickening of the leaching feed
- Production of sulfuric acid
- Leaching
- Primary settling with clarification of the impregnating solution.
- Backwashing
- Solvent extraction of copper
- Copper electrowinning
- The waste storage site
- The elimination of Iron and Manganese.
- Production of cobalt salts.
- The lime milk production module

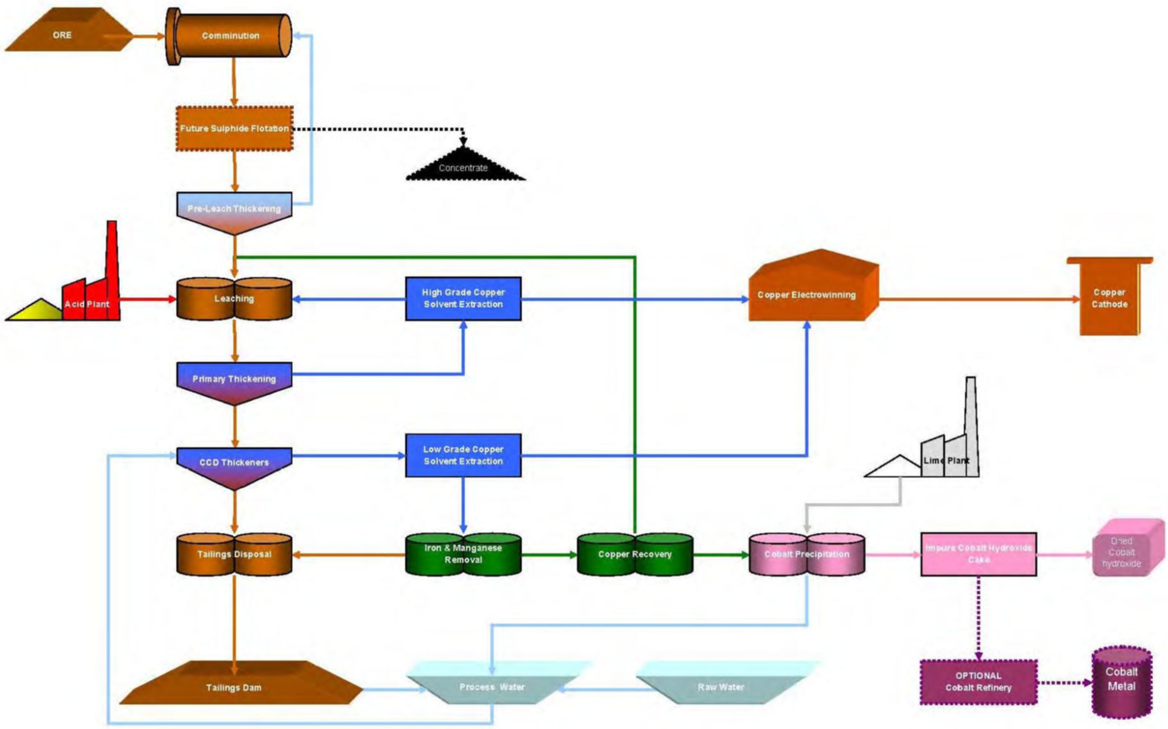
The processing circuit above is shown in the flow sheet of the figure below. The second phase of the project will involve the addition of sulphide ore treatment either by concentration followed by solid fluorescent treatment or by autoclave leaching of sulphide concentrates.

This second phase will require investment for studies, construction:

- a concentrator for mixed ores and sulphides
- A leaching module in autoclave and / or alternatively a fluorescent-solid.
- additional settling with backwashing of less importance because
  Dealing with concentrates.
- An additional tank house of 100,000 t/year of copper.

The project will thus move to a production of 200,000 t/year by the fifth year of production. A substantial reduction in the production cost will then occur with the reduction of acid requirements with the commissioning of the leaching of sulphide concentrates. Indeed, an additional 154,000 tn of acid can come from this route and will reduce the call of fresh acid by the same amount.

FLOW-SHEET OF THE PROCESSING PLANT



4.2 Crushing

The ore from the mine (ROM) is fed by skips to the embankment located at the metallurgical processing plant. From the backfill, the ore is fed into a static screen for particle size separation. Blocks larger than 500mm are sent in a rock breeze for reduction of dimensions. The part less than 500 mm feeds a primary crusher and the crushed ores are unloaded a conveyor which conveys them to grinding. On this conveyor a drop plate is installed as a bypass to load the ores into the primary crusher in case of a prolonged stop on the crushing circuit.

4.3 Grinding

From the milling feed stock, the ore is fed into one or two semi-autogenous SAG mills mills. The Under flow of hydro cyclones is also replenished at SAG mills. To speed up the grinding of ores, pellets are fed into the mills.
The discharge of the mills is done through the crusher screen in the sump. The cleaning water is manually applied to the discharge screen to wash away the fines of the coarse particles. The coarse particles are recovered in a suitable compartment from where they are recovered to be recharged to the mill. The passers-by are sent to the cyclones to separate the under flow that returns to the crusher and the overflow that feeds the thickener before leaching.

4.4 The thickener before leaching.

The overflow of the hydrocyclone is pumped to the thickener before leaching. To accelerate the thickening of the flocculent is added to the inlet of this decanter. The thickened pulp is stored in the leaching feed tank while the water from the overflow is recovered for the grinding circuit.

4.5 The sulfuric acid plant

A sulphuric acid plant from the combustion of sulphur produces acid necessary for the metallurgical processing of ores. In addition, this plant will produce sulfur dioxide necessary for the reduction of trivalent cobalt during leaching.

4.6 Leaching

The underflow of the thickener before leaching is pumped to the lix through distribution boxes where it is mixed with fresh sulfuric acid, SX raffinate and pulp returning from the cobalt plant de-sinking. The leaching is done in six mechanically agitated cascading tanks, fed at their heads while the discharge is done in a channel. A manual bypass is provided to allow each tank to pass in case of cleaning. The sulphur dioxide is bubbled below the lower agitator to reduce the trivalent cobalt and pass it into solution. Desanding pumps are provided on each tank to prevent the accumulation of coarse particles in the leaching tank. The sulphur dioxide is bubbled below the lower agitator. After leaching the pulp is pumped to the primary decanter.

4.7 Primary decanter and impregnating solution clarifier.

The discharge of the leaching tanks is pumped to the feed sheet of the primary decanter where it is mixed with the under flow of the clarifier and flocculent. The flocculent is diluted with the overflow of the primary decanter before being added to the feed channel. The residue from the primary decanter feeds the counter-current decanters while the overflow is pumped to the clarifier.

4.8 Solvent extraction of copper

The solvent extraction of copper will be done in two or three stages Extraction with washing of the organic, two stages of stripping. The extraction will be done against the current with an organic ^phase containing copper extractant in a diluent with low aromatic content. Copper is selectively extracted by the organic by giving an equivalent amount of acid that enriches the raffinate. The raffinate is then stored in the raffinate pond from where it is pumped for leaching. Some of the raffinate is bled for impurity control. The organic loaded is put in contact with the electrolyte coming from the electrolysis room to strip the copper that has been charged there. The organic that has been released from its cooking is stored in the organic tank where an addition of fresh extractant is made to compensate for the losses. The enriched electrolyte is pumped to the electrolysis room.

4.9 Copper Electrolysis

The enriched solution is sent to the electrolysis room where it feeds the scavenging tanks to clean the incoming organic and then the commercial electrolysis tanks (140 tanks). Current rectifier transformers supply these tanks with current and ensure the deposition of copper on stainless steel cathodes from which the copper is stripped after a cycle of about six days.
High grade copper cathodes are thus produced.
The copper is thus stripped stainless steel blanks, arranged in packages and circles for shipment for sale.

4.10 The cobalt plant

The solution from counter-current scrubber settlers feeds solvent extraction into its low-grade circuit. The copper is removed from the solution leaving the cobalt and other impurities in the raffinate that feeds the cobalt plant. This solution first undergoes steps to remove these impurities before precipitation of cobalt in the form of hydroxides.

4.10.1 Removal of iron, aluminum, and manganese

The pH of the acid solutions from solvent extraction is raised to 3.9 by adding crushed limestone milk to precipitate iron and aluminum. In addition, the pulp is bubbled with an oxidizing mixture of sulfur dioxide and oxygen to oxidize manganese and precipitate it in the form of pyrolusite.

4.10.2 Copper Removal

The residual copper that has not been extracted at solvent extraction is then precipitated in the form of hydroxide by the addition of lime which raises the pH to about 5.2. The copper precipitate is returned to leaching to reduce losses and improve yield. The clear solution after settling is sent to the precipitation of cobalt which is done in two stages to ensure the quality of the hydroxide produced. The precipitate of the first precipitation is the product while that of the second precipitation is recycled at the iron removal stage.

4.10.3 Precipitation of cobalt

The precipitation of cobalt will take place in two stages to ensure the quality of the cobalt produced. Indeed, the first precipitation of cobalt will be at pH 7.8 and will produce a hydroxide of good quality saleable while the second precipitation which will be done at pH 8.2-8.3 will produce after exhaustion of the solution a polluted hydroxide which will be recycled at the iron and aluminum removal step.

4.11 Related project services

Provision shall be made for the following related services and infrastructure:

- Emergency diesel generator
- Steam boiler for leaching, cobalt plant and acid plant (start-up)
- drinking water as well as raw and demineralized water for the process and water for the anti-fire vents
- The treatment of wastewater from the plant.
- Compressed air
- Rainwater harvesting
- water collection basins
- dump for non-toxic litter
- Camp for the construction of the factory
- Barbed wire fence of the factory
- Reagent storage stores and maintenance workshops
- Administrative offices.
- Analytical laboratories.

5. INFRASTRUCTURE

a) Energy

The operations will have to be carried out using diesel equipment. As it is planned a work organization with 3 shifts, to illuminate the site, there will be a need for electrical energy for the dewatering system, the lighting of the mine site and supply some workshops on the site.

b) Workshops and Offices

A few workshops are planned on the mine site, in order to provide for urgent repairs. The offices will be built to allow direct management of mining operations.

6. ENVIRONMENT

6.1 INTRODUCTION

Under the provisions of the DRC Mining Code and Regulations, all mining operations must be subject to a feasibility study as well as an Environmental Impact Assessment (EIA) and an Environmental Project Management Plan (PGEP).

It is in this context that this document is prepared on the part devoted to the environment relating to the project for the development of the GOLDEN PLANT deposit, located about 3 km from the mining town of Kolwezi, in the Lualaba district of Katanga province.

The direct and indirect benefits described below are expected to result from the development of the GOLDEN PLANT development project:

- Generation of taxes and fees for the benefit of the public treasury;
- Participation in the revitalization of the city of Kolwezi in particular and the province of Katanga in general;
- Support for local industry;
- Contribution to the creation of a market for local goods and services;
- Creation of new jobs;
- General improvement of the standard of living of local populations;
- Contribution to the improvement of basic infrastructure (roads, drinking water network, sanitation and communications infrastructure);

6.2 GOLDEN PLANT MINING PROJECT POLICIES

GOLDEN Sarl undertakes to respect throughout its operating work:

- Legal provisions on environmental protection, health and safety of workers;
- Culture as well as local customs and to ensure sustainable development for the surrounding communities.

6.3 ENVIRONMENTAL STUDIES

An Environmental Impact Assessment (EIA) and an Environmental Project Management Plan (PGEP) will be developed on the GOLDEN PLANT deposit development project by an approved environmental study office, in accordance with the requirements of the DRC Mining Code and Regulations, in order to identify the impacts that will be generated by the said project and also to propose appropriate mitigation and rehabilitation measures.

6.4 IMPACTS OF THE PROJECT ON THE ENVIRONMENT

The GOLDEN PLANT deposit development project will generate the following environmental impacts:

- Modification of the landscape following prospecting, excavation, construction of waste rock piles and tailings ponds;
- Modification of soil quality;
- Deterioration of groundwater and surface water quality due to effluents from waste rock piles and ore storage piles, occasional discharges of wastewater during heavy rainfall and accidental spills;

- Dust due to vehicle and machine traffic that can have a negative impact on air quality;
- Emission of greenhouse gases that can contribute to global warming;
- Changes in air quality that may negatively affect crops and natural plants in the project area;
- Increased noise and vibration levels at the mine and along access roads due to vehicular and machine traffic, mining and transportation of waste rock and ore, and ore processing facilities;
- Considerable increase in vehicle traffic flows on the access roads to the mine;
- The mine's tailings facility, processing plant and other project infrastructure will alter the aesthetic appearance of the site;
- Disturbance of the flora following the development of infrastructures and the exploitation of the deposit;
- Significant human migration to the concession that could accentuate the destruction of forests for agricultural and charcoal production;
- Loss of habitats for the few animals still present on the perimeter.

6.5 MITIGATION MEASURES

- Use of best management practices during construction and operation to mitigate impacts on soils;
- Implementation of erosion control measures;
- The mine's water needs will be met by an adequate long-term groundwater supply;
- Spatial minimization of the scope of the project;
- Process water recycling;
- Use of runoff from waste rock piles and ore backfills at the wastewater treatment plant;
- Coating of the tailings facility with a geomembrane to minimize infiltration;
- Roads will be improved and widened to accommodate increased traffic
- Maintaining noise and vibration levels below the guideline limits applicable in the nearest communities during operation;
- Placement of safety signs in suitably selected locations to indicate potential hazards;
- Compliance with the company's own safety instructions, drawn up on the basis of government orders relating to the various trades;
- Installation of fire extinguishers in suitably targeted areas;
- Training of personnel in the use of fire extinguishers;
- Regular watering on work areas and their surroundings as well as limiting the speed of circulation of machines and other vehicles in order to avoid or minimize dust heaves;
- Provision of bins for the collection of solid waste;
- Respect for working hours;
- Personal protective equipment given to workers according to the requirements of their trades;
- Regular notification of accidents at work, followed by their analysis with a view to determining their causes and thus proposing preventive measures;

- Regular meetings of the various Health and Safety Committees, in accordance with the requirements of the law;
- Periodic medical check-up of persons assigned to positions where they are most exposed to the risk of occupational diseases;
- Organization of services providing first aid and care for agents in camps;
- Organization of a security service ensuring the guarding of the company's facilities.

7. ESTIMATED INVESTMENT COST

Capex Estimate

Earthworks	20,409,089
Concreting work	35,137,622
Metal structure	20,964,773
Walking feet and platform	20 947 043
Mechanical equipment	238 296 899
Piping and valves	25,363,402
Electricity and instrumentation	40 233 442
Construction buildings	1,105,000
Strategic equipment insurance	12,218,844
First refill and reagents	25,982,955
Total direct costs	**440,659,069**
Commissioning and representation of suppliers	5,995,630
% of Direct Costs EPCM	39 035 106
Contingency	21,014,343
Total indirect costs	**66,045,079**
Turnkey and unexpected acid plant	65,746,506
Construction Camp	20,624,946
Power supply	10 100 304
Tailings and stockpiles	22 057 832
Cost of hours worked and training of future workers	11 000 000

Total other costs 129,529,588

TOTAL PROJECT COST 636 233 736

8. OPERATING COST ESTIMATION (see economic model)

9. MARKET STUDY

9.1 Usefulness of copper

Copper is a non-ferrous closely related to human beings and is widely used in industrial fields such as electricity, electronics, mechanical engineering, construction and national defense. Second to silver, copper has the best electrical conductivity among all kinds of metals; Its performance in applications and its electrical conductivity justify its consumption. The consumption of copper and its alloys is second to that of steel, iron and aluminum. Copper is mainly used to produce electrical wires and cables and alloy for construction. It is also used for electrical parts of certain equipment, machinery and transport installations. Copper is used to make various kinds of bulletins, firearms and artillery as well as aircraft and boat heat exchangers, in the military field and to make bearings, pistons, switches, valves and high-pressure steam equipment. In addition, copper and copper alloy are also widely used in other fields such as hot processing technology, cooling devices and civil equipment.

9.2 Copper resource situation

Generally, the world has relatively rich copper resources and it is estimated that the Earth's copper resources are, worldwide, 3 billion tons and the copper resources in offshore nodules are 700 million tons. According to statistics from "United States Investigate (the Geologic Overview)", in 2012, the moderate copper reserves in the world in 2011 were 690 million tons, representing 23% of copper resources in the Earth. Based on total production, world production was 16.1 million tonnes in 2011. Statistics show a lifespan of existing reserves of 43 years.

Geographically, the distribution of the world's copper resource is not balanced; it is mainly concentrated in the west coast of North and South America, Central Africa, Central Asia and Siberia in Russia and secondly in the Alps and East-Central, the Southeastern zone of the USA, the Pacific Coast of the Southwest and islands.

Compared to countries, those with the largest copper reserves in the world include Chile and Peru, with 28% and 13% of global copper reserves, respectively. Other countries with relatively large reserves include Australia, Mexico, USA, China, Russia, Indonesia, Poland, Zambia, Canada and Kazakhstan.

Depending on the different geotectonic environment and geological conditions of deposits, significant industrial-type copper deposits include porphyry copper ore, sandstone (shale), pyrite copper ore, nickel-copper sulphide, skarn veins, and copper carbonate ore. The porphyry copper ore reserve takes the largest proportion of the world's total reserves, accounting for 53.5% and ranks first; sedimentary and sedimentary-metamorphic copper ores account for 31% of total reserves, in second place; the pyrite-copper volcanic rock reserve accounts for 9% of total reserves, ranking third; Reserves of magmatite, skarn and other types of copper ore account for only about 6.5% of the total.

9.3 International copper market

Currently, there are more than 50 copper concentrate producing countries in the world and over the past ten years, global copper production has continuously increased. In the second half of 2008, as affected by the global financial crisis, the price of non-ferrous metal fell sharply; And some mines reduced their production and world copper production was 15.591 million tons, slightly less than in 2007. But since 2009, copper production has gradually resumed. In 2011, world production of copper concentrate was 16.215 million tonnes, setting the highest annual production record in history. See Table 2-1 for details.

Table 2-1 Production of the world's main concentrated copper countries (10,000t copper)

Country	2001	2002	2003	2004	2005	2006	2007	2008	2009	2010	2011
Total	1375,7	1356,5	1370,0	1471,4	1518,0	1522,4	1565,0	1559,1	1580,0	1609,8	1621,5
Chile	473,9	458,1	490,4	541,3	532,1	536,1	555,7	533,03	538,9	541,5	524,4
USA	134,0	114,0	112,0	116,0	114,0	122,6	119,0	133,97	120,0	114,2	112,8
Indonesia	104,7	116,3	100,3	84,2	106,4	81,7	78,89	65,05	97,02	86,2	52,6
Australia	89,6	87,9	83,0	85,4	93,0	89,2	97,1	88,5	85,9	84,9	95,7
Peru	72,2	84,5	84,3	103,6	101,0	104,9	119,03	126,79	127,47	124,7	123,5
China	58,7	56,8	60,4	74,2	76,2	75,5	92,8	93,08	97,02	115,6	126,7
Canada	63,3	60,4	55,7	56,3	59,5	60,7	59,62	60,7	49,45	52,5	56,6
Russia	60,0	68,5	66,5	76,7	80,5	77,9	77,0	78,48	74,24	72,7	72,5
Poland	47,4	50,3	50,3	53,1	51,2	51,2	45,2	43,6	43,9	42,5	42,7
Kazakhstan	47,0	47,3	48,5	46,8	43,6	50,95	40,65	41,99	40,59	41,2	43,46

Sources: - data for the years 2001-2006 come from the Yearbook of World Metals Statistics 2007

- Data for the years 2007-2011 come from the Yearbook of Non-Ferrous Metals Statistics 2012

According to incomplete statistics from the World Non-Ferrous Metals, 22 copper projects will be put into production worldwide from 2011 to 2015, extracting 105 million tonnes of copper resources, requiring an initial investment of approximately USD 27.94 billion and creating an estimated potential production capacity of approximately 2.5 million t/year. For details see Table 2-2.

Table 2-2 Number and production capacity of planned copper projects to be carried out between 2011-2015

Year of operation	Number	Reserves (10,000t)	Copper content (t)	Initial investment (million USD)	Planned production capacity (t/year)
2011	7	393 870	20 017 000	5 380,7	464 860
2012	7	512 560	41 993 000	8 794,5	956 700
2013	6	333 400	26 125 000	9 185,1	617 100
2014	1	155 000	9 471 000	4 200,0	400 000
2015	1	30 080	7 716 000	380,0	45 000
Total	22	1 424 910	105 322 000	27 940,3	2 483 660

The development of the modern copper smelting industry began in the 18th century and in recent decades copper production has grown rapidly. Global production of refined copper was over 3 million tonnes in 1950, but reached 19.16 million tonnes in 2011, increasing by 630%.

The past long-term trend reveals that copper production and consumption have increased steadily; But the growth rate is different depending on the periods of inclination. The average annual growth rate during the periods of the previous century is detailed as follows: 4.7% during the 1950s; 4.2% during the 1960s; 2.1% during the 1970s; 1.5% during the 1980s and 3.2% during the 1990s. Over the past 10 years, the average annual growth rate of refined copper production has been 2.2%.

Over the past 40 years (1970-2011), the production of copper mines and smelters worldwide has steadily increased, the average annual growth rate is 2.89% and 2.73% respectively. During this period the melting capacity is about 10% higher than the extraction capacity. Copper consumption has also increased, with an annual growth rate of 2.93%. From 1995 onwards, demand for copper on the market began to slow down and the price of copper on the international market fell, affected by the international economic environment.
As a result, the world's leading copper producers began to reduce production in order to minimize economic losses. In 2003, the world economy began to recover and market demand increased to the point that supply became insufficient, leading to an increase in the price. In 2004, copper producers around the world attempted to quickly recover production capacity, but have still not succeeded in narrowing the gap between supply and demand, so that the price of copper on the international market skyrocketed during the period from 2005 to the first half of 2006. Throughout 2008, due to the global economic depression caused by the financial crisis, the consumption of refined copper by major developed countries around the world declined, and several major developing countries such as China, India and Russia were still the main driving force for global copper consumption. In 2011, refined copper consumption was 19.216 million tonnes, and production reached 19.158 million tonnes. In recent years, the overall supply and demand for refined copper have been in dynamic equilibrium.

As copper is a basic raw material, the evolution of its consumption is closely linked to the development of the world economy. The distribution of world copper consumption is mainly concentrated in two categories of countries or regions. The first is the traditional Western developed countries, such as the USA, Japan, Germany, Italy and France whose copper consumption has a relatively high base value. This value has remained relatively stable in recent years at c, but it has been relatively stable in recent years because of the low growth rate of their economies.

The second category is developing countries and regions of the world with rapidly growing economies, such as China, Korea, Russia, Taiwan, China, Malaysia, Thailand, Philippines, Indonesia, Brazil and Chile, the consumption of these countries and regions enjoys a relatively high growth rate. and has become the key factor for the increase in copper consumption worldwide in recent years. Especially in recent years, China's economy has grown at a relatively high rate, and copper consumption has also grown rapidly. At present, China has overtaken the United States and become the world's largest copper consuming country. In 2011, the main countries or regions consuming refined copper in the world are: China, the United States, Germany, Japan, Korea, Russia, Italy and Taiwan of China.

10. ECONOMIC ANALYSIS AND FINANCIAL EVALUATION

10.1 ECONOMIC ANALYSIS

10.1.1 Calculation philosophy

The mining extractions of the GOLDEN PLANT deposit supply a complex with modular Cu and Co hydrometallurgical units with a production capacity of 200,000 tCu/year and 10,606 tCo/year via heap leaching solutions.

10.1.2 Reference and compliance elements in a dynamic development of new GOLDEN projects

The constitution of a new project respects the GOLDEN dynamic consisting of four phases of development.

1. The exploitable potential estimation phase

- A rapid preliminary study based on available data that allow the establishment of the objectives summarized in an economic model that brings the final reality closer to **+/- 60% certainty**; followed by
- A pre-feasibility study or pre-project, whose purpose is to determine the scope of the project, what will be included in the objectives of the project and what will not, the feasibility of the project, if the expected benefits will be in line with expectations, this pre-feasibility study brings the final reality closer to **+/- 70% certainty**.

At these first two stages, the expected return on investment is roughly determined.

In particular, consideration is being given to whether the necessary resources and funds can be mobilized. We only move on to the next step if it is conclusive, otherwise the project stops there.

The GOLDEN raises capital on the project to be constituted to achieve the expected profit by bringing the project to its realization with a minimum possible risk on investment.

To do this, it appears that GOLDEN must successively:

a. Identify the various options for allocating CAPEX capital (project and variants: various phases)
b. Perform detailed analyses of the different phases of the project
c. From this deduce a cost-benefit profitability calculation (financial evaluation) that suits all parties
d. Choose or negotiate with the partners involved, in view of this calculation and other considerations, the best perspective.
e. Realize this project

2. The Feasibility Study Phase

This is the phase of carrying out the feasibility study work. These particularly include heavy drilling and delineation of deposits and can last several years; two to four years, depending on the size of the deposits to be exploited. The work can be carried out as subcontracted by Golden partners or by specialized houses according to the contractual provisions.

The purpose of this step is to design in detail what must be done to achieve the objective (depending on the method used, a specification may be drawn up at this time), as well as the organization, i.e. the composition of the team, the phasing (determine the tasks to be carried out, with their scheduling, duration, assignment and the necessary technical means), the possible technical environment and the means of monitoring the results.

The feasibility study phase ends with the decision to put the deposits into commercial exploitation or not.

The feasibility study brings the final reality closer to **+/- 80% certainty**

3. The Development Phase

This is the phase of setting up financing for the construction of industrial facilities defined by the feasibility study. It starts, in principle, with a period of about six months of seeking funding. The actual development phase lasts between six months and two years depending on the importance of the construction to be done. It should end with the technical acceptance of the completed infrastructure and the end of the development test period.

The assembly phase of the project applies the feasibility study with some corrections related to hazards and unforeseen events, it tries to bring the final reality **closer to +/- 90% certainty**: it is the realization of the assembly of the installations and their start in first productions.

Research and determination of solutions that consist in studying different technical solutions according to constraints of skills, equipment, deadlines as well as financial and marketing aspects.

Realization and control: the project is carried out by the realization of the production of goods or services. The progress of these tasks is monitored to ensure that deadlines are met.

For this, project management tools are used, including software that allows you to monitor the progress of each task, and if deadlines are exceeded, to reschedule the rest of the project.

In addition to the normal prediction of the EPCM conduct by an experienced company, GOLDEN always combines its expertise at this stage.

4. <u>The Commercial Production Phase</u>

This is the production phase itself: The production phase is never perfect; it also tries to bring the final reality closer for a good period (a year) so as to start **from +/- 90% certainty and achieve 100% compliance**: it is the achievement of objectives.

<u>These new projects are under the New Mining Code (NCM)</u>

This option offers clear advantages to the Government, in terms of income corresponding to the project's tax obligations vis-à-vis the DRC; income tax, royalties to the Government, net profits tax, export tax, import tax.

The New Mining Code provides the DRC Government with more security in terms of revenues, the largest of which come from the tax on net profits.

10.1.3 <u>Implementation planning</u>

- Construction period: 24 months (2 years)
- Trials 6 months effective start (1/2 year);
- Control and implemented capacity assured: 10 months

Total high-capacity production facility 48 months (4 years)

10.1.4 <u>Project Business Case</u>

The business model refers to studies conducted by GOLDEN.
It is a question of respecting cash flow in terms of different summarized values:

- Reserve
- Expected productions,
- Recipes
- Expenditure:
  - CAPEX: export capital (Investments, working capital and additional financing)
  - OPEX: operating and other costs
- Business model data:

  - Macroeconomic data beyond the control of the partners:
    - metal prices,
    -interest rate
    - hazard pay,
    - exchange rate.

  - Project variables:
    - the contents,
    - recovery yields,
    - operating costs,

The most important positions are:

- reagents
- fuel oil
- (global) transport
- Energy

During the construction period and after: application of the New Mining Code;

- net income tax: 30% (Corporation Tax rate)
- Royalties to the DRC Government on gross cash flow 2%
- Export tax (Services)1 %
- Import tax
  - Construction period2%
  - operating period5%
  - Consumable (lubricating fuel ...) 3 %
  - investment,
  - production in general,
  - the metals produced,
- Gross cash flow:

This exercise is accompanied by a detailed analysis of the study itself.
It focuses on an evaluation with mainly the following financial ratios:

1. Net Present Value (NPV) or NPV (Net Present Value)
2. Internal rate of Return (IRR) or IRR
   The IRR being the rate that generates a zero NPV

These ratios reflect the real possibility that a project has to repay external loans.
The overall evaluation criterion is then the IRR internal rate of return (IRR) which verifies:

$$- I + \sum_{1}^{n} \frac{a_t}{(1+r)^t} = 0$$

where $I = I_p + I_c$

I_p = amount of project investment
I_c = amount of additional project investment
a_t = additional value added created in year t
n= project life

The internal rate of return for the project is r = 21%
For one USD invested, the project will create (on average) USD 21 million of added value per year over an infinite period1 M

To this criterion must be added the notion of NPV because it is not the classification by internal rate of return that leads to the maximization of income, but the classification by discounted profits:

$$B = - I + \sum_{1}^{n} \frac{a_t}{(1+i)^t}$$

Where i is the discount rate of the economy concerned

r (IRR) is the rate that makes B(NPV) = 0

Thus the scenario that will generate the highest B for the same i is the one to choose, although its r is lower than the other.

In fact, the problem with GOLDEN is to derive maximum benefit from the available capital that can be committed.

The project was analyzed over a production period of 21 years including 2 years of preparation.

This study is based on an average ore feed grading: 3.54% Cu and
0,27 % Co,

a) The plant is completely built in 24 months as indicated in the study. Commissioning is done in 6 months at the end of the 3rd year.
b) The plant is fully in production at its nominal capacity and at full capacity of 200,000 tCu/year in the 10th year.

It normally takes: six months of financing, award, orders, acceptance and 2 years of construction

For an overall copper recovery yield of 92.0% and an overall cobalt yield of 65.0%, the plant is sized to process +/- 6.2 million ts/year of ores, which leads to 21 years of operation

Copper is produced as high-grade cathodes and cobalt as cobalt salt as precipitated cobalt hydroxides.

Assumptions

The average reference selling price over the operating life is: 7,000 USD/tCu and
12 USD/lbCo.

Total production expenses

Total production expenses amount to: 10,806,878 million USD/year.

This cost is broken down as follows:

The cost of production per tonne of ores is +/- 354 USD/ts.
For mining: 5.83 $US/m³, or 16.5 $US/ts
For metal extraction:

- 1.393 $US/tCu including grinding, leaching, clarification, solvent extraction and electrolysis
- 1.9 $US/lbCo including purification, and/or solvent extraction and electrolysis

Net cash flow discounted at 10% (NPV) is USD 1,413,960 million for an initial investment of USD 1,697,115 million.
The internal rate of return (IRR) is 26%.

The profitability index is 1.3
The payback period is 7 years and 8 months and 15 days.
The calculation details of this study are given in the annexes.

10.2 FINANCIAL EVALUATION

During the first five years, the project will experience a production rate of 100,000 tons per year for copper and 5,000 tons per year for cobalt (in the form of cobalt salt), then it will reach 200,000 tons for copper and 10,000 tons for cobalt (in the form of cobalt salt).

Considering the above rhythms, the productions for the entire duration of the project will be:
- For copper: 2,949,981 tonnes
- For cobalt: 156,433 tonnes.

The trend in metal prices during the project will be:
- copper: 7,000 $US per tonne
- cobalt: $12 per pound.

Expected revenues will be in the order of USD 21,251,226 million, broken down as follows:

- copper revenue: USD 18,614,336 million
- cobalt revenue: USD 2,636.89 million.

Production trends and related revenue are set out in detail in the Annexes.

The overall investment of this project is shown in the summary table below:

10.2.1 Summary of the investment cost for the GOLDEN PLANT DEPOSIT

1	Coût de prospection	**64 243 164**
2	Equipements miniers	**76 342 749**
3	Equipements metallurgiques	**784 450 811**
4	Infrastructures	**114 548 674**
5	Management	**128 354 297**
6	fond de roulement	**211 418 894**
7	Découverture	**167 874 602**
8	Imprévus	**165 829 899**
	Total	**1 697 115 308**

10.2.2 Method of financing

The cost of the investment, estimated at USD 1,697.115 million, will be financed as follows:

Equity (25%): USD 424.270 million.
Borrowings (75%): USD 1,272,836 million.

10.2.3 <u>FINANCING PLANNING FOR THE VALORIZATION</u> OF THE PROJECT <u>DE GOLDEN PLANT</u>

			2013	2014	2015	2016	2017	2018	2019	2020
No	Name of cost	Investment ($)	an 1	an 2	an 3	an 4	an 5	an 6	an 7	an 8
INVESTISSEMENT (ARRONDI)			11%	18%	10%	0%	12%	12%	16%	16%
			0	0	35 796	100 000	100 000	100 000	120 000	120 000
1	Coût de prospection	**64 243 164**	7 287 615	11 724 541	7 006 035	0	8 200 203	8 200 203	10 890 963	10 933 604
2	Equipements miniers	**76 342 749**	8 660 167	13 932 746	8 325 554	0	9 744 633	9 744 633	12 942 172	12 992 844
3	Equipements metallurgiques	**784 450 811**	88 986 517	143 164 270	85 548 241	0	100 129 812	100 129 812	132 985 741	133 506 416
4	Infrastructures	**114 548 674**	12 994 170	20 905 425	12 492 100	0	14 621 359	14 621 359	19 419 115	19 495 146
5	Management	**128 354 297**	14 560 252	23 424 986	13 997 671	0	16 383 553	16 383 553	21 759 543	21 844 738
6	fond de roulement	**211 418 894**	23 982 933	38 584 486	23 056 276	0	26 986 184	26 986 184	35 841 251	35 981 579
7	Découverture	**167 874 602**	19 043 356	30 637 542	18 307 556	0	21 428 051	21 428 051	28 459 309	28 570 735
8	Imprévus	**165 829 899**	18 811 409	30 264 379	18 084 571	0	21 167 059	21 167 059	28 112 677	28 222 745
	Total	**1 697 115 308**	**178 378 638**	**312 638 376**	**186 818 004**	**0**	**218 660 856**	**218 660 856**	**290 410 771**	**291 547 808**

CONCLUSION

The installation of copper and cobalt extractive metallurgy in the polygon is fully justified in view of:

1. With significant geological potential.
2. The relatively low investment cost in this new metallurgy compared to the old one that uses concentrators.
3. In the vicinity of the deposits.
4. The existence of infrastructure in the perimeters of the deposit in question.

The capital invested as well as the interest can be paid over a relatively short period compared to the life of the deposit.

On the other hand, the chosen location can be easily developed, because it benefits from several facilities from the start.

In view of the results of the financial evaluation, there are consistent net profits on turnover.

This is ample proof that the project to develop the GOLDEN PLANT deposit is profitable.

GOLDEN MOUNTAIN TECHNOLOGY TEAM

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