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Foreword

The 2019 Engineering Institution of Zambia (EIZ) National Symposium, held in Livingstone, Zambia on 26th April 2019, was an historical one having been officially opened for the first time by a sitting Republican President, His Excellence Edgar Chagwa Lungu. A record number of engineering professionals from academia, industry, organisations and government were in attendance to witness the occasion. The theme for the 2019 Symposium was "Sustainable National Development: Issues and Challenges for Zambia's Development Agenda". Accordingly, the opening ceremony was proceeded by keynote and plenary presentations from high level government officers and chief executive officers to share current developmental aspects in various economic sectors. Dr Roland Msiska, Head of the Zambia Atomic Agency (ZAMATOM), Zambia's Nuclear Energy Programme Implementing Organization (NEPIO) encouraged engineering professionals to actively participate in the programme that will see the construction of a Centre for Nuclear Science and Technology (CNST), and later a Nuclear Power Plant (NPP). His presentation on "Generation of Power in Zambia using Nuclear Technology: Is Zambia Ready in Terms of Policy, Financing and Skills Required" outlined the long term plan for Zambia's Nuclear Programme (2020-2166), and concluded by expressing Government's expectations of engineering professionals' involvement throughout the development, commissioning and decommissioning of the both the CNST and NPP. Other plenary presentations included the Zambia Development Agency (ZDA) made by the Acting Director General, Mr Matongo Matamwandi, who presented on "Business Climate and Opportunities in Zambia"; the National Road Fund Agency (NRFA) made by CEO Eng Wallace D. Mumba, who presented on "Sustainable Financing for Road Infrastructure Development"; Zambia Railways Limited (ZRL) presentation on "Railway Transportation" and the "Vision of the Railway Network on Heavy and Bulk Cargo and the Role of Engineers" made by CEO Mr Christopher C. Musonda; Zambia Airports Corporation Limited (ZACL) presentation on "Airports Development Projects: Overview and Status" made by Managing Director Eng Fumu Mondoloka; Lusaka South-Multi Facility Economic Zone (LS-MFEZ) Limited presentation on "Progress of the Lusaka South-MFEZ Development Since Inception" made by Mr Maxwell Zulu; and the Road Development Agency (RDA) presentation on "Vulnerability of Zambia's Road Infrastructure to Climate Change" made by Eng Joseph Chibwe. All these presentations can be accessed on the EIZ website (www.eiz.org.zm). For the technical papers that were approved for presentation at the 2019 EIZ National Symposium, the EIZ Publications and Editorial Committee decided to prepare this Book of Proceedings.

It is with deep satisfaction that I write this foreword to the collection of fourteen insightful 2019 EIZ National Symposium papers. The papers contained in this book cover a wide range of topics including mining, manufacturing, telecommunication, energy, water and sewerage, engineering education and training, and infrastructure. The authors have provided state-of-the-art contributions, thereby successfully providing value-adding inspiration and stimulation among the engineering professionals. All the papers were peer reviewed to assure their quality. On behalf of the EIZ Publications and Editorial Committee, I wish to thank each contributing author and paper reviewer whose commitment has made it possible to produce this book. This is a valuable addition to the record of ideas and experiences that we share as engineering professionals.

In conclusion, I wish to take this opportunity to thank all the participants, and sponsors of the 2019 EIZ National Symposium for an invaluable contribution. I also wish to express my utmost appreciation to the EIZ Publications and Editorial Committee and EIZ Secretariat for their valued advice and support at every stage of organising this year's Symposium which led to the continued high quality and popularity of the Symposium.

Enjoy the read.

Eng Prof Levy Siaminwe
Chairperson
EIZ Publications and Editorial Committee

April 2019

Mining method selection for the Synclinorium mining project: A Case Study of Mopani Copper Mines, Zambia.

Hugo Nicacio[1], Bunda Besa[2] and Webby Banda[3]

ABSTRACT

Mining method selection is one of the most critical activities of mining engineering with the ultimate goal of maximizing profit, mineral recovery and arrive at a method with the least problems among feasible alternatives. The geology at the Synclinorium is complex with four main lithologies. A critical geological feature is the foliation or bedding that replicate the folding along the strike of the orebody. This poses a challenge in terms of selecting a suitable mining method for this orebody that will be less costly with high recoveries and low dilution. The main aims of the research were to select a suitable mining method, identify the current challenges encountered in the existing mining methods and carry out an economic evaluation of the Synclinorium mining project. The methodology used in data collection involved underground visits to various sections of the mine to understand the geology, rock types, orientations and geological discontinuities, current mining methods and associated challenges. The results gathered were then subjected to mine design criteria for selecting a mining method. In this research, University of British Colombia (UBC) mining method selection criteria was used for the selection of a suitable mining method. Both methods revealed that, Sublevel stoping, Cut and fill, Sublevel caving and Block caving were selected in that order. However, after subjecting the selected mining methods to geotechnical, technical and economic evaluation, Sublevel Open (SLOs) stoping with fill in the anticline, Sublevel caving (SLC) in the limbs and VCR in the synclines were recommended. The current SLC and Vertical Crater Retreat methods require modification to address the current challenges due to hang-ups, delayed hangingwall exposure and creation of a huge void prior to hangingwall collapse as mining progresses and hole deviations in VCR at 50m that results in re-drilling and ultimately increase the costs of mining. The economic evaluation indicates that the Synclinorium mining project is viable with a projected average annual income of US$350 million.

Keywords: **Mining method, economic evaluation, geology, orebody, Synclinorium**

1.0 Introduction

Mopani Copper Mines (MCM) Nkana site, is anticipating a fifty percent decline in ore production in the coming years due to the projected depletion of copper ore at Central and Mindola North Shafts. Currently, Nkana produces between 3.5 and 4.0 million tpa (tonnes per annum) of ore from its four underground mines. Due to depleting ore reserves in the Central Shaft and Mindola North Shaft, it is projected that the total amount of ore to be mined from the South orebody shaft (SOB) and Mindola Main shafts will be 1.8 million tpa by 2019. Mopani Copper Mines intends to compensate for this decline in production by mining the copper ore in the Synclinorium orebody by sinking one rock-hoisting shaft and one ventilation shaft. However, the geology at the Synclinorium is complex. There are four main lithologies within the Synclinorium i.e. SOB Shale (SOBS), Hangingwall Argillite (HWA), Near Water Sediments (NWS), and the Upper Quartzite (Upper-Q). Each unit is folded with variable thickness and

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rock mass conditions. Another critical geological feature in the Synclinorium is the foliation or bedding that replicate the folding along the strike of the orebody. This ultimately poses a challenge in terms of selecting a suitable mining method for this orebody as shown in Figure 1.

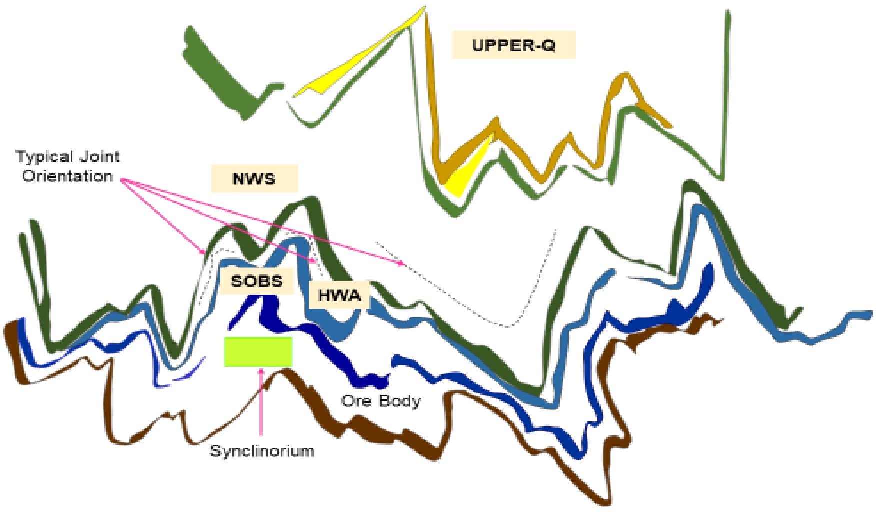

Fig 1: Geological section showing the main lithologies within the Synclinorium (Mopani geology, 2013)

Therefore, the main objectives of the research were to identify the current challenges encountered in the existing mining methods and recommend solutions. Other objectives included designing a suitable mining method for Nkana Synclinorium Mining Project and Carry out economic analysis of the design.

2 Framework for mining method selection using UBC method

This methodology is the modified version of the Nicholas approach. It followed a very similar pattern to the Nicholas method. A value of -10 was introduced to strongly discount a method without totally eliminating with the -49 value. Moreover, the rock mechanics ratings were adjusted to reflect improvements with ground support and monitoring techniques. Instead of the RQD in the Nicholas method it uses the RMR and it does not focus on fracture strength (Bieniawski, 1968 ; Bieniawski, 1981). This methodology gives a scoring for suitability of a mining method to the predefined parameters. Adding up all the scoring will results in a final score, the mining method scoring the highest will be ranked highest and is considered most suitable although it is further subjected to economical, technical and other factors.

3 Materials and methods

The methodology used in data collection involved underground visits to various sections of the mine to understand the geology, rock types, orientations and geological discontinuities, current mining methods and associated challenges. Other methods involved a review of research works on underground mine planning and design, literature on mining method selection, mining journals and research done by Murray and Roberts and SRK consulting firms on the design. Detailed discussions on the input parameters for mine design with mine planning, geologists, geotechnical engineers, senior mining officials and cost accountants were undertaken to understand the current cost of mining. Furthermore, diamond drilling and logging of borehole core was also conducted.

4 Data collection

Data required for mining method selection is presented in Tables 1 to 4.

Table 1: General characteristics of the ore deposit

INPUT DATA	DESCRIPTION
General deposit shape	Irregular
Ore thickness	60 m
Ore plunge	70 degrees
Grade distribution	Uniform
Depth	1170 m

Table 2: Geotechnical information of the ore deposit

INPUT DATA	DESCRIPTION
Rock substance strength	175 MPa
RQD	60 %
Joint Spacing	0.2 – 0.6 m
Joint Condition	Slightly rough surface, separation < 1 mm, soft joint wall rock, dry condition
Principal insitu stress	17.1 MPa
RMR	63 – 77 %
UCS / Principal stress	10.2

Table 3: Geotechnical information on Hangingwall

INPUT DATA	DESCRIPTION
Rock substance strength	300 MPa
RQD	71 %
Joint Spacing	0.6 m
Joint Condition	Slightly rough surface, separation < 1 mm, soft joint wall rock, dry condition
Principal insitu stress	17.1 MPa
RMR	67 %
UCS / Principal stress	17.5

Table 4: Geotechnical information on Footwall

INPUT DATA	DESCRIPTION
Rock substance strength	322 MPa
RQD	71 %
Joint Spacing	0.8 m
Joint Condition	Slightly rough surface, separation < 1 mm, soft joint wall rock, dry condition
Principal insitu stress	17.1 MPa
RMR	68 %
UCS / Principal stress	18.8

5 Data analysis and discussion of results

The results gathered were then subjected to mine design criteria for selecting a mining method. In this study, University of British Colombia (UBC) mining method selection criteria was used for the selection of a suitable mining method (Hamrin, 1982, Bullock, 1982a). The mining methods selected were further subjected to geotechnical, technical and economical evaluation to arrive at the most suitable mining method (Morrison and Russell, 1973; Boshkov, and Wright, 1973). Using this input data in Tables 1 to 4, the UBC method selection process was calculated and the results are shown in Table 5;

Table 5: Results of the UBC mining method selection criteria

Geometry/Grade distribution											
		O/P	*B/C*	*SST*	*S/C*	*LWM*	*R&P*	*SHS*	*C&F*	*T/S*	*SOS*
General shape	Irregular	3	0	1	1	-49	2	2	4	0	4
Ore thickness	Thick	4	3	4	4	-49	-49	-49	1	2	0
Ore plunge	Steep	1	4	4	4	-49	-49	4	4	0	2
Grade distribution	uniform	3	3	4	3	4	4	3	2	2	0
Depth	Deep	-49	3	2	2	3	2	2	4	1	2
Rock mechanics characteristics											
		O/P	*B/C*	*SST*	*S/C*	*LWM*	*R&P*	*SHS*	*C&F*	*T/S*	*SOS*
Rockmass ratings											
Ore zone	Strong	3	0	4	1	2	5	3	3	1	0
Hanging wall	Strong	4	2	4	2	3	5	4	3	3	0
Footwall	strong	4	2	3	3			3	2	2	0
Rock substance strength											
		O/P	*B/C*	*SST*	*S/C*	*LWM*	*R&P*	*SHS*	*C&F*	*T/S*	*SOS*
Ore zone	medium	3	1	4	3	2	3	3	3	1	1
Hanging wall	medium	4	2	4	2	2	2	3	4	2	1
Footwall	Strong	4	1	3	2			3	2	1	0
GRAND TOTAL		-16	21	37	27	-131	-75	-19	32	15	2

O/P – Open Cast; B/C – Block Caving; S/C – Sublevel Caving; R&P – Room and Pillar; C&F – Cut and Fill; SOS – Sublevel Open Stoping

From Table 5, the top four mining methods are; Sublevel Open Stoping (37), Cut and Fill (32), Sublevel Caving (27) and Block Caving (21). Furthermore, the data was entered using UBC as shown in Figure 2, the results also showed the following top four mining methods ; Sublevel Stoping (37), Cut and fill (32) , Sublevel Caving (27) and Block Caving (21)

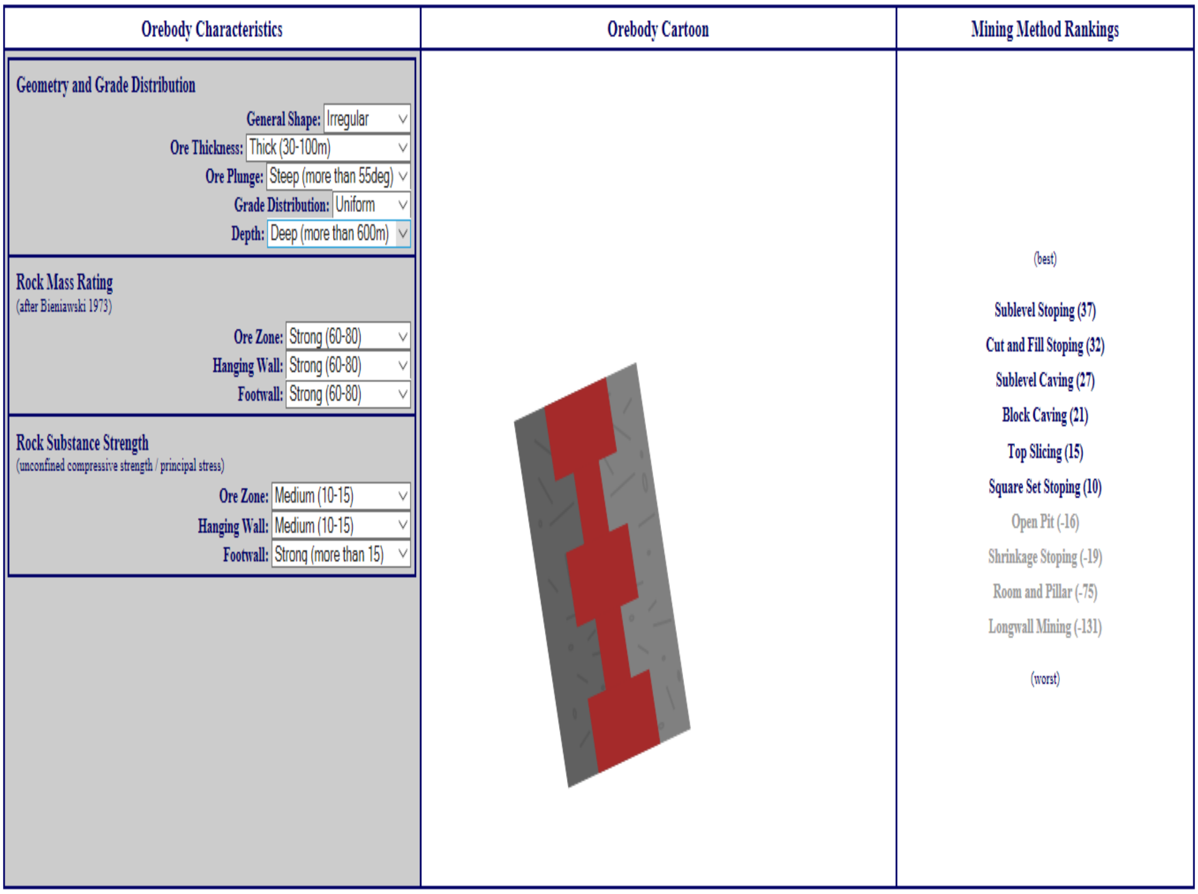

Fig 2: UBC windows results

6 Discussion on exclusion of mining methods

(a) The mining method selection using the UBC method resulted in sublevel stoping as most suitable for the Synclinorium orebody.

(b) Cut and fill stoping scores just below Sublevel stoping. The shape and the dip of the orebody are in favour of Cut and Fill. However, this mining method is not workable for the scale of operations due to the following reasons;

(i) It is labour intensive it will be very expensive and will take long to set up a backfill plant and will be difficult to source for waste and cement.

(ii) The activity of filling complicates the mining cycle causing reduction in production. For these reasons, this mining method is not suitable for this deposit.

(c) Sublevel caving would be possible as the orebody shape, volume and dip are favourable for sublevel caving mining method.

(d) Block caving is suitable mining method owing to the nature of the orebody. However, the method has more disadvantages compared to the existing mining methods being applied at SOB. These include; high capital investment, high subsidence, low selectivity and flexibility, high dilution from hangingwall when overburden fragmentation is higher than expected and the slow development rate but very high development cost. This will make the project take longer than projected before the areas come into production compared to other new available mining methods. Based on these reasons, block caving may not be applied for this deposit.

6.1 Included mining methods

After excluding the unsuitable mining methods, sublevel stoping and sublevel caving options are the only mining methods likely to be viable for this deposit. However, there are multiple stoping methods available in modern day mining practiced world over. These methods are variations of conventional sublevel stoping. The differences between them is the direction of mining, the design of the stopes and the methods of backfilling. These include; Longitudinal Bench Stoping, Transverse Bench Stoping and Vertical Crater retreat (VCR). VCR mining method is currently in use at the shaft and all the machinery and experienced personnel are available at the mine to undertake it unlike the other mining methods.

6.2 Proposed SLC Methodology

In order to avert the risk posed by the current SLC arrangement it is proposed to change the position of the slot drive. Preferably the slot drive should be positioned closer to the hangingwall of the eastern limb (c-fold). Mining should first proceed along the slot drive just to open up the hanging on the various sublevel. The hangingwall of the eastern limb (c-fold) is systematically undercut to induce natural cave prior to transverse multiple front retreat.

Though the undercut will create a very limited opening along hangingwall, it will completely undercut the eastern limb hangingwall (c-fold), creating the required HR or footprint to induce complete hangingwall collapse. Considering the volume of the undercut the potential for a catastrophic hangingwall collapse and the subsequent air blast could be minimized if not completely avoided. Following the hangingwall cave transverse SLC could resume. This will progressively and gradually induce further hangingwall cave which is likely to link up with the eastern "B" limb cave.

It should be noted that the hanging of the current SLC stopes have been caving naturally. Considering the medium and long term production schedules, by the year 2023 and beyond, the production stope faces are likely to extend deeper. This is likely to increase the potential for the current cave and the SOBS cave to merge.

6.3 Cavability Assessment

This assessment was based on an empirical approach. It was deduced that to induce a complete cave in the hangingwall requires a hydraulic radius in the ranges of 28 m – 37 m. The Laubscher's stability graph was used for cavability assessment as shown in Figure 3.

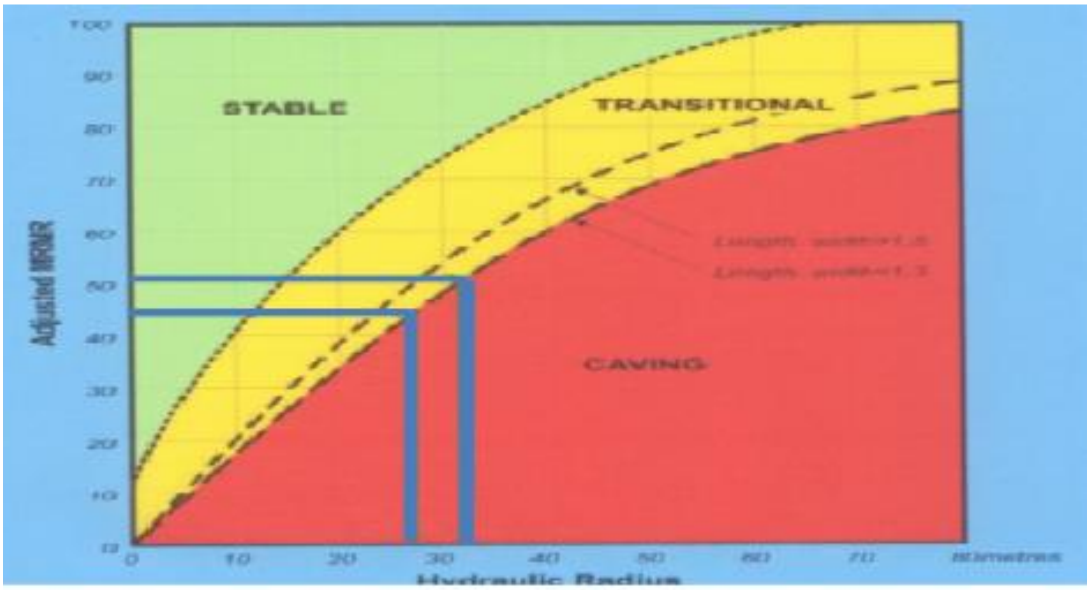

Fig 3: Laubscher's Stability Graph – showing the average HR values required for the Synclinorium ore and Hangingwall Argillite to completely cave naturally (Laubscher, 1981).

Hangingwall collapse is a function of hangingwall exposure or footprint and it's measured in terms of hydraulic radius. It could be inferred from Table 6, the hydraulic radius matrix that should mining extent along strike over a distance of 130m and down dip towards or beyond 3585L the HR of the eastern hangingwall (c-fold) will exceed 37 m, and it's likely to induce complete cave as read from Laubscher's stability graph shown in Figure 3. The strike length under consideration at the SOBS is about 180 m. Thus, though the Synclinorium plunges to the north, there exists the potential to create a footprint likely to induce hangingwall cave as the hangingwall undercut mining advances down dip.

Table 6: Hydraulic Radius matrix – Synclinorium

MINING ALONG STRIKE

MINING DOWN DIP

	H/W EXP. [M]	100	110	120	130	140	150	160	170	180	190	200	210	220	230	240	250
3360L	78	22	23	24	24	25	26	26	27	27	28	28	28	29	29	29	30
3435L	112	26	28	29	30	31	32	33	34	35	35	36	37	37	38	38	39
3510L	142	29	31	33	34	35	36	38	39	40	41	42	42	43	44	45	45
3585L	175	32	34	36	37	39	40	42	43	44	46	47	48	49	50	51	51
3660L	213	34	36	38	40	42	44	46	47	49	50	52	53	54	55	56	58
3735L	251	36	38	41	43	45	47	49	51	52	54	56	57	59	60	61	63
3810L	289	37	40	42	45	47	49	51	54	55	57	59	61	62	64	66	67
3885L	327	38	41	44	47	49	51	54	56	58	60	62	64	66	68	69	71

6.4 Vertical Crater Retreat (VCR)

This mining method is currently being used at SOB shaft in the synclines. The method involves setting up a top drilling level and the holing level. The use of a 165 mm diameter holes drilled with the down the hole (DTH) hammer allows accurate drilling of 50m. However, the method encounters challenges in drilling due to hole deviation (Almgren and Klippmark, 1981) that lead to improper charging and redrilling at 50 m. However, in case of Synclinorium this will be reduced to 40 m to allow for increased accurate in drilling.

Another advantage of VCR is that the blasted ground (ore) is dropped uniformly across the stope to leave only sufficient void for the next blast. Therefore, the rock mass in the stope continuously supports the stope walls. This reduces waste dilution from hangingwall and footwall. The width of the stopes will vary depending on the orebody but stope strike length and rib pillar will be 15 m and 8 m respectively. The stope will be taken first followed by the rib and crown pillar. The crown drilling will be done using a drifter machine as per the current practice at SOB shaft.

7 Economic analysis

(a) Capital expenditure

An estimated capital cost of US$96 million is required to set up the areas for production. This expenditure will push production to 4.0 million tonnes per annum by 2023. The capital costs are the costs incurred in capital development, buying and replacing of equipment (Botin and Singh, 1981; Chatterjee and Just, 1981). Tables 7 and 8 shows the capital costs and operating cost for the period 2019 to 2022 and the projection beyond.

Table 7: Projected Capital expenditure

CAPEX ACTIVITY		2019	2020	2021	2022	2023	2024	2025	2026	AV 2026-2048
Stope chutes for Dump truck	USD'000	350	350	350	350	350	350	350	350	350
Ventilation fans	USD'000	325	325	325	325	325	325	325	325	325
Pumping system	USD'000	731								
loaders	USD'000	2,800	2,800	3,500	4,900	2,800	2,800	3,500	4,900	4,900
Dewatering & Diamond drilling	USD'000	469	469	469	469	469	469	469	469	469
Conveyors	USD'000			3,600						
Fill plant	USD'000			2,000	2,000					
Services	USD'000	828	847	794	766					
Labour cost(MCM)	USD'000	819	1,025	1,025	1,025					
Sumps,settlers,pump chamber	USD'000		659	4,058	446					
Crusher system	USD'000		620	5,445						
66KV Substation	USD'000	4,500	3,000							
Primary development	USD'000	6,048	1,765	3,487	4,357	4,357	4,357	4,357	4,357	4,357
Dump trucks	USD'000				1,600	1,600	3,200	1,600	2,400	2,400
Total	USD'000	**16,870**	**11,860**	**25,053**	**16,238**	**9,901**	**11,501**	**10,601**	**12,801**	**12,801**

(b) Operating costs

Table 8: Schedule of annual operating costs

OPEX ACTIVITY		2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	AV 2030-2048
Development	USD'000				6,218	6,755	14,181	24,971	32,217	32,212	32,217	23,217	32,217
Stoping	USD'000					14,685	39,131	67,274	85,897	86,152	86,333	86,323	86,323
Support	USD'000				1,512	1,650	2,868	5,680	7,915	7,915	7,915	7,915	7,915
Tramming	USD'000					1,682	4,373	7,479	9,923	9,923	9,923	9,923	9,923
Shafts	USD'000					2,170	5,644	9,651	12,804	12,804	12,804	12,804	12,804
Auxillary services	USD'000					5,260	7,741	13,238	17,562	17,562	17,562	17,562	17,562
Technical	USD'000					392	1,020	1,745	2,315	2,315	2,315	2,315	2,315
Workshops	USD'000					439	1,142	1,953	2,591	2,591	2,591	2,591	2,591
Supervision	USD'000				256	333	850	1,451	1,923	1,923	1,923	1,923	1,923
Total	USD'000				**7,986**	**33,366**	**76,950**	**133,442**	**173,147**	**173,397**	**173,583**	**164,573**	**173,573**

The expected production, development and longhole metres in the different parts of the orebody are shown in tables 9 to 10 below;

Table 9: Schedule of mining activities in the anticline

SLOS Production		2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	AV 2030-2038
Tonnes hoisted	*'000t*					271	843	1,325	1,600	1,600	1,600	1,600	1,600
Cu Grade	*%*					1.93	1.96	1.96	2.21	2.20	2.21	2.21	2.21
Co Grade	*%*					0.04	0.04	0.05	0.05	0.05	0.05	0.05	0.05
Copper in ore hoisted	*Tonnes*					5,236	16,528	25,928	35,321	35,171	35,321	35,321	35,321
Cobalt in ore hoisted	*Tonnes*					120	326	573	855	854	855	855	855
Secondary development	*metres*				2,284	2,494	6,470	10,360	12,500	12,500	12,500	12,500	12,500
Long hole drilling	*metres*					46,132	104,910	165,000	202,700	202,700	202,700	202,700	202,700

Table 10: Schedule of mining activities in the limbs

SLC Production		2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	AV 2030-2038
Tonnes hoisted	*'000t*					135	453	879	1,300	1,300	1,300	1,300	1,300
Cu Grade	*%*					1.64	1.83	1.91	1.54	1.57	1.59	1.59	1.59
Co Grade	*%*					0.07	0.10	0.05	0.05	0.05	0.05	0.05	0.05
Copper in ore hoisted	*Tonnes*					2,214	8,290	16,789	20,020	20,410	20,670	20,670	20,670
Cobalt in ore hoisted	*Tonnes*				-	95	453	440	650	650	650	650	650
Secondary development	*metres*				1,731	1,890	3,478	6,980	10,011	10,011	10,011	10,011	10,011
Long hole drilling	*metres*					19,243	56,380	109,450	162,340	162,340	162,340	162,340	162,340

Tables 11 and 12 shows schedule of mining activities in synclines and expected revenue and expenditure of the synclinorium project respectively.

Table 11: Schedule of mining activities in synclines

VCR Production		2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	AV 2030-2038
Tonnes hoisted	*'000t*					271	468	811	1,100	1,100	1,100	1,100	1,100
Cu Grade	*%*					1.88	1.65	1.58	1.64	1.63	1.64	1.64	1.64
Co Grade	*%*					0.06	0.05	0.05	0.07	0.07	0.07	0.07	0.07
Copper in ore hoisted	*Tonnes*				-	5,095	7,722	12,814	18,040	17,930	18,040	18,040	18,040
Cobalt in ore hoisted	*Tonnes*				-	163	234	406	770	770	770	770	770
Secondary development	*metres*				1,192	1,301	1,874	3,530	4,348	4,348	4,348	4,348	4,348
Long hole drilling	*metres*					20,655	21,570	37,400	50,000	50,000	50,000	50,000	50,000

Table 12: Expected revenue and expenditure of the Synclinorium project

PARAMETER		2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	AV 2030-2038
Copper in ore hoisted	*Tonnes*	-	-	-	-	12,545	32,540	55,531	73,381	73,511	74,031	74,031	74,031
Cobalt in ore hoisted	*Tonnes*	-	-	-	-	377	1,013	1,418	2,275	2,274	2,275	2,275	2,275
Final processed cu	*Tonnes*	-	-	-	-	11,039	28,635	48,867	64,575	64,690	65,147	65,147	65,147
Final processed co	*Tonnes*	-	-	-	-	332	891	1,248	2,002	2,001	2,002	2,002	2,002
Projected $/Tonne CU	$	-	-	-	-	6,800	6,800	6,800	6,800	7,800	7,800	7,800	7,800
Projected $/Tonne CO	$	-	-	-	-	40,000	40,000	40,000	40,000	40,000	40,000	40,000	40,000
TOTAL COSTS	$	*16,870,000*	*11,860,000*	*25,053,000*	*24,224,000*	*43,267,000*	*88,451,000*	*144,043,000*	*185,948,000*	*186,198,000*	*173,583,000*	*164,573,000*	*173,573,000*
REVENUE CU	$	-	-	-	-	75,068,083	194,718,762	332,295,709	439,111,904	504,579,504	508,148,784	508,148,784	508,148,784
REVENUE CO	$	-	-	-	-	13,273,920	35,657,600	49,913,600	80,080,000	80,044,800	80,080,000	80,080,000	80,080,000
TOTAL REVENUE	$	-	-	-	-	88,342,003	230,376,362	382,209,309	519,191,904	584,624,304	588,228,784	588,228,784	588,228,784
REVENUE LESS COSTS	$	**- 16,870,000**	**- 11,860,000**	**- 25,053,000**	**- 24,224,000**	**45,075,003**	**141,925,362**	**238,166,309**	**333,243,904**	**398,426,304**	**414,645,784**	**423,655,784**	**414,655,784**

From Table 12, it is clear that the Synclinorium mining project is viable with an average annual projected revenue of US$350 million.

8 CONCLUSION

According to the traditional and online UBC mining method selection tools, economical and geotechnical evaluation, the mining methods applicable in the Synclinorium, are Open Stoping with fill in the anticline, SLC in the limbs and VCR in the synclines. However, the current SLC and Vertical crater retreat methods require modification to address the current challenges due to hang-ups, delayed hangingwall exposure and creation of a huge void prior to hangingwall collapse as mining advances from the centre toward the economic boundaries in SLC and hole deviations in VCR at 50 m that results in re-drilling and ultimately increasing the costs of mining. The empirical analysis, indicated that to induce a complete cave in the hangingwall required a hydraulic radius in the ranges of 28 m – 37 m. The Laubscher's stability graph was used for cavability assessment. The modifications to the two mining methods will increase production, minimize dilution and costs. Furthermore, Sublevel Open Stoping will help address the challenges of waste handling and will reduce costs of crushing, transporting and hoisting of waste to surface. The economic evaluation indicates that the Synclinorium mining project is viable with a projected average annual income of US$350million.

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Perceptions on Distribution Challenges in the Management of Mineral Wealth for Zambia

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Abstract

Distribution of mineral wealth is a concept explaining how surpluses or economic rents can be shared equitably from mineral production among key stakeholders; private mining companies, government at all levels, local communities, and other organisations and entities. Many resource-rich countries fail to meet goals of sustainable development linked to equitable distribution of economic rent resulting in the mining and minerals being a curse for such countries. This research is review-based which aimed to highlight perceptions on Zambia's mineral revenue distribution challenges centred on equity principles of proportionality, parity and priority in the country's management of mineral wealth. It also aimed at assessing the performance of fiscal tools in rent appropriation and establishing an ideal benefit-sharing mechanism for Zambia. The study methodology involved wide literature reviews on mineral wealth distribution practices. Findings indicated that Zambia has no well-defined rules or guidelines to grant equitable distribution of rents from the mining and mineral sector among parties involved. Zambia faces distributional challenges in terms of poor rent allocation (compensation) based on proportions to each party's financial contribution to mineral development. The parity concept indicated deprived rent distribution in Zambia meant to maximize the total welfare of all interested parties in the sector. Furthermore, it was established that challenges of prioritising wealth allocation to the least well-off parties in the sector that have shouldered external costs or emotional liabilities of mineral project development are regular. Non-implementation of the National Decentralisation Policy has equally restricted the transfer of mining revenues to subnational levels in Zambia. Based on the employed revenue distribution fiscal tools, Zambia still faces administrative challenges resulting in inequitable allocation of mineral resource revenues to various parties. The research recommends that government improves institutional capacities needed for rent appropriation and puts in place recognised benefit-sharing mechanisms to facilitate equitable distribution of mineral wealth in the country.

Keyword: mineral wealth, distribution challenges, economic rent, proportionality, Zambia.

1. Introduction

Zambia is a mineral economy country highly dependent on mining as a major productive industry with most copper mining operations located on the Copperbelt and Northwestern provinces of the country. The copper mining sector in Zambia is integrated with different mining companies

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involved in the production of varied products in the value chain. Companies might produce copper ore, intermediate products (concentrates, blister copper etc.), copper anodes and copper cathodes. The main economic impacts of most extractive industries are reflected on; the macroeconomic performance, the government revenues, the direct employment, and the economic externalities and spill over effects on other sectors of the economy (Sigam and Garcia, 2012). In Zambia, the percentage of extractive industry contribution to the economy in 2014 was 78% of export earnings, 1.6 % of total Foreign Direct Investment (FDI), 32% of government revenues, 6% of Gross Domestic Product (GDP) and only 1.7% of direct employment (ZEITI, 2015a).

According to Mining, Minerals and Sustainable Development [MMSD] (2002) distribution deals with sharing of mineral rent equitably among private mining companies, government at all levels, local communities, and other organizations and entities. However, this is not a straightforward mechanism because of varied interests and expectations of the different stakeholders from the resource wealth. In most resource rich countries, there is wide public opinions necessitating sharing of mineral revenues to be a highly debatable issue. Jeffrey (2001) noted that the allocation of revenue from natural resources is a critical political question. Tensions over the division of natural resource exports have been repeatedly cited as a central contributor to open conflict in some countries and as the major source of political volatility in many others. Ignoring the distributional effects of revenue surplus from mining may create disruptive social tensions, thereby increasing business risks for mining companies (Söderholm and Svahn, 2014).

Countries face policy challenges in the management of mineral wealth (Eggert, 2001) which include benefit sharing (Söderholm and Svahn, 2014). In Zambia, despite the long history of copper mining, benefit sharing or economic rent distribution could be regarded as sub-optimal. This is due to lack of appropriate principles followed to guide in the allocation of mineral surpluses, weak mechanisms and institutional arrangements to achieve the desired rent distribution procedures and existing mistrust in rent appropriation from the mining sector that is considered asymmetrical. Mining companies are believed to appropriate more rent than other parties a situation debated in most resource-rich African states.

This study aimed to:

(a) examine the current distribution challenges Zambia faces in the sharing of revenues (rents) from the mining sector based on the equity allocation principles involving proportionality, parity and priority;
(b) assesses the performance of the fiscal options available in the taxation regimes of Zambia for collection and distribution of rent; and
(c) suggest ideal mechanisms for rent sharing in Zambia based on the evaluations.

2. Theoretical background

Some authors (Bauer *et al.*, 2016a; Eggert, 2001; MMSD, 2002) have described rent distribution. Natural resource revenue sharing is not the only way that producing states and regions or affected communities can capture a share of the benefits from resource extraction. It should be viewed within the broader concept of "benefit sharing" in the extractive sector with five ways residents affected by oil, gas or mining activities can benefit (Bauer *et al.*, 2016a). These include first national governments targeting services directly to producing areas or affected communities and second, companies making in-kind payments in the form of infrastructure or public services. Third companies making voluntary payments to communities in form of infrastructure, services or cash, through Corporate Social Responsibility (CSR) package. Fourth local citizens receiving a share of the resource in-kind and finally producing areas or affected communities benefiting from ‘local content’ policies.

2.1 Equity principles

The theories on equity principles based on Eggert (2001) and Young (1994) describe the structure of resource allocation rules of equity which discuss the concepts of proportionality, parity and priority. Young (1994) explained that equity is concerned with the proper distribution of resources, rights, duties, opportunities and obligations in society. This represents a balance between competing principles of need, desert and social utility.

2.1.1 Proportionality

Proportionality concept explains rent distribution based on the proportion of a party's financial contribution to mineral development. This is a principle Eggert (2001) considers appealing since each party deserves an outcome that is in proportion to its contribution to that outcome. In real situation, determining each party’s contribution in a mining project is not quite an easy task. This contribution could be based on financial capital by shareholders, public infrastructure by the government used by the mining projects or mineral resources deemed to belong to the government for its nationals.

2.1.2 Parity

Parity is an equity principle explaining the allocation rule that should treat claimants equally (Young, 1994). Under this principle, Eggert (2001) noted that all interested parties are treated equally, either because they are viewed as equals or because it is difficult to distinguish among parties. Different parties have different competing needs from the mineral projects.

2.1.3 Priority

This is as an allocation concept describing that the least well-off communities (parties) should be given rent distribution priority (Eggert, 2001). The principle requires handling the challenge of reducing the incentives for a mining project to generate the rents in the first place. According to Söderholm and Svahn (2014), benefit sharing also involves the management and the allocation of the revenues across different priorities as well as over time.

2.2 Fiscal options for rent sharing

Many authors (Bauer *et al.*, 2016b; Fischer. 2007; Baunsgaard, 2001; Söderholm and Svahn, 2014; International Finance corporation [IFC], 2015) have presented on resource revenue sharing regimes and fiscal instruments. In Zambia, all tax revenues including tax revenues from the mining sector are collected by Zambia Revenue Authority (ZRA) on behalf of the government and are deposited in the general account (Control 99) at Bank of Zambia. This bank account contains revenue from all the sectors of the economy (ZEITI, 2015c). Therefore, ZRA receives most of the funds from mining companies. The design of mechanisms and institutional arrangements that achieve the desired rent distribution is assessed at project level and in the area covering the fiscal policy- the level, form, and disposition of taxes and royalties (Eggert, 2001). Therefore, mineral resource rich nations have different fiscal tools enshrined in the taxation regimes to appropriate rents to different stakeholders.

2.3 Ideal mechanism for surplus (rent) allocation

Managing resource wealth in developing countries requires not only good governance, in the sense of transparent management and absence of corruption, but also good policy, such that resource exploitation benefits the economy and society as a whole (Fischer, 2007). There is no straightforward definition of natural resource revenue sharing (Bauer *et al*., 2016b) and studies by Söderholm and Svahn (2014) and Fischer (2007) have showed that the global experience with benefit-sharing varies widely in terms of summarised options for collecting and distributing revenues. The purpose of benefit-sharing mechanisms is to ensure that a significant part of the economic benefits is retained in the region in which the rent is generated (Söderholm and Svahn, 2014).

3. Implications for Zambia

The distribution challenges faced by Zambia based on the allocation principles are discussed.

3.1 Proportionality concept

State equity participation - In the extractive industries, state equity participation ensures that capacity building, improved monitoring and direct financial benefits (Natural Resources Governance Institute [NRGI], 2015a) are achieved from the mineral projects. Though Zambian copper mining is essentially a private industry, the government has retained a sizeable shareholding (ranging from 10 to 20% of the shares in the privatised mines) through Zambia Consolidated Copper Mines- Investments Holding (ZCCM-IH). Such an arrangement is not unprecedented in the copper mining industry (World Bank, 2011). However, Zambia's equity participation (10-20%) in the privatised mines has been a product of privatisation process which has not performed well with respect to rent appropriations when compared to equity stake arrangements in countries like Chile and Botswana. Even the details of participation agreements and contractual relationship between the ZCCM-IH and the companies in which it holds equity are generally none available (Conrad, 2012; Manley, 2012).

There are still challenges of erratic generation of allocates of proceeds in terms of dividends and price participation dues in Zambia based on proportionality participation. Lundstøl *et al*. (2013) indicated that the payment of dividends in most years since privatisation in Zambia has been rather limited implying that minority government ownership interests in the mining sector are not an efficient way to secure and collect a significant share of the economic rent and profits. Only four mining companies in Zambia made payments to ZCCM-IH based on the revenue stream of dividends and other investment dues (price participation fees) for the period 2013/2014 (Table 1).

Table 1: Payments (ZMW) of mining companies (ZEITI, 2014; 2015a)

Company	Revenue Stream	Amounts received (2013)	Amounts received (2014)
Kansanshi Mining Plc	Dividends	154,145,194	710,783,760
NFC Africa Mining Plc	Dividends	-	9,582,708
Chibuluma Mines Plc	Dividends	-	10,158,385
Konkola Copper Mines Plc	Dividend	17,174,000	16,545,000
Konkola Copper Mines Plc	Price Participation fee	85,595,105	-
Total		248,914,299 ($45.3m)*	747,069.851 ($116.7m)*

*Calculated using the respective exchange rates of K5.5/US$ (2013) and K6.4/US$ (2014)

Conflicts between equity partners in Zambian copper mining industry are ongoing. For instance, litigations were reported in which ZCCM-IH planned to claim as much as $1.4 billion from First Quantum Minerals (FQM). This is after ZCCM-IH accused the Vancouver-based company of fraud in which Kansanshi Copper Mines borrowed $2.3bn loan which ZCCM-IH claims could have been avoided (Bloomberg, 2016).

Compensation for infrastructure contribution - The Zambian Government compensation for its contribution to public infrastructure (roads, power and railways) used by mining companies is poor. IFC (2015) noted that if a country supplies a project with a service such as electricity or other infrastructure below its cost or market value, supplying this service will be a cost to the country. Zambia Business Times (2016) recounted that the copper mines in Zambia consume 55% of Zambia's generated electricity and these companies do not pay cost reflective tariffs thereby putting the burden on the national or state treasury. This indicates poor compensation for the country for its public infrastructural provision. Equally, most of the copper mines are currently using the road network in Zambia to transport their products. CUTS (2013) observed that there is a shift from trains to road traffic estimated to cost the Zambian economy US$100-150m per year in increased road deterioration and fuel costs.

Equity participation in Greenfield projects - The Zambian Mineral Policy of 1995 aimed in particular at encouraging private investment in exploration and in the development of new mines (Government of the Republic of Zambia [GRZ], 2013). Currently, lack of proportionality or equity participation by government in Greenfield projects like Lumwana and Kalumbila and other viable mining projects involved in mineral processing means loss of extractive industry revenues for the country in form of dividends.

Compensation for depletion of resources - Mineral deposits are gifts of nature that are contributed by government and society. These contributors are neither fully compensated for use of the deposit nor share completely in the surpluses from the mining activities. Manley (2013) presented broad guidelines to mining tax policy that are relevant to Zambia which policymakers need to consider. This requires that the state should be compensated for a loss of wealth which is a concern for most governments to obtain a fair compensation for the depletion of their non-renewable mineral resources.

Value of contributed mineral reserves - Although mineral reserves are important to mining companies, the practice is that value attached to these subsurface resources is not reflected in the balance sheets of mining firms causing some complications to quantify the contributions from each party. PricewaterhouseCoopers [PWC] (2007) reported that in International Accounting Systems (IAS 16), property, plant and equipment excludes application to mineral reserves. Despite the International Accounting Standard Board (IASB) considering the accounting treatment for mineral resources and reserves as part of its extractive activities project, in the absence of a specific standard, mining entities usually recognise mineral resource and reserve assets on the balance sheet at historical cost. This value could even be a lot lower than the market value.

3.2 Parity concept

Balancing infrastructure needs in mining and non-mining areas - There is a challenge in properly developing rent allocation or balancing of resource revenue distribution between resource and non-resource rich regions. While the Zambian government priority is to provide investment of public or social infrastructure in mining and non-mining areas, there are still discontents by local communities in areas where mining activities occur. The communities claim that they do not receive appropriate benefits based on the environmental and social costs they bear from mine development activities when compared to levels of development noted in non-mine areas.

Reinvestment needs - Mining companies as part of the sector similarly need to receive portions of the allocation (surpluses) to create additional mineral wealth through investment in mineral exploration and other marginal projects. Eggert (2010) reported that successive exploration and development can lead to; profits for mining companies, jobs for local communities or nation, new infrastructure needed for regional economic development, and increased government revenues that can be invested in social priorities such as education, health care, and poverty alleviation. According to International Council of Mining and Metals [ICMM] (2014), the collapse of the state-owned ZCCM was linked to falling income and lack of capital to reinvest which resulted in the fall of copper production. The company's (ZCCM) operations became increasingly unprofitable.

Expectations of local communities - Local communities have high expectations of surplus allocation needed to improve health, schools and poverty reduction projects. However, this is a parity concern not fully realised in Zambia. This is because the mining companies are not legislated or compelled to contribute to local development. They do it voluntarily to uphold and represent their CSR commitments. This situation is consistent with Hamann and Kapelus (2004) who noted that though these CSR initiatives are commendable, they have not been able to fundamentally introduce accountability and fairness into the development process.

Public Financial Management (PFM) systems - Effective PFM and expenditure management are critical to transforming revenue from natural resources into broad-based sustainable economic and social development (ICMM, 2014; ZEITI, 2015a). However, the Public Finance Act of 2004 of the Republic of Zambia states that a consolidated fund be established into which all general revenues and other public moneys accruing to the treasury shall be credited. The contributions by mining companies, therefore, lose their identity once they are deposited into the consolidated fund. Their use cannot therefore, be tracked to public investment/expenditure or to expenditure units/cost centres or project (ZEITI, 2015a). This means that revenues from the mining companies are not ring-fenced (based on the use of consolidated fund) and end at the discretion of the central government in terms of allocation for investment priorities.

Cost-benefit analysis - This is difficult to determine in many jurisdictions in that it is unsuccessful in providing adequately a framework needed for active allocation of rents from the sector in line with the needs of the various parties. According to IFC (2015), investments will likely run into problems at some point in their life cycles if there are imbalances in the sharing of fiscal, economic, environmental, and social costs and benefits. Revenue distribution should be decided through equitable decision-making structures involving representatives of the affected stakeholder groups (MMSD, 2002). Zambia's engagement in consultative agreements to decide the allocation of rents to parties affected is not fully accomplished. In jurisdiction like Canada, Söderholm and Svahn (2014) observed that community development agreements have been frequently used to help increase the local capture of benefits and the involvement of indigenous peoples. Such Canadian agreements suggest that mining can contribute to the socio-economic well-being of communities, as well as to the establishment of industrial clusters centred on activities related to mining.

Trust deficit in rent distribution - Lack of trust and transparency in resource revenue distribution to different parties exist. Tilton (2004) gave arguments from an economic perspective for increasing taxes on mining because of many mining companies not paying enough taxes. The argument was that too much of the wealth created by mining goes to mining companies, and too little to the State to promote economic growth and development. Equally, Cawood and Minnitt (2002) noted that the distribution of wealth may be perceived to be inequitable because of the different bargaining powers amongst stakeholders.

For Zambia, mistrusts in rent allocation are regular. The Finance Minister (Times of Zambia, 2015) hinted that Zambia made various changes to tax policies in the last 10 years with a view to optimise benefits from the mines which have not yielded the desired results. This resulted in the government changing the tax regime in January 2015 as a solution to have tax policies that guarantee a win-win situation by curbing all forms of tax planning schemes such as transfer pricing, hedging and trading through 'shell' companies.

3.3 Priority concept

Prioritising least well-off communities - In certain instances**, g**ranting a lot of wealth distribution priority to least well-off communities can result in reduced or eliminated incentives for mining companies to create the surplus (profits) needed for sharing. Brealey (2011) mentioned that a corporation has goals to increase the current value of the company's shares and the wealth of its stockholders including profit maximisation as part of the goals of the firms.

Zambian mining companies operating marginal projects need to generate enough profits in order to remain competitive. Competitiveness is entirely a matter of the cost of production and of transporting the product to market (World Bank, 2011). Challenges in the Zambian mines include high transportation costs, labour costs, input costs and electricity charges (ICMM, 2014) which are a function of the nature of the resource (the quality of the ore, the depth it is found) and the extent to which the most accessible resources have been exploited (World Bank, 2011). Based on these needs to address costs and productivity issues in different Zambian mines, any profits generated should be well balanced between sustaining profitable mine operations and rewarding the least-well of communities in the mine areas. As designated by ICMM (2014), the former state owned mine ZCCM collapsed on account of falling income and the costs of expanding social responsibilities done at the expense of reinvestment.

Compensation of parties affected by negative externalities - Parties who bear environmental and displacement costs are not prioritised and sometimes not well compensated from the mineral projects' proceeds. Davis and Tilton (2005) observed that few would dispute that most of the environmental and other social costs of mining are inflicted on the local community, while most of the rents realised by the country flow elsewhere.

In most cases, the mechanisms that have been used for compensating the displaced people are cash compensation and sometimes this involved land for land. Cash compensation for occupants owning land meant that they eventually lose their livelihoods (Centre for Science and Environment [CSE], 2011). The compensation in most cases have been inadequate which have been one off payments not enough to be converted into sustainable livelihood projects for the displaced. Simutanyi (2008) observed that the Chinese-owned Non-Ferrous Corporation Africa (NFCA) embarked on a project to build a smelter in Chambishi. The firm applied for and was awarded 1,000 hectares of land, which was previously inhabited. The company paid an equivalent of US$130, to each of the affected families as compensation, which many independent observers considered inadequate. Reasons for under compensation are attributed to poor and subjective cost-benefit analyses that fail to attach relative value to concerned groups in society.

Economic incentives - The mining companies embrace economic motives to make decisions on redistribution of income from the rich to the poor and these companies follow such decisions mindful of not undermining their profitability. McMahon and Moreira (2014) noted that most companies either voluntarily or due to government legislation or pressure provide other benefits to local communities. However, in Zambia the mining companies are not compelled and give priority in terms of contribution to local development voluntarily, without government legislation, through CSR undertakings.

Sharing of mineral royalty taxes - The fiscal policy dealing with allocation of taxes and royalties by the state to communities is not well followed. It is also uncertain what proportion of funds appropriated by the government from the exploitation of mineral resources flow to local communities where mining activities occur. This is despite the stipulation in the Mine and Mineral Development Act of 2008 about the sharing mechanism of mineral royalties with the communities which was not implemented. Even before this mineral royalty sharing mechanism for distributing revenues could be actualised in Zambia, this section has been removed from the new Act of 2015 (ZEITI, 2015a).

National Decentralisation Policy - According to MMSD (2002), a proportion of the benefits, such as revenue received in royalties or taxes, needs to be distributed through local administrative structures to enable them to take advantage of some important development opportunities for communities. In Zambia, GRZ (2002) embarked on the National Decentralisation Policy officially launched in 2004 which set out greater responsibilities for local government in delivery of public goods and services (ZEITI, 2015a). This 2002 National Decentralisation Policy has not been effectively implemented, and service delivery at district/sub-district levels is still provided predominantly through the various structures of central line ministries (ICMM, 2014). To date, these reform initiatives in Zambia leave subnational levels without the autonomy to execute decisions.

Local content - Riesco *et al*. (2005) in their studies observed that in addition to gaining hard currency from taxes and royalties, benefits from mineral development should include employment, infrastructure such as roads and hospitals, linkages upstream to industries that supply goods and services or downstream to industries that process mineral outputs, and technology transfer.

In Zambia, the practice of local content does not give much precedence to local input suppliers. Africa Progress Report (2013) noticed that Zambia suffers from a dearth of practical measures aimed at encouraging the development of local firms. Since privatisation of the mining sector, a number of local businesses have found that the new mine owners are less interested in buying from Zambian firms (Christian Aid, 2007). The industrial base in the country is inadequately developed which precludes local businesses from realizing a reasonable share of benefits from the mining sector. ICMM (2014) estimated that from four mining companies, the total industry procurement of goods is likely to be around US$1.75 billion annually, of which 5 per cent (or US$87million) represents locally manufactured goods.

Participation in social investment - Some relevant decision-making on social investment projects are exclusively determined by mining companies without full involvement of the local communities in Zambia. This leaves local communities in Zambia to have marginal participation in matters of getting involved in advancing and implementation of some social investment projects. The country has no sound legal framework outlining the rights of the local communities in matters to deal with mineral exploitation or resource exploration in inhabited areas. Occasionally, the only input emanating from communities is restricted to consultations during the Environmental Impact Assessments (EIAs) prior to mineral project development. However, only a few mines in Zambia have set up mechanisms to allow locals to participate in decision-making regarding social investment which include initiatives like trust funds (ICMM, 2014).

3.4 Rent sharing tools

The revenues available for appropriation will depend on the government’s application of the combination of various tax instruments ideal to maintain a competitive tax bundle instead of focusing on any individual fiscal tool. World Bank (2011) observed a problem of failure by the government and the industry to agree on an operating regime that strikes an acceptable balance between the interests of the industry and its contribution to national prosperity. This includes, among others, determining tax levels and administering tax revenues. The fiscal instruments affecting the mining industry as identified by Kumar (1991) include royalties, corporate tax, additional profits tax, withholding taxes on interest and dividends, and government equity holding acquired on concessional terms. These are explained below.

Royalties - Royalty collection as indicated by Otto *et al*. (2006) is a payment to the owner of the mineral resource in return for the removal of the minerals from the land. Royalties are relatively simple to administer from an accounting perspective but require significant and highly specialized capacity for physical audit, including specialized skills in mineralogy to determine ore grades, volumes, and value (Halland *et al*., 2015). In Zambia, royalty is a share of the government and not related to the project affected people/communities.

Corporate income tax (CIT) - This is a tax on profits assessed as a percentage of the net profits of a project after deducting allowable expenses (NRGI, 2015b). Profit taxes can be harder to administer than royalties since they require monitoring of 'transfer pricing' within firms and setting sensible rules on the allowable pace of cost recovery and depreciation of assets (ICMM, 2007). Relying on CIT is not ideal as it gets offset for years by allowable tax deductions such that the country receives very little return needed for sharing.

Dividend from government equity - State or government equity participation is the share of the state in the distributed profits of a company (NRGI, 2015a). Equity participation without dividend pay-out makes the country fail to share the benefits from its mineral wealth. The Zambian government still faces some challenges to benefit from its mineral wealth based on the current equity stake arrangement. This is because certain mining companies do not pay dividends to the minority share holding company ZCCM-IH claiming to be perpetually loss- making. Similarly, the mining companies claim to carry the financial risk due to the current 'free carried' interest arrangement where ZCCM-IH does not contribute to capital structure in the privatized firms but maintains an equity stake.

Border taxes (import, export duties) - Customs/import duty is a tax levied on ZRA specified goods imported into Zambia. These are levied as import duties, although export duties may also be levied (e.g. levies of export duty on exports of copper concentrate from Zambia). Manley (2012) discussed that customs duty has two main functions for governments: good source of revenues, particularly in developing countries where other forms of taxation are harder for authorities to administer; and used as a tool of economic engineering, most often to protect domestic industries from foreign competition or to encourage certain activities.

Withholding Taxes (WHT) – The WHT is generally imposed on payments to non-residents who are sourced in the country imposing the tax (Conrad, 2012). The WHT rate in Zambia is 15% charged on rentals, bank interest, dividends, and management and consultancy fees (ZEITI, 2015a). Dividend paid by a mining company in Zambia holding a large-scale licence and carrying on the mining of base metals is taxed at 0% (Zambia Development Agency [ZDA], 2013).

Property and license taxes - Property taxes paid by mining companies are assigned mostly to local governments with fewer problems of tax evasion (Bauer *et al*., 2016b). The license taxes are charged for awarding the contract to explore/develop the resource (Fischer, 2007).

Excess profits taxes - Based on strong commodity prices, some governments have considered introducing a special tax ("windfall profit tax") on 'excess profits'. Excess profits taxes are ideal if well indexed to commodity price. The Zambian government recognised the potential of excess profits from its mineral wealth and therefore, imposed the tax instruments - Windfall Profits Tax (WPT) and Variable Profit Tax (VPT) in the 2008 fiscal regime meant to capture surpluses. To date, both these taxes are discharged in the fiscal regimes for Zambia.

In Zambia, the mining sector generates various revenue streams for government needed for sharing, most significantly royalties, CIT and pay as you earn (PAYE). Mining companies also pay other taxes, including local government (property) taxes, withholding taxes, and fees relating to licences and permits. For mining companies with government shareholdings through ZCCM-IH, payments are also made in the form of dividends or price participation arrangements (ICMM, 2014). The contributions of mining to total GRZ revenues, based on various taxation instruments are presented (Figure 1).

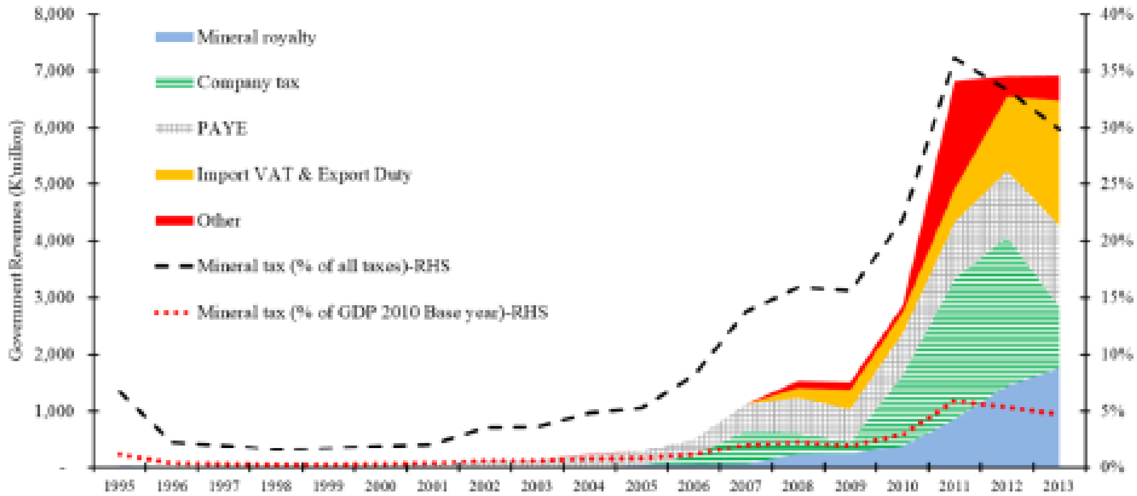

Fig 1: Tax instruments' contribution to total mineral revenue (Zambia Revenue Authority, 2014; Central Statistics Office, 2014)

In Zambia, the sector's contribution to governement reveune started showing improvements since 2007 due to increased production and rise in the commodity price. The mineral tax revenue as a percent of GDP has been low (less than 6%) for the past one and a half decade while the sector accounts for 4-12% of the GDP. Mine tax revenues for Zambia represent 1-14% of export revenues compared to 25-40% in the rest of the world as noted by World Bank (2011).

In terms of understanding the revenue appropriation, Conrad (2012) and Lundstøl and Isaksen (2018) made estimation that mineral royalty and corporate income taxes are the two direct taxes making a large payment to the mineral revenue in Zambia since privatisation. Pay as you earn (PAYE) is a tax paid by mine employees based on their emoluments and this tax is not considered as being paid by mining companies but employees. Africa Progress Report (2013) reported that the first EITI report in Zambia indicated that, between 2005 and 2009, half a million Zambians employed in the mining sector were carrying a higher tax burden than companies. Equally, import value added tax (VAT) is tax that is refunded to the mining companies and usually the refunds are larger than the payments received. This has led to disputes between mining companies and ZRA surrounding the VAT refunds.

Figure 2 shows the contribution (calculated in US$) made to GRZ budget including PAYE and dividend payout to ZCCM-IH. Most of these reveunes centralised appropriated in Zambia are not clearly distributed to the designated mining regions.

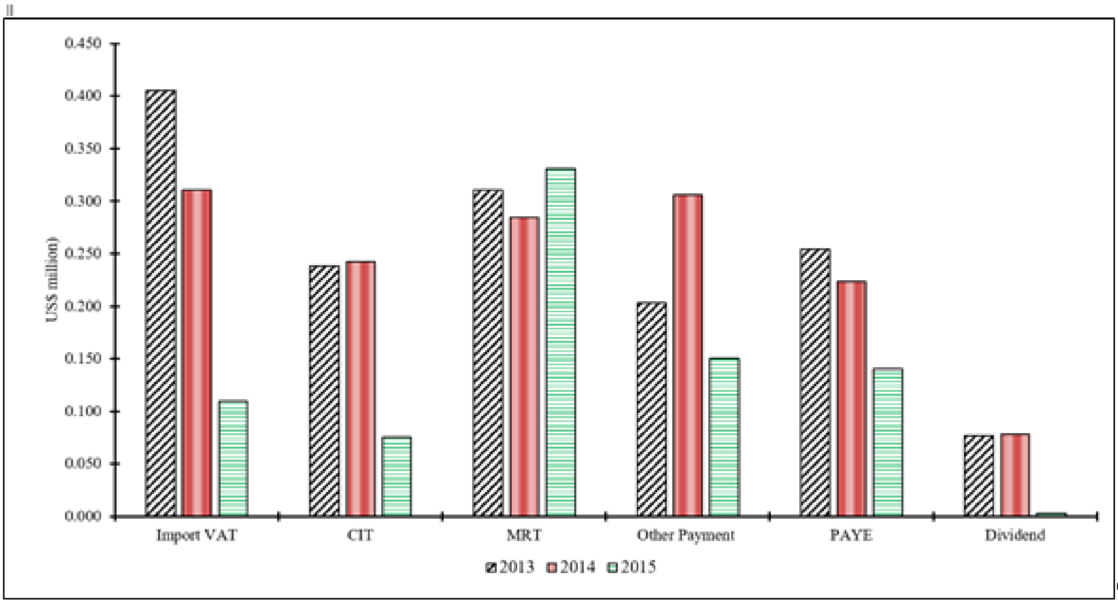

Fig 2: Contribution to GRZ budget from mining companies (ZEITI, 2015b)

3.5 Mechanisms ideal for rent distribution in Zambia

There is nothing termed as 'best practice' for benefit sharing from minerals. Every country has developed a mechanism to suit its ground realities (CSE, 2011). The following concepts could be contemplated ideal for Zambia's successful consideration of resource revenue distribution.

Vertical rent distribution - Resource revenue distribution on a top to bottom criteria has been the main practice for Zambia which has not worked very well. Söderholm and Svahn (2014) noted that a large part of the rent has been appropriated by the national governments, and the financial benefits that local communities could expect to receive have been those that eventually have reached the community through central government spending.

Zambia is not fully decentralised with respect to resource revenue collection. The central government through ZRA collects the vast majority of resource revenues contributing to national budget. These revenues are then distributed to subnational government and directly to local nationals in form of set priorities for infrastructure development. The local governments only collect some minor taxes related to property rates and land taxes. There is also no defined transfer formula of centrally collected resource revenue to various stakeholders. Against this backdrop, horizontal rent distribution of revenues needs to be supported by implementation of the outstanding National Decentralisation Policy to ensure that subnational authorities or local government are granted permissions to get involved in the collection and preservation of taxes.

Administrative capacities - Capacities at national and sub-national levels differ which influence rent collection and redistribution needs. Haglund (2013) noted the existence of weak capacity of the different government agencies involved in enforcing and administering revenue mobilisation from the sector. At an administrative level, local governments often do not have the capacity for effective management of the collection and distribution of revenue (MMSD, 2002). According to GRZ (2002), at local level, government realised lack of capacity to attain decentralised system of government and has therefore, de-concentrated some of its functions, powers and resources to provincial and district administration levels. This mechanism needs to be strengthened by enacting a National Decentralisation Policy.

Certain wealth distribution tools are more appropriate for national levels (CIT, royalties, dividends and WHT) than for sub-national levels (property taxes). Consistent with Bauer *et al*. (2016b), profit taxes are least likely to be collected by local governments due to the administrative complexities involved in calculating them accurately, including dealing with tax avoidance measures. Property taxes and licence fees are those most often assigned to local governments and these streams are relatively stable and predictable and there are fewer problems with tax avoidance.

Transparency, government policies and institutional capacities - These concerns influence equitable rent distribution in Zambia. Fischer (2007) noted that because of poverty and weaker institutional capacity, developing countries face greater challenges in collecting, managing, and distributing resource revenues. Transparency is only one tool among many that are needed to improve resource and revenue governance, particularly where institutions are weak. According to ICMM (2014), in the Zambian mining industry, the issues of governance have been attributed to weak institutional capacities dealing with policy formulation and regulatory framework.

Zambia should ensure that it sets its fiscal instruments through laws rather than individual contracts as a means to enhance transparency. This is in line with NRGI (2015b) observation that setting fiscal regimes through laws increases transparency and accountability, because contracts are more likely to be kept secret. Negotiations also bring additional opportunities for corruption or manipulation.

Cash payments - Cash payments appropriated by the central government in Zambia are usually influenced by the Public Financial Management systems as they are channelled through a Consolidated Fund. In some jurisdictions, in-kind revenue sharing is used whereby a portion of tax obligations can be spent on local infrastructure or social programs rather than paid in currency (Bauer *et al*., 2016a). Since companies can make mandatory in-kind payments in the form of infrastructure or health services, this practice represents a shift of benefits from tax collection to company expenditure on local projects (Bauer *et al*., 2016b).

Zambia should consider in-kind payments as an appealing form of benefit allocation to the communities. Although the gains could only be measured by physical infrastructure developed by the mining firms, the cost outlay for in-kind payment declared by mining companies are difficult to verify and remain a secret for these firms. The social payments and transfers made during 2015 by mining companies and non-copper mining companies (ZEITI, 2015b) are given in Table 2.

Table 2: Social payments and transfers made in 2015 (US$) (ZEITI, 2015b)

Company	Corporate Social Responsibility in Kind payments	Corporate Social Responsibility cash payments	Total
Copper Mines	328,901,294	44, 004,208	372, 905,502
Non - Copper Mines	8,907,790	4,601,355	13,509,145
Total	337,809,084	48,605,563	386, 414,647

Stabilisation funds - Mineral revenue management requires that stabilisation funds get created to cope with commodity price volatility (Fischer, 2007). Based on the volatility of resource revenues, some governments create a stabilisation fund that will receive deposits when prices are high and supplement the budget when prices are low (NRGI, 2015a).

The contribution of Zambia's mining sector to the economy and its development is very dependent on the movements in the world prices of copper and cobalt as well as exchange rates (UNCTAD, 2011). These volatilities in the commodity prices affect foreign exchange earnings and mineral resource revenues that are needed for equitable distribution. Zambia should consider establishing the stabilisation funds, as practiced in other jurisdictions like Canada, Chile, Norway and Botswana, as an additional measure to manage market volatility.

Investment funds - An investment fund is a fund that invests its money in assets that earn income, or that due to other strategies is able to increase its capital stock (Eggert, 2001). One widely used instrument to deal with the financial benefits resulting from revenue sharing is to allocate part of (or all) the revenues to a fund. The aim of the fund is to make the wealth created by mining permanent, and thus generate a financial source to support sustainable regional economic development for the future (Söderholm and

Svahn, 2014). These funds need to be created for Zambia to support its economic development.

A summary of ideal situations for consideration in the sharing of resource benefits (rents or revenues) for Zambia is given in Table 3.

Table 3: Consideration for benefit-sharing mechanisms in Zambia

Initiative	Benefit - sharing mechanisms
Revenue based royalty	Certain portion of royalties should be distributed or shared with the local communities to fund specific social and sustainable investment projects in the mining regions.
Local procurement	Government to legislate preferential public procurement practice as applied in government (under the Zambia Public Procurement Act) to coerce mining companies to grant preferences to Zambian products and services through local contracting and service provisions.
Stabilisation Fund	Zambia should set up stabilisation funds to receive commodity revenues that can mitigate the fluctuations in government revenues and export earnings because of commodity-price volatility experienced in the commodity markets.
Investment Fund	Government to consider allocating revenues to the investment fund as a means to grow savings and distribute future mining benefits to regional stakeholders.
Stakeholder consultations/cost-benefit analysis	The country needs to employ multi-stakeholder consultations in matters dealing with resource benefit sharing. With this approach, communities, companies and governments will congregate to discuss the ideal situations to generate responsibilities and related costs and benefits from the mineral resources.
In kind payment	Zambia needs to focus more on measures to employ in-kind payments from its mineral resources than cash payments. These would provide physical and non-fiscal benefits that include education facilities, medical facilities, employment goals, local procurement, training of staff and improved service access.
Horizontal rent distribution	Government should implement the National Decentralisation Policy to support sub nationals to carry out partial horizontal rent distribution in the related mining regions.

4. Conclusions and recommendations

The practicality of the insights in this research was to present distribution challenges Zambia faces in its sustainable management of resources from its mineral wealth. Emanating from this, conclusions were made that Zambia faces challenges in appropriation of resource revenues proportional to its contribution. The country also lacks well-defined principles of sharing resource revenues to various parties. The review further made conclusions that Zambia is still faced with failures to engage in multi-stakeholder consultations leading to rent allocation problems based on strengths and powers of some parties. Additionally, tax administration in the country based on used fiscal tools for rent collection and sharing is not fully accomplished by related government agencies because of weak institutional capacities. It is also concluded from this review that Zambia has varied circumstances with competing needs by various stakeholders from its mineral wealth resulting in no 'best fit' for allocation of rents among different parties.

The review recommended that government strengthens its institutional capacities to enhance the processes of collection and distribution of mineral rents and it should engage in full multi-stakeholder discussions on rent allocation processes. It is also recommended that state should employ proper cost-benefit analysis for all parties to assess the appropriate rent allocation mechanism and it must equally actualise National Decentralisation Policy to promote subnational horizontal rent distribution in mining regions.

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Recovery of Zinc and Iron from Electric arc furnace dust: An attainable region approach

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Abstract

This study investigated the application of the attainable region optimization technique to establish the optimum hybrid hydro and pyro-metallurgical process conditions required to treat the Electric Arc Furnace Dust (EAFD) and reclaim both iron and zinc. The EAFD from a steelmaking plant is typically a complex, fine aggregate material containing between 20 to 50 % iron and 18 to 30 % zinc and light particles of different metals usually that of aluminum, lead and cadmium. It is produced from mechanical movements of the furnace and non-metal inclusions made during the melting of scrap and refining of liquid steel. The particle size distribution (PDS) of the EAFD sample taken from Kafue steel plant showed the degree of fineness with half of its mass retaining the particle size of 150 µm while its particle distribution followed a triple-modal curve. The chemical composition showed that the EAFD sample contained 33.25% of iron oxide (FeO), 19.85% of Zinc oxide (ZnO) with the combined oxide of zinc ferrite proportionating at 30.62%. An Attainable Region (AR) method was applied to establish optimum parameters for iron and zinc recovery. A hydrometallurgical approach (leaching with NaOH) was the technique employed to treat the EAFD for selective zinc recovery, while a pyro-metallurgical method (carbothermic roasting) was used as the next method in reclaiming iron from its oxides in the leached dust residue. A combination of an agitation speed of 800 rpm, sodium hydroxide concentration of 8.0 mol/L and a leaching temperature of 80 ^{o}C was the optimum conditions for the hydrometallurgical process. The roasting temperature of 1200 ^{o}C, the carbon content of 35.27% and a roasting period of 36 hours were the optimum conditions for the pyro-metallurgical process.

Keywords: ***Electric arc furnace dust, iron, zinc, optimization.***

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1. Introduction

The production of steel using the Electric Arc Furnace (EAF) has become more prominent than any other process because of its flexibility in operations, low capital investment and ability to use up to 100% scrap (iron-containing) material as feed, Ruiz et al (2007). In an EAF, the iron-containing raw material is usually melted at high temperatures of up to 1600 ºC which enhances the elements to volatilize from the melted steel bath, Oustadakis et al (2010). In a continuous melting process, the volatilized particles enter the gas phase before being cooled and oxidized to form fugitive matter known as the Electric Arc Furnace Dust (EAFD), Guezennec et al (2005). The electric arc furnace dust usually contains a wide variety of elements, mostly metal oxides in spinel type. These metal oxides are generated by the oxidation of zinc, iron and manganese at high temperature and in the presence of iron (III) oxide (Fe_2O_3), Havlik et al (2006). On average, about 20 kg of steel dust produced from every tonnage of recycled steel is generated in different sizes and forms, Geldenhuis (2002), depending on the type of the process used, the kind of steel made, and the quality of raw material used. The composition of EAFD is therefore directly linked to the chemistry of the metallic charge (EAF feed) and refinery additives. The composition of the dust demands different processing methods, Havlik et al (2005), and the possible means of recycling valuable metals back to the steel-making process.

Effective processing of EAFD is usually achieved by the use of process optimization techniques and analysis tools. The attainable region method is one of the analysis tools used to solve optimization problems, Hildebrandt et al (1999). Analogous to the 'wanted region' in linear programming, the AR has guidelines for the construction of the achievable region and has some necessary conditions on how to obtain the optimum solution. The AR is defined as a set of all possible outcomes, for the system under consideration, that can be achieved using the fundamental processes operating within the system, and that satisfies all constraints placed on the system, Asiedu et al (2014). The method uses a set of values of all the output variables that can be achieved by any possible process using a given feed. These output variables are then applied to the AR technique in order to define a possible geometric space within which the achievable points can be found, Hlabangana et al (2016). This study aimed to establish hybrid hydrometallurgical and pyro-metallurgical process conditions required to treat the dust using the AR method. The study investigated the possible recovery of iron from EAFD by leaching the dust to dissolve zinc, with iron remaining as the residue. Leaching conditions such as leaching reagent concentration and temperature were crucial points of the study. The objective of leaching was to establish a necessary environment for best zinc yield by allowing more zinc to get into the solution with minimized iron dissolving. The chemistry and kinetics of leaching process of EAFD were also studied with the goal of establishing the type of a reaction taking place between the oxides of zinc and sodium hydroxide. The study also focused on the thermodynamics of iron reduction during energy-controlled reactions and investigated the carbothermic reduction of iron when complex iron oxides reacted with low-rank coal.

2. Materials and methods

2.1 Chemical Composition and Particle size analysis

A prepared sample of the EAFD was dried for 30 min in an oven set at a temperature of 50 °C.

Cone and quartering was then performed on the dried sample in order to split it into smaller 100 g samples for sieve analysis. The homogenized 100 g sample was then added to the stack of eight clean sieves, 200 mm in diameter, arranged in descending order of mesh sizes from 350 µm, 240 µm, 150 µm, 100 µm, 72 µm, 52 µm, 36 µm and the pan below, Hlabangana et al (2017). The stack was then mounted on an electrically powered mechanical sieve shaker, and wet sieved for 20 min. The elemental chemical composition of EAFD was determined using Atomic Absorption Spectrometer (AAS220), and EDS G100 spectra was used for metal oxides.

2.2 Leaching of EAFD using sodium hydroxide

Leaching of the EAFD was carried out in three stages with the first stage involving the determination of optimum agitation speed required for effective leaching of Zinc at the constant temperature of 65°C, hydroxide concentration of 2 mol/L and the liquid-solid ratio being kept constant at 10 ml/g for all the leaching tests. This was in accordance to the AR optimization technique applied that requires some parameters to be fixed while those that needed to be optimized are varied. The effect of agitation speed on a rate of zinc leaching was determined in the range of 400 rpm to 1000 rpm at 100 rpm interval. The second leaching stage was done to establish the optimum concentration of sodium hydroxide required to leach the dust effectively, therefore the agitation speed, leaching time and solution temperature were all kept constant at 800 rpm, 110 minutes and 65°C, respectively while sodium hydroxide concentration was adjusted from 2 mol/L, 4 mol/L, 6 mol/L 8 mol/L and 10 mol/L. The third leaching stage involved changing the sodium hydroxide solution temperature from 25°C to 100°C. This was aimed at determining the optimum leaching temperature required at optimum agitation speed and specific leaching time. Investigations were done on the rate of leaching at optimum agitation speed, leaching temperature and leaching concentration.

2.3 Carbothermic Roasting of EAFD

Carbothermic treatment of the EAFD leach residue was achieved by preparation of carbon-containing material (coal), followed by determination of its moisture, volatile matter and fixed carbon content. This was followed by blending the fine (- 75 µm + 25 µm) milled coal with EAFD. The first stage of roasting was to determine the optimum roasting time and the rate of iron oxide reduction; therefore, the sample was heated 3, 6, 12, 36 and 48 hours at a temperature of 800°C. The second stage of carbothermic leaching involved elevating the roasting temperature for the maximum timeframe. The procedure was repeated for temperatures 800, 1000, 1200 and 1400°C respectively. The samples were analyzed using the AAS 220 spectra after each heating interval.

3. Results and Discussion

3.1 Chemical Composition and Particle size distribution results

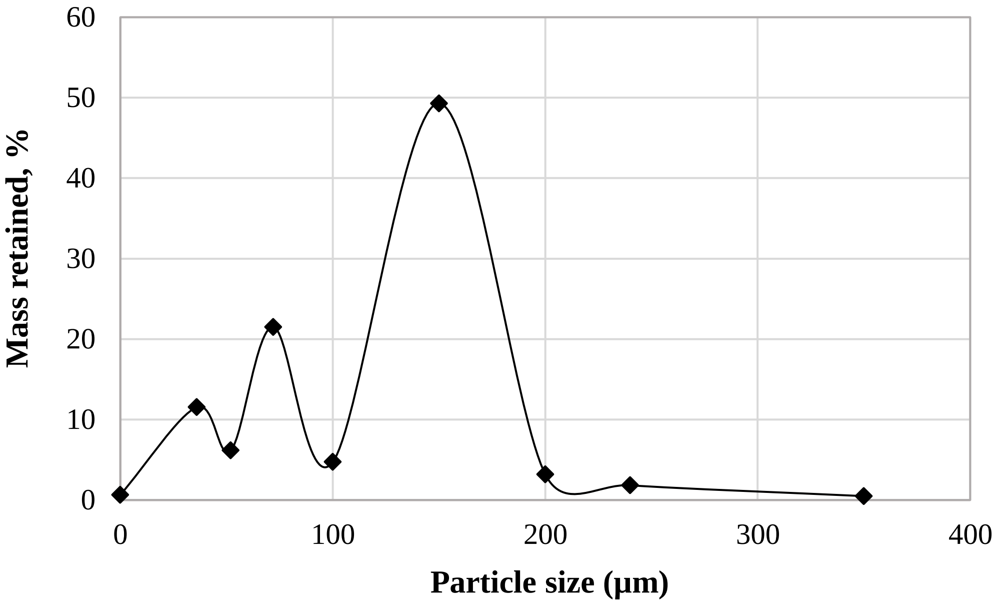

Figure 1: Particle size distribution (PSD) profile

Figure 1 shows a particle size distribution (PSD) plot of an EAFD sample used as feed material in this study. It can be seen from this plot that the mass of particles retained on each sieve size expressed as a percentage of the total mass (100 g) of the sieved sample is plotted against the sieve size. The figure reveals that half of the mass of the dust was retained on the 150 µm sieve which indicates the degree of fineness of the EAFD. The degree of fineness of the EAFD gives a clear picture of how the surface of the particles per unit volume shall affect the rate at which individual particles can participate in a chemical reaction, (Ujam and Enebe, 2013). For example, during a liquid-solid phase contact reaction, the resistance to flow is dominated and influenced by the size and shape of the particles. Larger and spherical particles flow easily than the high aspect ratio particles because the bigger the particle the higher is the suspension viscosity, Guezennec et al (2005).

Table 1: Chemical Composition of EAFD

% FeO	% ZnO	% SiO	% CaO	% $ZnFe_2O_3$
33.25	19.85	9.70	5.10	30.62

Table 1 shows the chemical composition of EAFD confirming the presence of iron and zinc in higher proportion. Table 1 is also represented in Figure 2, which shows an attainable region (AR) plot of the main chemical composition of the EAFD and the relative abundance of these chemical components in the dust. The data points are discrete, hence their connection using a dotted and not a continuous line. The dotted line that joins the data points in Figure 2, is not a model fit to experimental data but was added in order to make the trend easy to follow. The AR plot shows that the oxides of iron (28.62%), zinc (19.85%) and their combination (35.25%) when added together give a total of 83.72% of the chemical composition of the dust.

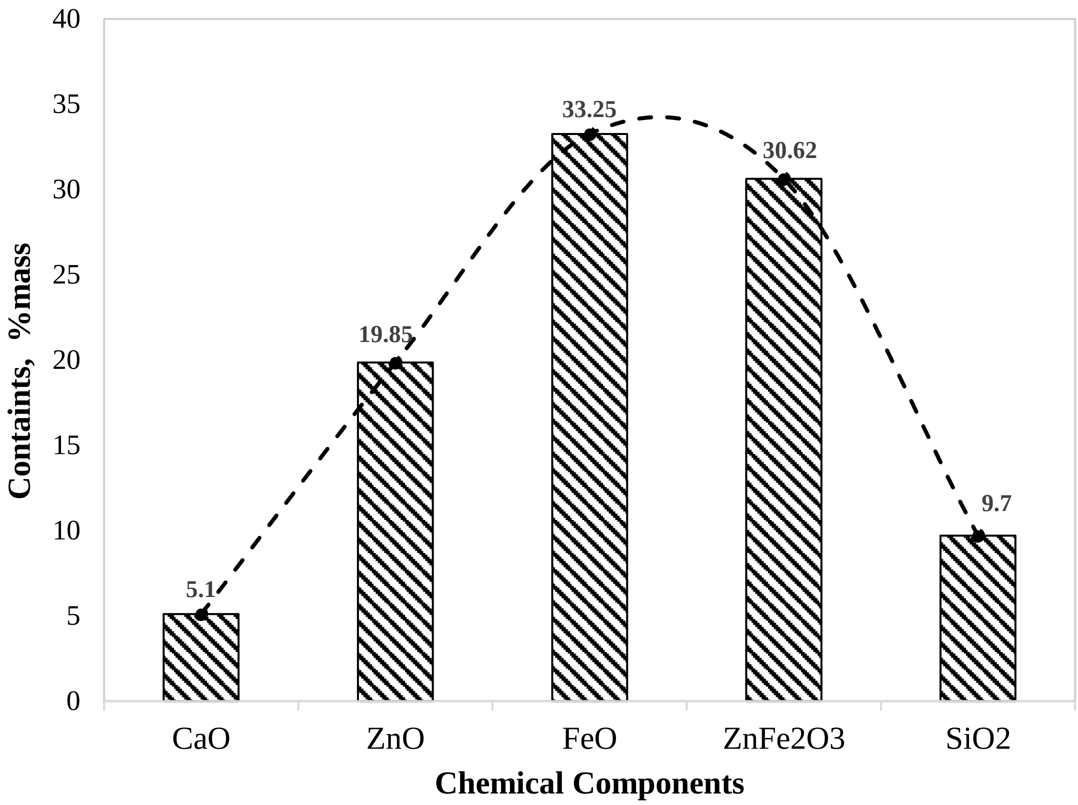

Figure 2: AR-Curve for chemical composition of EAFD

This chemical analysis provided the statistics that justified the treatment of EAFD for the recovery of both zinc and iron. The presence of zinc in oxide form demands use of a hydrometallurgical method in order to recover the metal, while the presence of iron in oxide form demands the use of a pyro-metallurgical technique to reclaim the metal. Since both oxides are present in sizable quantities in the EAFD, the two treatment methods were applied consecutively to process the dust for both zinc and iron metal recovery.

3.2 Leaching and Kinetics of Zinc dissolution

Figure 3 depicts an AR plot of the effect of agitation speed on the leaching efficiency of zinc. A constant solution temperature of 65ºC, the residence time of 110 minutes, NaOH concentration of 2.0 mol/L with varying speeds of agitation used as the experimental conditions. The results show that as the speed of agitation increases, the efficiency of zinc extraction increases up to a limit value and then start decreasing. The increase in stirring speed reduced the thickness of the slurry layer and therefore increased the eddy diffusion of the material which subsequently transferred the particles to the bulk of the solution. The AR plot shows that the optimum agitation speed for maximum zinc recovery was 800 rpm, under the experimental conditions tested. As the speed of agitation increased beyond 800 rpm, the contents of the vessel started to centrifuge, thereby reducing the intimate contact between the lixiviant and the filtrate. This reaction condition negatively affected the extraction efficiency of zinc.

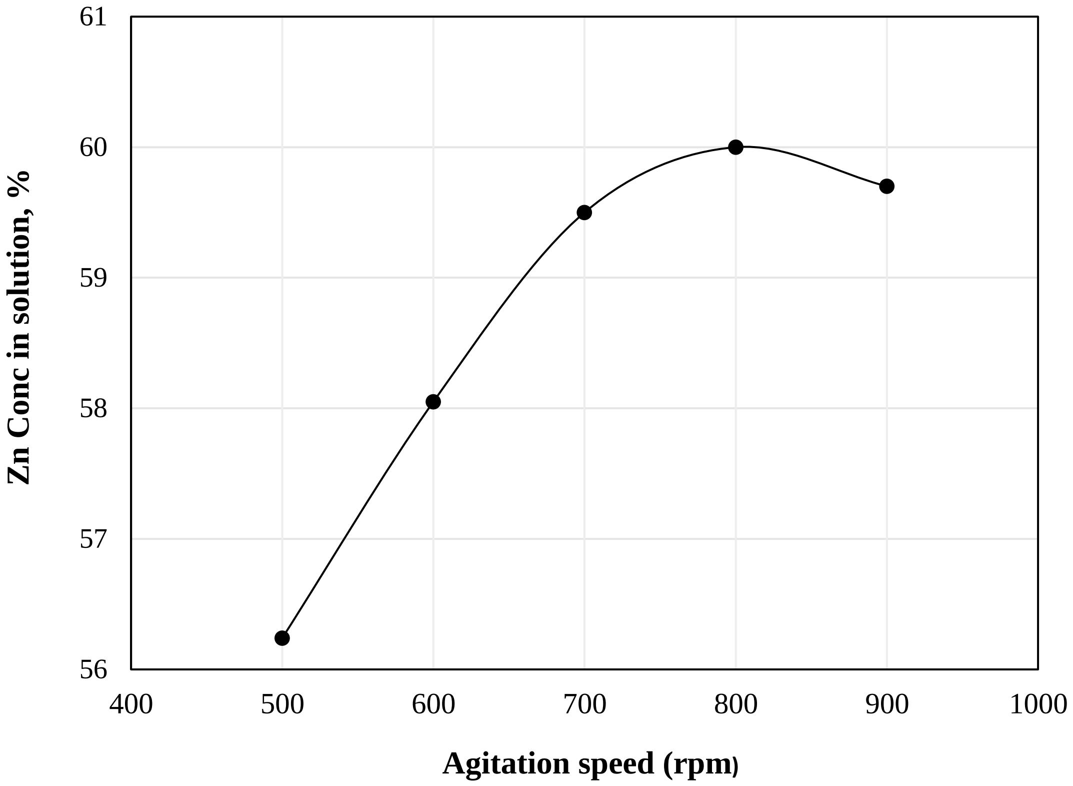

Figure 3: Effect of agitation speed on Zinc dissolution

The AR plot in Figure 4 shows the results of leaching the dust with different concentrations of sodium hydroxide, using an agitation speed of 800 rpm, solution temperature of 65ºC and a residence time of 110 minutes. The area bounded by the curve and the x-axis represents the various combinations of the concentration of zinc in the solution obtained from leaching with a specific concentration of NaOH solution. The leaching efficiency of zinc increases with increase in the concentration of the alkaline solution used, up to a limit. A further increase in solution concentration increases the viscosity of the leaching solution thereby reducing the solid-liquid separation efficiency. The AR plot shows that for the objective function of maximizing the concentration of zinc in solution, the optimum concentration of NaOH to be used to achieve this is the one that corresponds to the turning point on the curve. Therefore, a NaOH concentration of 8.0 mol/L was required to obtain a maximum concentration of zinc in a solution of 65.5%.

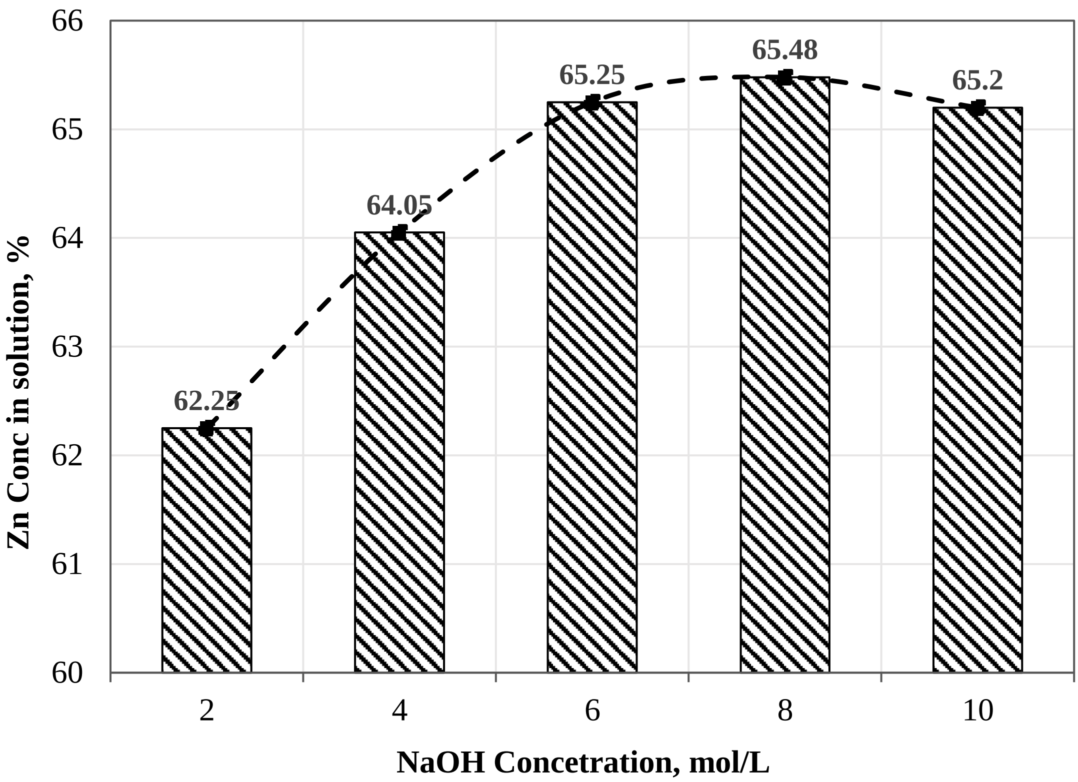

Figure 4: The AR – Curve for the effect of NaOH concentration Zinc dissolution

Figure 5 shows an AR plot obtained to investigate the effect of temperature on leaching the EAFD at an optimum agitation speed of 800 rpm, NaOH concentration of 8.0 mol/L and a residence time of 110 minutes. According to the guiding principles of the AR optimization technique, the area bounded by the curve and the *x*-axis represent all the possible coordinates of leaching temperature and concentration of zinc in solution, under the experimental conditions investigated, Havlik et al (2006). The optimum coordinates for achieving the desired objective function are found on the peripherals or boundaries of the attainable region. For the desired aim of maximizing the concentration of zinc in solution, the AR technique reveals that an optimum temperature of 80°C results in a maximum zinc concentration of 75.25% in solution.

The rate of leaching EAFD was investigated by considering the dissolution of ZnO directly from the suspended zincate particles and from the complex zinc ferrite. The reduction of zinc from the EAFD and the increase of Na_2ZnO_2 in the aqueous solution followed the second order reaction rate;

$$\frac{dC_{zn^{2+}}}{dt} = k(C_{zn^{2+}n} - C_{zn^{2+}})^2 \qquad (1)$$

Where $C_{zn^{2+}}$ is the concetration of zinc going into solution at a particular time.

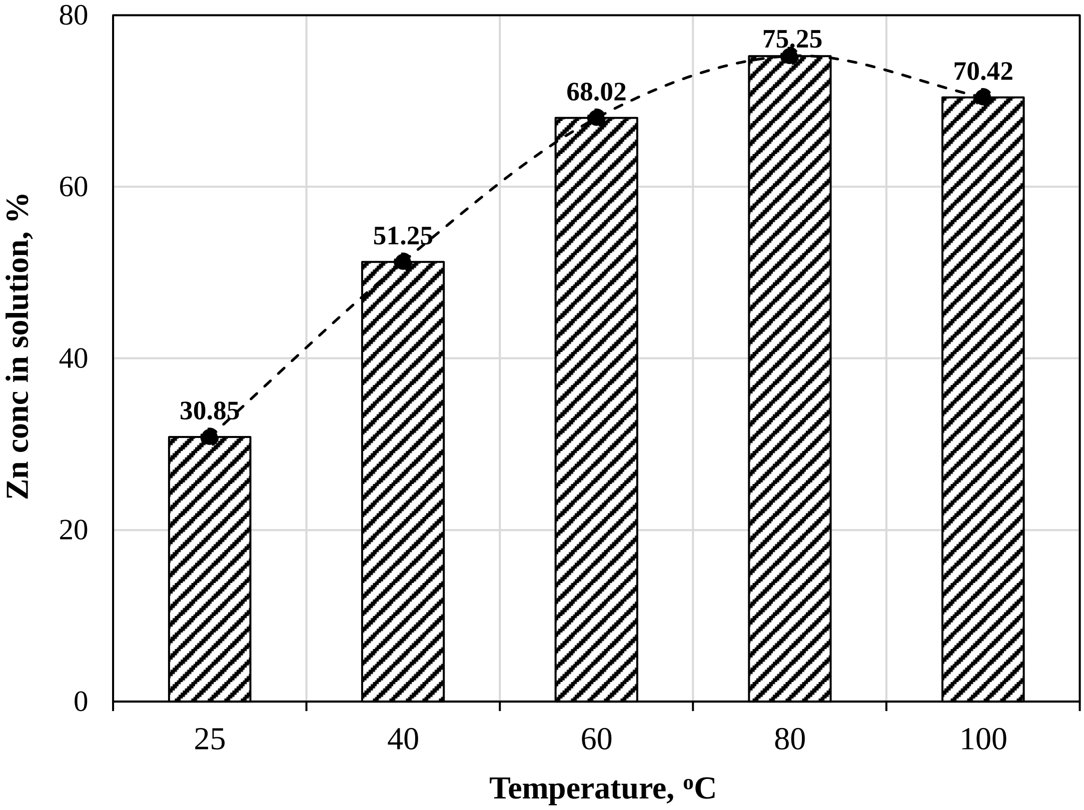

Figure 5: The AR – Curve for the effect temperature on Zinc dissolution

The function is the concentration of zinc in saturated concetration of the leach solution. At a fixed leaching time, *t*, the leaching rate, *k* follows a shrinking core model and therefore the values of *k* for each zinc fraction ratio during leaching process can be calculated using equation (2);

$$1-(1-C_{zn^{2+}})^{\frac{1}{3}} = \oint_0^t C_{zn^{2+}}\,dt = kt \qquad (2),$$

Where the leaching rate constant *k* is expressed as:

$$\mathrm{k} = k_n exp\left(\frac{-ER}{T}\right) \qquad \text{(3), Junca et al (1999)}$$

and k_n is the temperature dependent coefficient factor, E the activation energy, R is the gas constant and T the absolute temperature.

Taking natural logarithm for equation (3) on both sides, the leaching kinetics equation for Zinc is expressed as shown in equation (4);

$$\ln k = k_n + \left(\frac{-E}{R}\right)\frac{1}{T} \qquad (4)$$

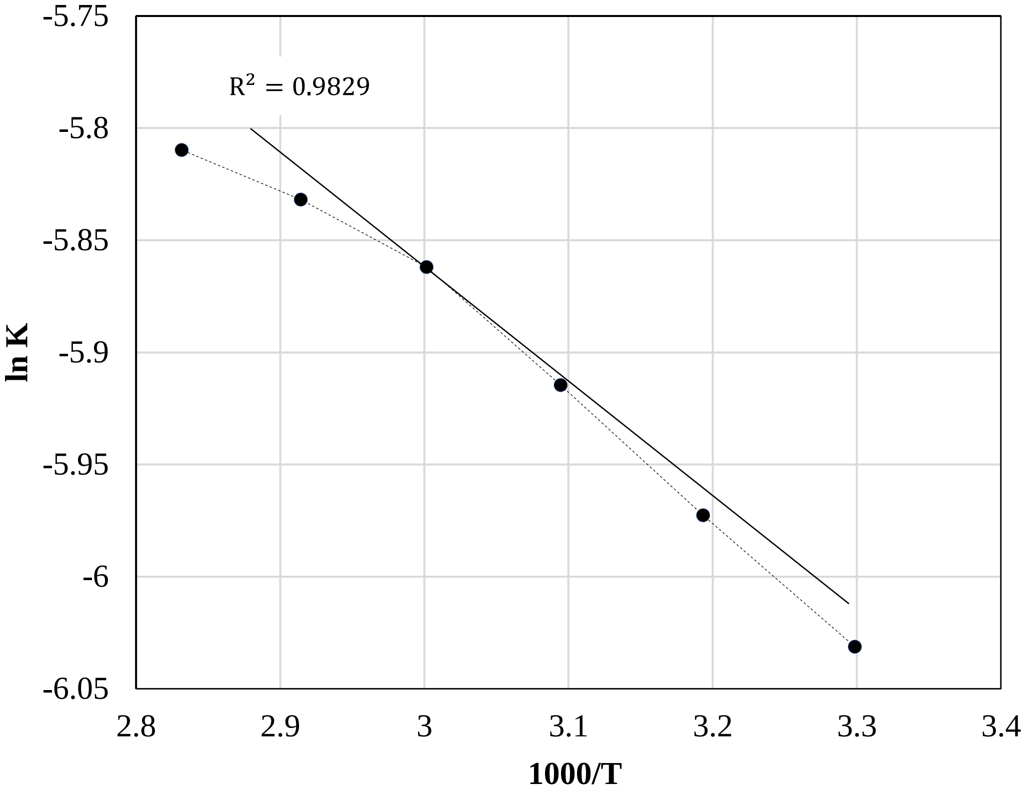

Figure 6. Arrhenius graph for leaching Kinetics of EAFD

Figure 6 is a plot of equation (4), in which the leaching rate plotted against the reciprocal of absolute temperature. The linear plot in Figure 6 implies that the leaching kinetics of the EAFD follows a linear second order relationship and the apparent activation energy was calculated to be 7.069 kJ/mol. This plot also indicates that the leaching of zinc from EAFD was an endothermic reaction and the chemical reaction controlled the rate between 30 and 80°C.

3.3 Carbothermic roasting of EAFD

Figure 7 shows attainable region plots of the metallization of iron from the leached residue against roasting time for a fixed carbon content of 25.04 % and under different roasting temperatures. As observed from Figure 7, the amount of iron metallization increases with an increase in the roasting time and roasting temperature, up to a limit, beyond the turning point (1200 °C), the product (iron) and the by-products (carbon monoxide) begin to react, thereby shifting the position of equilibrium towards the reactants. This phenomenon is also observed in all the plots presented in Figure 8. The AR plots indicate that to archive the objective function of maximizing the amount of iron in the roasted residue, a roasting time of 36 hours and a roasting temperature of 1200°C were the optimum parameters to employ, under the experimental conditions tested. Figure 8 also shows attainable region plots of the concentration of iron in the roasted leach residue against the concentration of carbon in the coal that used as the reducing agent.

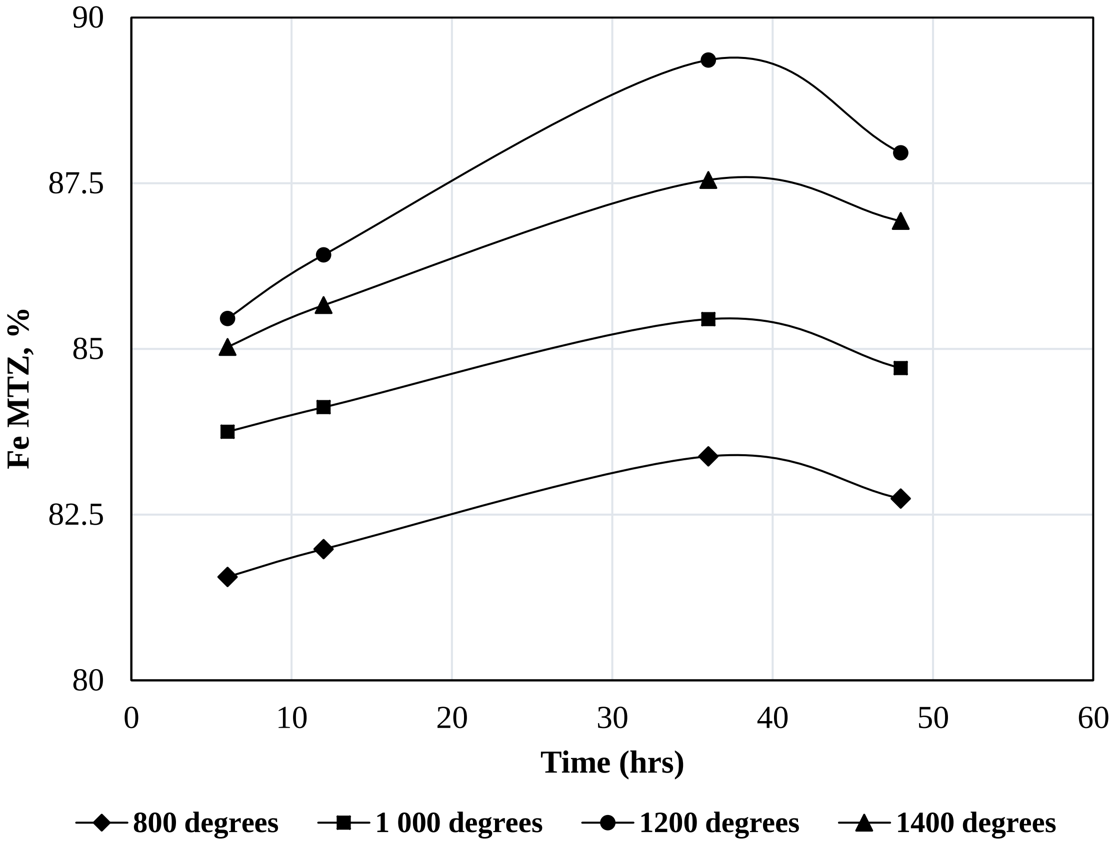

Figure 7. Metallization of Iron in leach residue

Although the concentration of carbon in the coal was fixed, the kinetic reactions of carbothermic reduction of oxides of iron from the EAFD depended mainly on the rate at which the iron oxide was converted to metallic iron by coal. The reaction involves various reduction steps which are controlled by diffusion and phase-chemical boundaries. The diffusion-controlled reaction consists of the penetration of the gaseous carbon into the iron oxide particle layer and the release of product gas outward through the iron layer. Since the reaction was chemically controlled, the diffusion rate expression is described by equation (5), Kukurugya (2015);

$$1 - \frac{2}{3}Fe^{2+}(1 - Fe^{+2})^{\frac{2}{3}} = k_T t \qquad (5)$$

$$1 - (1 - \%\mathrm{Fe}^{2+})^{\frac{1}{3}} = \oint_0^t \%Fe^{2+}\, dt = k_T t \qquad (6)$$

Where: %Fe is the percentage by mass of iron reduced during the reaction, t the roasting temperature and k_T the temperature dependante reaction coefficient. The chemical boundary-controlled reaction involves the diffusion of the reducing gas (in our case carbon monoxide) into the boundary across the outer layer of the iron oxide particle, Kim et al (2006).

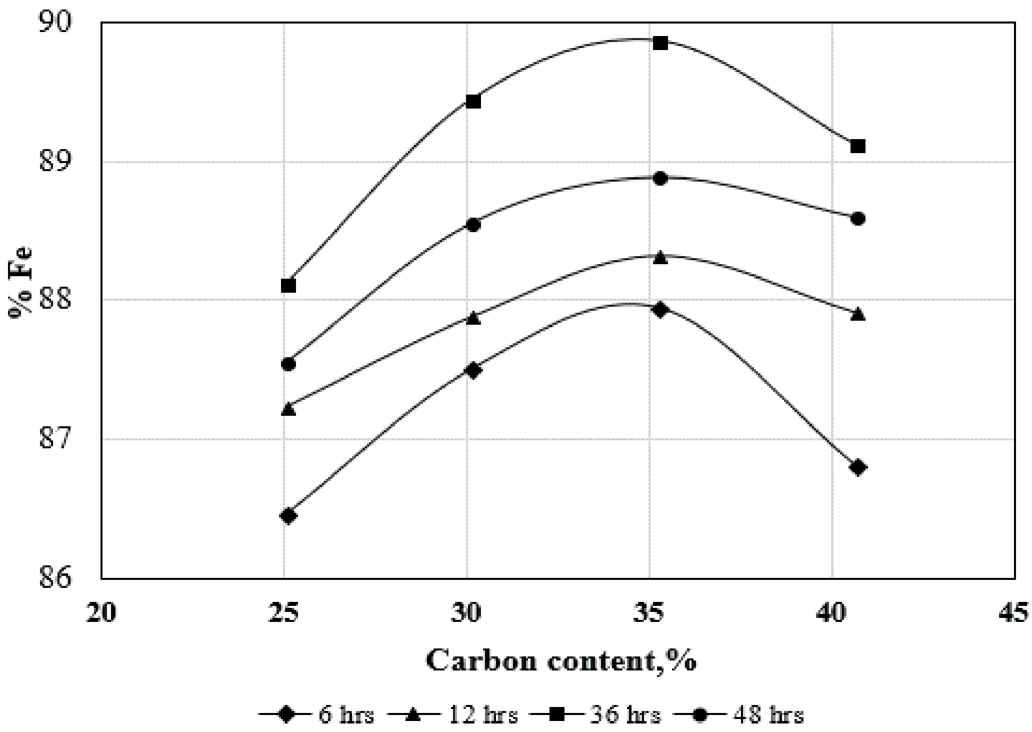

Figure 8. Concentration of iron in the leach residue vs. coal carbon content

If the plot of percentage of iron oxide reduced to metallic iron and roasting time *t*, is linear, then the rate of reaction is chemical reaction controlled and the values of the rate constant *k* can be obtained from equation 7;

$$\mathrm{k} = k_n exp\left(\frac{-ER}{T}\right) \quad (7)$$

Where: k_n is the temperature dependent coefficient factor, E the activation energy and *R* is the gas constant, Kukurugya, (2015).

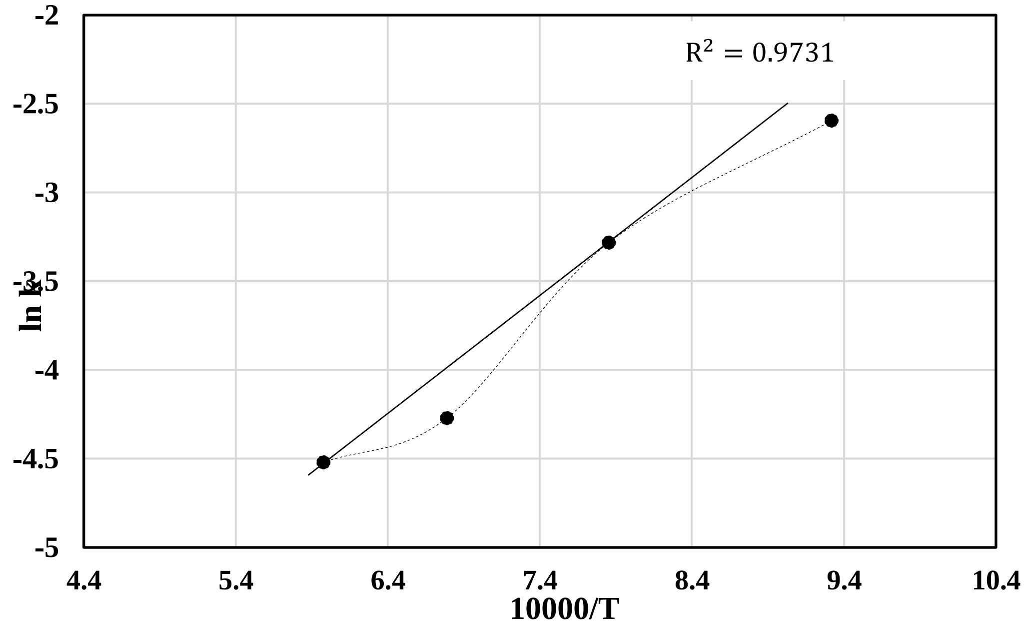

Figure 9. Arrhenius graph for roasting of Zinc-leach residue

Figure 9 is a plot of equation (6) and shows the relationship between and roasting time for a temperature range from 1073.15 K to 1673015 K. From Figure 9, it can be observed that this relationship is linear and therefore k_T can be expressed as in equation (7) from where the values of k_T at particular reduction temperature can be evaluated. The activation energy of the carbothermic reaction is calculated from the plot of and the reciprocal of absolute temperature. The activation energy was calculated to be -5.06 kJ which implies that the reaction was exothermic and chemical reaction controlled.

4.0 Conclusion and Recommendation

4.1 Conclusion

The chemical composition analysis results indicated that 43.79 % of iron and 24.46 % of zinc were the major constituents of the EAFD. The dust also contained complex zinc oxide compounds with a combined percentage proportion of 30.62 %. The particle size of the EAFD was also determined, and it was established that more than 50 % of the particles had a particle size of 150µm.

The attainable region AR technique used on the experimental data indicated that a combination of an agitation speed of 800 rpm, sodium hydroxide concentration of 8.0 mol/L and a leaching temperature of 80°C was the optimum conditions for the hydrometallurgical process recovering 75.25% of Zinc. For the pyro-metallurgical process, the optimum conditions were a roasting temperature of 1200°C, carbon content of 35.27% and a roasting period of 36 hours with 89.36% of iron metallization.

The leaching kinetics of the EAFD was established and indicated that leaching of EAFD with sodium hydroxide followed a linear first order relationship and the apparent activation energy was calculated to be 7.069 kJ/mol, implying that the leaching of zinc from EAFD was an endothermic reaction and the chemical reaction controlled the rate between 30 and 80 °C. The activation energy for cabothemic roasting was also calculated to be -5.06 kJ which implies that carbothemic roasting of EAFD was exothermic and chemical reaction controlled.

Two products, Zinc and iron were therefore recovered from electric arc furnace dust. Zinc concentrate contained 75.25% of Zn, 1.2% of SiO_2 and 0.15% CaO while sponge iron produced contained 57% of Fe, carbon content at 35.27%, 5.20% of SiO_2 and 4.25% of CaO.

4.2 Recommendation

The reaction behaviour of iron oxide during carbothermic heating must be studied to establish the direct effect of lump size of the iron oxide containing material on degree of reduction. The mode of heating the iron oxide containing material should also be studied to establish the best methods of heating and affecting the rate of iron oxide reduction.

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Cut-off grade optimisation encompassing mineral royalty

Webby Banda[1], Bunda Besa[2]

Abstract

Cut-off grade optimisation is an important step in any production planning setting. Being cognizant of this fact, it is imperative that this parameter be optimised. One of the earliest applied method of determining the optimal cut-off grade is the Lane algorithm. In this paper the Lane algorithm has been modified to encompass ad valorem mineral royalty. This is done with the objective of exhuming the relationship between the rate of ad valorem mineral royalty and optimal cut-off grade using the profit function. Results show that encompassing mineral royalty in the Lane algorithm increases the cut-off grade and reduces the total achievable profit of a mining project. Additionally, results show that there is a positive linear relationship between the ad valorem mineral royalty rate and optimal cut-off grade.

Keywords: Cut-off grade, optimisation, Lane algorithm; mineral royalty, Ad valorem

1. Introduction

Cut-off grade is used to attach material destination labels to mined ore. Material above the mill cut-off grade is taken to the concentrator whilst that below the cut-off grade is taken to other material destination points which include waste dumps, leach pads and stockpiles. Cut-off grade is an important techno-economic parameter that has significant influence on the profitability of a mining business. A high cut-off grade sends low quantity of high grade mineral material to the concentrator. To the contrary, a low cut-off grade sends high quantity of low grade mineral material to the milling plant that may lead to high processing costs. Therefore, it is the objective of any cut-off grade policy to maintain an optimal balance between these two extremes. Increasing the cut-off grade beyond normal depletes the potentially economically exploitable reserves of a mine. This has future implication of narrowing the life span of the project. Notwithstanding this, cut-off grade values are usually set high at the starting phase of mining because of the need to expedite the recovery of capital injected. Being cognizant with this fact, it is essential that this parameter is optimised. According to Whittle and Vassiliev (1998), cut-off grade optimisation comes with various benefits:

- It assists in maximizing economic return over the life span of the mine; and
- It can be used to simulate different mine, milling and refining configurations so as to determine the one which yields the maximum economic benefit.

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Several methods of determining the optimal cut-off grade have been presented in literature. One of the earliest applied method is the Lane algorithm (Lane, 1964). This method optimises the cut-off grade by optimising the Net Present Value (NPV) subject to mine, mill and refinery constraints. Maximization of NPV aids in enhancing mining business success. This enhancement reduces financial failure. Most of the later cut-off grade optimisation models that have been developed are simply a refinement of the Lane algorithm. The Lane's model suffers a number of deficiencies. For instance, it cannot be applied in situations where one wants to optimise cut-off grade in light of multiple processing streams, stockpiles or multi-element commodities (Dagdelen and Kawahata, 2007). Furthermore, Lane's method only optimises cut-off grade for a predetermined extraction sequence (Myburgh et al., 2014). Although Lane's algorithm has shown some technical lapses, it still remains a mathematically sound method of optimising cut-off grade. This is due to the fact that it employs the NPV as a pivotal parameter of undertaking optimisation. This fact also justifies why some surface mines still use Lane's algorithm as a method of undertaking an optimal pit design. According to Whittle (1989), the pit outline with the highest NPV value cannot be determined if the block values are not known. The block values cannot be defined if the extraction sequence is not determined and the extraction sequence cannot be determined unless the pit outline is available.

Many researchers have contributed in devising methods and algorithms to resolve the issue of cut-off grade optimisation. Ataei and Osanloo (2003) applied the golden search method to determine the optimum cut-off grades of multiple deposits. This was pursued because the original Lane model deals with single deposits and cannot be applied to multiple deposits due to the infinite number of possible candidates of the optimum cut-off grade. Ataei and Osanloo (2004) later integrated a genetic algorithm to this golden search method. The reason for this pursuit stems from the fact that the former method is inefficient when applied solely. Azimi and Osanloo (2011) applied a combination of the genetic algorithm and non-linear programming to determine the mining cut-off grade strategy. Asad and Dimitrakopoulos (2013) applied a heuristic approach to stochastic cut-off grade optimisation for open pit mining complexes with multiple processing streams. The proposed approach determines the optimal cutoff grade policy based on a stochastic framework that accounts for uncertainty in supply of ore to the multiple ore processing streams. Yasrebi (2015) utilized non-linear programming by applying a computer based model. He et al (2009) in their research used a genetic algorithm and neural network nesting method to simulate the highly non-linear relationship between variables in the mining system so as to optimise the cut-off grade and grade of crude ore.

In the past decade cut-off grade optimisation has been extended to include the environmental aspect. Gholamnejad (2008) extended the Lane algorithm to include rehabilitation cost of waste dumps. Results showed that encompassing rehabilitation cost in the optimisation exercise increases the total achievable NPV of the project. Osanloo et al (2008) improved Lane's algorithm on the basis of maximization NPV by minimizing environmental costs. Narrei and Osanloo (2015) extended the Lane algorithm to include reclamation costs and income from waste dumps, tailings dam and pits. The developed hybrid model was validated by data gathered from Gol-Gohar iron mine in Iran. Results showed some improvement in the NPV of the project. King (1998) dichotomized a deposit into high-low grade portions and demonstrated that the best discounted value of a project was achieved when closure and rehabilitation costs were encapsulated in the design process.

Bascetin (2007) determined the cut-off grade strategy based on lane's algorithm by adding an optimisation factor based on the generalized reduced gradient algorithm through computer programming using excel and visual basic. Asad (2005) extended the lane algorithm to incorporate a stockpiling option of two economic minerals for open pit mining operations. The algorithm developed is applicable to metallic ore deposits because it considers long-term stockpiles. Asad and Topal (2011) applied an NPV maximization model for optimum cut-off grade policy of open pit mining operations. In their study a combined impact of introducing economic paramaters, escalation and stockpiling options into cut-off grade optimisation was assessed. Birch (2016) assessed the impact of the South African mineral resource royalty on cutoff grades for narrow, tabular Witwatersrand gold deposits. Seven individual gold mines were considered. The impact on each of the seven mines was different but an estimated 7.6 billion Rand was expected to be paid in mineral resource royalty over their anticipated remaining lives. The overall revenue decreased by 10 billion Rand because of the costs of the mineral resource royalty and increased cut-off grades. Abdollahisharif et al (2012) modified the Lane algorithm to consider variable processing capacities in order to determine the optimal cut-off grade. The modified Lane algorithm generated an optimal cutoff grade and maximal NPV that was less than that generated by the original model. A computer based program was developed in Microsoft Excel for facilitated use of the proposed model.

Asad (2005b and 2007) improved the Lane algorithm by considering cost escalation and dynamic metal prices. Cairns and Shinkuma (2003) showed that the rate of growth of price net of the interest rate has an influence on the cut-off grade but that the sign of the effect is ambiguous because it depends on a complicated factor involving technical, mine-specific elasticities.

Considering the different cut-off grade optimisation models presented in literature, it is not sufficing to cling to one method of cut-off grade optimisation. This is because no one method is technically superior in all circumstances. The approach should be to embrace all techniques when faced with real practical problems. In view of all these adjustments and models, no modification of the Lane algorithm has encapsulated mineral royalty (ad valorem) tax. This study transforms and refines the Lane algorithm by integrating mineral royalty in the cut-off grade optimisation exercise. This is done to uncover the relationship between the optimal cut-off grade and the rate of ad valorem mineral royalty.

2. Lane algorithm

Lane algorithm is a mathematical procedure used to determine the optimum cut-off grade. The algorithm in its objective considers three main constraints which are mining, concentrating and refining (Figure 1). It uses economic factors (selling price and unit costs) and technical factors (grade distribution and capacities of mining operations) to maximize the NPV and profit value. The NPV is consistently quoted as the principal economic criterion because of its superiority of encapsulating cash flow values over the production period. Secondly, it takes into consideration the time value of money. The profit criterion on the other hand considers profit values at a particular point in time. Additionally, it does not consider the time value of money. Table 1 shows the parameters employed in defining the Lane's model.

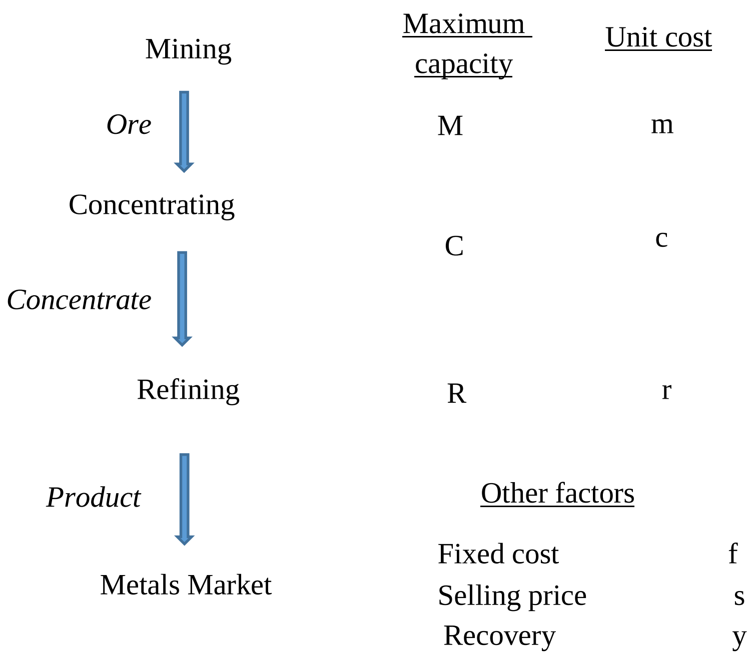

Figure 1: Lanes cut-off grade optimisation model (Lane, 1964)

Table 1: Parameters of Lane algorithm

Description	**Symbol**
Maximum capacities	
Mining capacity (tonnes/year)	M
Concentrator capacity (tonnes/year)	C
Refinery capacity (tonnes/year)	R
Quantities	
Quantity to be mined (tonnes)	Q_m
Quantity of ore sent to the concentrator (tonnes)	Q_c
Quantity of final product (tonnes)	Q_r
Costs	
Mining costs (United States Dollar (USD)/tonne of material	m
excavated)	c
Concentrating costs (USD/tonne of material milled)	r
Refining and marketing costs (USD/tonne of product)	f
Fixed costs per year excluding mineral royalty (USD/year)	
Other parameters	
Production period (years)	T
Profit (USD)	P
Annual discount rate (%)	d
Selling price (USD/tonne of final product)	s
Overall metallurgical recovery (%)	y

2.1 Lane algorithm based on profit

The main objective of the Lane algorithm under the profit objective function is to select a cut-off grade that maximizes profit. This section attempts to derive the Lane algorithm encompaasing mineral using parameters defined in Table 1. According to Lane (1964) the main profit equation is as follows:

$$P = sQ_r - [mQ_m + cQ_c + rQ_r + fT] \quad [1]$$

2.1.1 Economic cut-off grade based on profit

Three economic cut-off grades exist. The formulae of economic cut-off grade considering one constraint as a major element to maximize profit are depicted in Table 2. These are cut-off grades considering mining (g_m), concentrating (g_c), and refining (g_r) capacities as sole limiting constraints.

Table 2: Economic cut-off grades based on profit method

Cutoff Grade	Constraint
$\boldsymbol{g_m = \frac{c}{(s-r)y}}$	Mining
$\boldsymbol{g_c = \frac{(c+\frac{f}{C})}{(s-r)y}}$	Concentrator
$\boldsymbol{g_r = \frac{c}{\left(s-r-\frac{f}{R}\right)y}}$	Refinery

2.1.2 Balancing cut-off grade based on profit

The balancing cut-off grades are determined by capacities and grade distribution. They consider two limiting constraints at a time. Three cases exist, namely:

g_{mc}: cut-off grade when mining and concentrator capacity are limiting constraints;
g_{mr}: cut-off grade when mining and refinery capacity are limiting constraints; and
g_{rc}: cut-off grade when refinery and concentrator capacity are limiting constraints.

Case 1: Mining and concentrator capacity as limiting constraints

If both mining and concentrator capacity are the limiting constraints then:

$$\frac{\mathrm{Q_m}}{M} = \frac{Q_c}{C} \quad [2]$$

Therefore, g_{mc} is the cut-off grade that satisfies Equation 2.

Case 2: Mining and refinery capacity as limiting constraints

If both mining and refinery capacity are the limiting constraints then:

$$\frac{Q_m}{M} = \frac{Q_r}{R} \quad [3]$$

Therefore, g_{mr} is the cut-off grade that satisfies Equation 3.

Case 3: Concentrator and refinery capacity as limiting constraints

If both concentrator and refinery capacity are the limiting constraints then:

$$\frac{Q_c}{C} = \frac{Q_r}{R} \quad [4]$$

Therefore, g_{cr} is the cut-off grade that satisfies Equation 4.

2.1.3 Determination of optimal cut-off grade based on profit

Six possible cut-off grades have been identified. Three (g_m, g_c, g_r) are based on capacities, metal price and costs the other three (g_{mc}, g_{mr}, g_{cr}) are based on grade distribution and capacities. The optimum cut-off grade among these is determined using rules in Equation 5 and Table 3.

As a general rule, the overall optimum cutoff grade (G) is calculated using Equation 5 (Hustrulid et al., 2013).

$$G = Middle\ Value\ (G_{mc},\ G_{mr},\ G_{rc}) \quad [5]$$

Table 3: Determination of balancing cut-off grades (Hustrulid et al., 2013)

Cutoff Grade	**Constraint**
$\boldsymbol{G_{mc}} = \{(\boldsymbol{g_m},\ \boldsymbol{if}\ \boldsymbol{g_{mc}} \leq\ \boldsymbol{g_m})\ (\ \boldsymbol{g_c},\ \boldsymbol{if}\ \boldsymbol{g_{mc}} \geq\ \boldsymbol{g_c})\ (\boldsymbol{g_{mc}}, otherwise)\}$	Mining and concentrator
$\boldsymbol{G_{mr}} = \{(\boldsymbol{g_m},\ \boldsymbol{if}\ \boldsymbol{g_{mr}} \leq\ \boldsymbol{g_m})\ (\ \boldsymbol{g_r},\ \boldsymbol{if}\ \boldsymbol{g_{mr}} \geq\ \boldsymbol{g_r})\ (\boldsymbol{g_{mr}}, otherwise)\}$	Mining and refinery
$\boldsymbol{G_{rc}} = \{(\boldsymbol{g_r},\ \boldsymbol{if}\ \boldsymbol{g_{rc}} \leq\ \boldsymbol{g_r})\ (\ \boldsymbol{g_c},\ \boldsymbol{if}\ \boldsymbol{g_{rc}} \geq\ \boldsymbol{g_c})\ (\boldsymbol{g_{rc}}, otherwise)\}$	Refinery and concentrator

3. Refined Lane algorithm

Mining typically constitutes of three main stages of operation: (i) the mining stage, where mineral ore of various grades is excavated up to some capacity. (ii) the concentrating stage, where the ore is milled, floated and dewatered to a concentrate up to some limited capacity. (iii) the refining stage where the product is smelted and refined at a limited refinery capacity to a beneficiated product that is sold to the metals market. As highlighted in Section 2.0, the two most important economic criteria which can be applied in cut-off grade optimisation are:

(1) Maximum profit; and

(2) Maximum present value.

The maximum present value is the most generally applied method because it gives the economic optimum between the two. Technical and economic parameters involved in the definition of the refined lane's model encompassing mineral royalty are presented in Table 4.

3.1 Refined Lane algorithm based on profit

The main objective under this criterion is to select a cut-off grade that maximizes profit. With respect to the definitions in Table 4, the profit equation encompassing mineral royalty can be defined as:

$$P = sQ_r - [mQ_m + cQ_c + rQ_r + fT + R_p] \quad [6]$$

Combining like terms yields:

$$P = (s - r)Q_r - mQ_m - cQ_c - fT - R_p \quad [7]$$

R_p although a fixed cost is not treated as part of f because its calculation is dependent on the average grade of the concentrate material. This average grade is a function of the cut-off grade of the mined material. This means the average grade of the concentrate material is highly dependent on the cut-off grade of the mined material. Six possible cut-off grades exist. Three (g_m, g_c, g_r) are based on capacities, metal price, costs and mineral royalty tax (economic cut-off grade) the other three (g_{mc}, g_{mr}, g_{cr}) are based on grade distribution and capacities (balancing cut-off grade).

3.1.1 Derivation of economic cutoff grade based on profit

Economic cut-off grade may be limited by mining, concentrator and refinery capacity. Three cases arise.

g_m: cut-off grade when mining capacity is the limiting constraint;

g_c: cut-off grade when concentrator capacity is the limiting constraint; and

g_r: cut-off grade when refinery capacity is the limiting constraint.

Case 1: Mining capacity as limiting constraint

If mining capacity (M) is the limiting constraint, then the time (T) needed to mine the material Q_m is given by:

$$T = \frac{Q_m}{M} \quad [8]$$

Substituting Equation 8 into Equation 7 gives:

$$P = (s - r)Q_r - mQ_m - cQ_c - f\frac{Q_m}{M} - R_p \qquad [9]$$

Combining like terms yields:

$$P = (s - r)Q_r - Q_m(m + \frac{f}{M}) - cQ_c - R_p \qquad [10]$$

R_p can be defined as:

$$R_p = s.y.g.R_r.Q_c \qquad [11]$$

Where, g is the average grade of material sent to the concentrator. Substituting Equation 11 into Equation 10 gives:

$$P = (s - r)Q_r - Q_m\left(m + \frac{f}{M}\right) - cQ_c - s.y.g.R_r.Q_c \qquad [12]$$

To find the grade which maximizes profit under this constraint, Equation 12 must be differentiated with respect to g:

$$\frac{dP}{dg} = (s - r)\frac{dQ_r}{dg} - \left(m + \frac{f}{M}\right)\frac{dQ_m}{dg} - c\frac{dQ_c}{dg} - s.y.R_r.Q_c \qquad [13]$$

Q_m is independent of g. This is because Q_m is determined by the mining cut-off grade and not g. Thus:

$$\frac{dQ_m}{dg} = 0 \qquad [14]$$

Additionally, the lowest acceptable value of g is that which satisfies:

$$\frac{dP}{dg} = 0 \qquad [15]$$

Thus, Equation 13 becomes:

$$s.y.R_r.Q_c = (s - r)\frac{dQ_r}{dg} - c\frac{dQ_c}{dg} \qquad [16]$$

Multiplying both sides of Equation 16 by dg and applying integration gives:

$$\int(s.y.R_r.Q_c)\,dg = \int(s - r)dQ_r - \int c\,dQ_c \qquad [17]$$

Taking $(s.y.R_r.Q_c)$, $(s - r)$ and c as constants gives:

$$(s.y.R_r.Q_c)\int 1\,dg = (s - r)\int 1\,dQ_r - c\int 1\,dQ_c \qquad [18]$$

Integration in Equation 18 gives:

$$s.y.R_r.Q_c.g = (s - r)Q_r - cQ_c \qquad [19]$$

Q_r is related to Q_c by the following equation:

$$Q_r = Q_c.y.g \quad [20]$$

Substituting Equation 22 into Equation 21, gives:

$$s.y.R_r.Q_c.g = (s - r)(Q_c.y.g.) - cQ_c \quad [21]$$

Cancelling out like terms and rearranging Equation 21 gives the cut-off grade (g_m) based on the mining capacity as the limiting constraint.

$$g_m = \frac{c}{[(s-r)y - s.y.R_r]} \quad [22]$$

Case 2: **Concentrator capacity as limiting constraint**

If concentrator capacity (C) is the governing constraint, then the time (T) to mine and process a Q_c block of material is:

$$T = \frac{Q_c}{C} \quad [23]$$

Substituting Equation 23 into Equation 6 gives:

$$P = (s - r)Q_r - mQ_m - (c + \frac{f}{C})Q_c - R_p \quad [24]$$

Following the same steps as in case 1, the cut-off grade (g_c) when the concentrator capacity is the limiting constraint is:

$$g_c = \frac{(c + \frac{f}{C})}{[(s-r)y - s.y.R_r]} \quad [25]$$

Case 3: **Refinery capacity as limiting constraint**

If refining capacity (R) is a governing constraint, then the time (T) needed to refine Q_r is given by:

$$T = \frac{Q_r}{R} \quad [26]$$

Substituting Equation 26 into Equation 6 gives:

$$P = \left(s - r - \frac{f}{R}\right)Q_r - mQ_m - cQ_c - R_p \quad [27]$$

Following the same procedure as in case 1, the cut-off grade (g_r) when the refinery is the limiting constraint is given by:

$$g_r = \frac{c}{[\left(s - r - \frac{f}{R}\right)y - s.y.R_r]} \quad [28]$$

Table 4: Technical and economic parameters of mode

Description	Symbol
Maximum capacities	
Mining capacity(tonnes/year)	M
Concentrator capacity (tonnes/year)	C
Refinery capacity per year (tonnes/year)	R
Quantities	
Quantity mined (tonnes)	Q_m
Quantity of ore sent to the concentrator (tonnes)	Q_c
Quantity of final product (tonnes)	Q_r
Costs	
Mining costs (USD/tonne of material excavated)	m
Concentrating costs (USD/tonne of material milled)	c
Refining and marketing costs (USD/tonne of product)	r
Fixed costs per year (USD/year)	f
Other parameters	
Production period as governed by mining, milling and concentrating (years)	T
Profit (USD)	P
Annual discount rate (%)	D
Selling price (USD/tonne of final product)	s
Overall metallurgical recovery (%)	y
Mineral royalty payable (USD)	R_p
Mineral royalty rate (%)	R_r

3.1.2 Derivation of the balancing cut-off grades based on profit

A second set of cut-off grades (balancing grades) exists. To calculate this, one needs to know the grade distribution of the mineral material. These include:

g_{mc}: when mining and concentrator capacity are limiting constraints;

g_{mr}: when mining and refinery capacity are limiting constraints; and

g_{rc}: when refinery and concentrator capacity are limiting constraints.

The above cut-off grades (g_{mc} g_{mr} g_{rc}) are determined using equations 2, 3 and 4. Mineral royalty has no influence on the balancing cut-off grade in all three cases because it does not dictate the material balance between limiting constraints. The capacities and grade distribution are what determine the balancing cut-off grades.

3.1.3 Determination of optimal cut-off grade based on profit

Six possible cut-off grades have been identified. Three (g_{mc}, g_{mr}, g_{cr}) are based on grade distribution and capacities the other three (g_m, g_c, g_r) are based on capacities, metal price, costs and mineral royalty. The optimum cutoff grade is determined using rules under the Lane algorithm (Table 3 and Equation 5).

4. Application of algorithms to a numerical example

Numerical hypothetical data (Tables 5, 6 and 7) is examined and the optimal cut-off grade is consequently calculated for better understanding of the refined Lane's algorithm encompassing mineral royalty. The results obtained are compared to the results of the original Lane model (Section 2.0). For simplicity, the numerical example is calculated based on the profit criterion.

Table 5: Hypothetical mineral inventory data

Grade Interval (lbs/tonne)	Quantity (tonne)
0.0 – 0.1	150
0.1 – 0.2	150
0.2 – 0.3	150
0.3 – 0.4	150
0.4 – 0.5	150
0.5 – 0.6	150
0.6 – 0.7	150
0.7 – 0.8	150
0.8 – 0.9	150
0.9 – 1.0	150
	Σ 1500

Table 6: Mineral royalty, costs, price and capacities

Parameter	Symbol	Quantity	Unit
Maximum capacities			
Mining capacity	M	100	tonnes/year
Concentrator capacity	C	50	tonnes/year
Refinery capacity per year	R	40	tonnes/year
Costs			
Mining costs	m	1	USD/tonne
Concentrating costs	c	2	USD/tonne
Refining and marketing costs	r	5	USD/lb
Rehabilitation cost	h	1.5	USD/tonne
Fixed costs per year	f	300	USD/year
Other parameters			
Selling price	s	25	USD/lb
Overall metallurgical recovery	y	100	%
Mineral royalty rate	R_r	9	%

Table 7: Average concentrator grade (g), Q_m , Q_c and Q_r as a function of cutoff grade

Cut-off (lbs/tonne)	**Tonnage**			
	Q_m (tonne)	**Q_c (tonne)**	**Average Grade (lb/tonne)**	**Q_r (lbs)**
0.0	1500	1500	0.50	750
0.1	1500	1350	0.55	742.2
0.2	1500	1200	0.60	720
0.3	1500	1050	0.65	682.5
0.4	1500	900	0.70	630
0.5	1500	750	0.75	562.5
0.6	1500	600	0.80	480
0.7	1500	450	0.85	382.5
0.8	1500	300	0.90	270
0.9	1500	150	0.95	142.5

4.1 Solution by Lane algorithm

This section provides a solution to the hypothetical numerical example using the Lane cut-off grade optimisation model. The formulae in Table 2 have been used to determine the economic cut-off grades. The balancing cut-off grades have been determined using Table 10 and Equations 2, 3 and 4. Table 8 shows the values of the economic (g_m, g_c, g_r) and balancing (g_{mc}, g_{mr}, g_{cr}) cut-off grades. Figure 2 shows a graphical representation of profits P_m, P_c, P_r as a function of cut-off grade when mining, concentrating and refining are limiting constraints, respectively.

Table 8: Economic and balancing cut-off grades (Lane algorithm)

Cut-off Grade	**Value (lbs/tonne)**
g_m	0.10
g_c	0.40
g_r	0.16
g_{mc}	0.5
g_{mr}	0.45
g_{cr}	0.6

Using values in Table 8, rules in Table 3 and Equation 5 gives the optimal cut-off grade as:

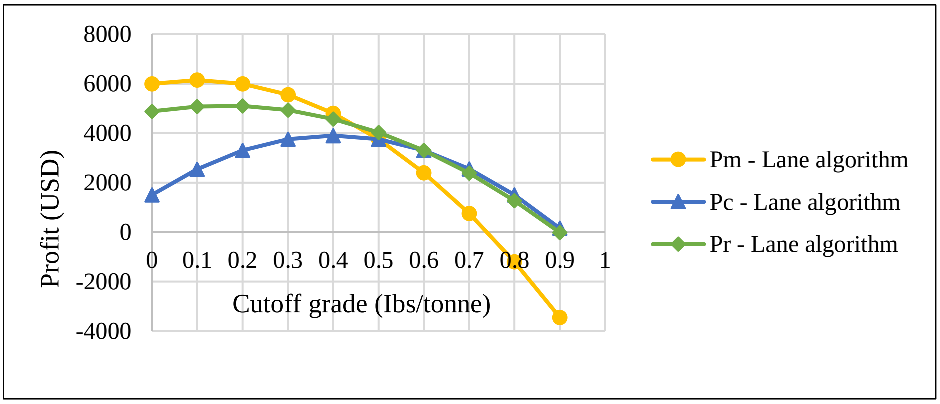

Figure 2: Profit as a function of cut-off grade (Lane algorithm)

$$G = 0.40 \text{ lbs/tonne}$$

From Table 10, the average grade of ore sent to the concentrator (g) at the cut-off grade of 0.40 lbs/tonne is 0.70 lbs/tonne. For 100% recovery the quantities are (Table 9):

Table 9: Quantity (Q_m, Q_c, Q_r, Q_h) (Lane algorithm)

Quantities	**Value (tonnes)**
Q_m	1500
Q_c	900
Q_r	630
$Q_m - Q_c$	600

Applying the respective capacities to the above quantities (Table 12) one finds the following production periods (Table 10) when either mining, concentrator or refinery is the limiting constraint:

Table 10: Production period when either mining, concentrator or refinery is the limiting constraint (Lane Algorithm)

Production period	**Value (years)**
Production period when mining is the limiting constraint (T_m)	15
Production period when concentrating is the limiting constraint (T_c)	18
Production period when refining is the limiting constraint (T_r)	15.8

Since the concentrator requires the longest time (18 years), it controls the production capacity. Therefore, total profit, annual profit and NPV are calculated based on the time period of the concentrator (Table 11).

Table 11: Profit, annual profit and NPV (Lane algorithm)

Parameter	**Value (USD)**
Profit	3900
Annual profit	325
NPV based on annual profit	1992

4.2 Solution by refined Lane algorithm

This section provides a solution to the hypothetical numerical example using the refined Lane cutoff grade optimisation model encompassing mineral royalty. The formulae in Table 5 have been used to determine the economic cut-off grades. The balancing cut-off grades have been determined using values in Table 10 and Equations 2, 3 and 4. Table 12 shows the values of the economic (g_m, g_c, g_r) and balancing (g_{mc}, g_{mr}, g_{cr}) cutoff grades. Figure 3 shows a graphical representation of profits P_m, P_c, P_r as a function of cut-off grade when mining, concentrating and refining are limiting constraints, respectively.

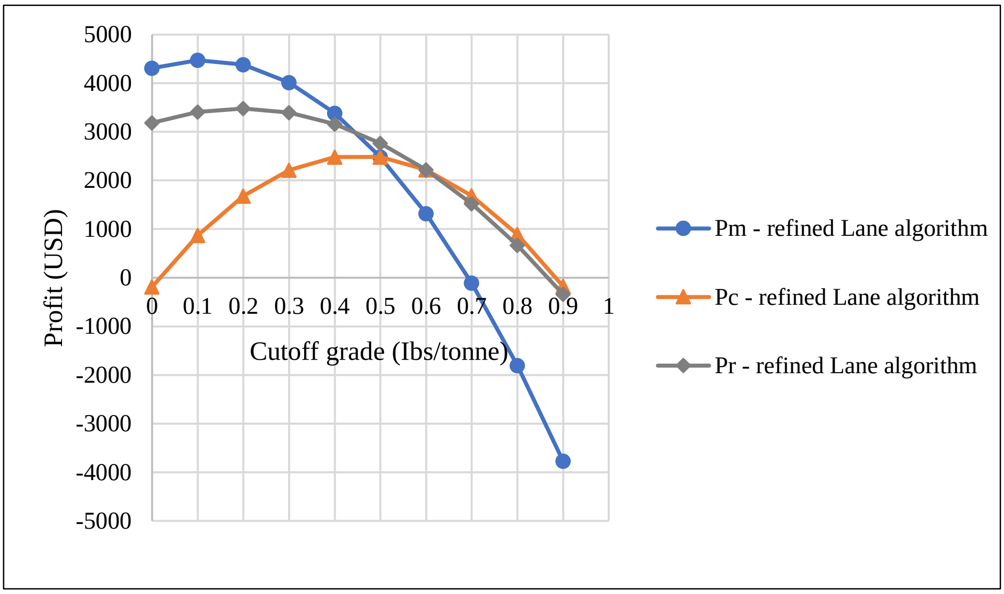

Figure 3: Profit as a function of cut-off grade (refined Lane algorithm)

Table 12: Economic and balancing cut-off grades (refined Lane algorithm)

Cut-off Grade	**Value (lbs/tonne)**
g_m	0.11
g_c	0.45
g_r	0.19
g_{mc}	0.5
g_{mr}	0.45
g_{cr}	0.6

Using values in Table 12, rules in Table 3 and Equation 5 gives the optimal cut-off grade as:

$$G = 0.45 \text{ lbs/tonne}$$

From Table 10, the average grade of ore sent to the concentrator (g) at the cu-toff grade of 0.45 lbs/tonne has been obtained by employing the mathematical technique of interpolation. Thus, the average grade (g) of material sent to the concentrator is 0.73 lbs/tonne. Just like g the quantities were obtained by applying the mathematical technique of interpolation. For 100% recovery the quantities are (Table 13):

Table 13: Quantities (Q_m, Q_c, Q_r, Q_h) (refined Lane algorithm)

Quantities	**Value (tonnes)**
Q_m	1500
Q_c	825
Q_r	596
$Q_m - Q_c$	675

Applying the respective capacities to the above quantities (Table 13) one finds the following production periods (Table 14) when either mining, concentrator or refinery is the limiting constraint:

Table 14: Production period when either mining, concentrator or refinery is the limiting constraint (refined Lane algorithm)

Production period	**Value (years)**
$\boldsymbol{T_m}$	15
$\boldsymbol{T_c}$	16.5
$\boldsymbol{T_r}$	14.9

Since the concentrator requires the longest time (16.5 years), it controls the production capacity. Therefore, total profit and annual profit are equal to USD 2,329 and 141, respectively.

4.3 Discussion of results

As shown in Table 15, the Lane algorithm encapsulating mineral royalty (refined Lane algorithm) gives a higher optimum cut-off grade when compared to the original Lane model. This results in the refined Lane model having a higher average grade of material sent to the concentrator than the original model. This is the case because average grade of material sent to the concentrator is a function of the cut-off grade. This means when cut-off grade is higher, the average grade of material sent to the concentrator increases and vice versa. The refined Lane model because of its higher cut-off grade gives a reduced amount of mineral material sent to the concentrator than the original Lane model. This translates into reduced profit and NPV. This also translates into increased total amount of material ($Q_m - Q_c$) sent to other destinations (waste dump, leach pads and stockpiles). Figure 4 shows the relationship between ad valorem mineral royalty rate and optimum cut-off grade.

Table 15: Comparison of results of original and refined Lane algorithm

Parameter	**Lane algorithm**	**Refined Lane algorithm**	**Units**
$\boldsymbol{G}$	0.40	0.45	lbs/tonne
$\boldsymbol{g}$	0.70	0.73	lbs/tonne
$\boldsymbol{Q_m}$	1500	1500	tonnes
$\boldsymbol{Q_c}$	900	825	tonnes
$\boldsymbol{Q_r}$	630	596	tonnes
$\boldsymbol{Q_m - Q_c}$	600	675	tonnes
$\boldsymbol{Profit}$	3900	2329	USD

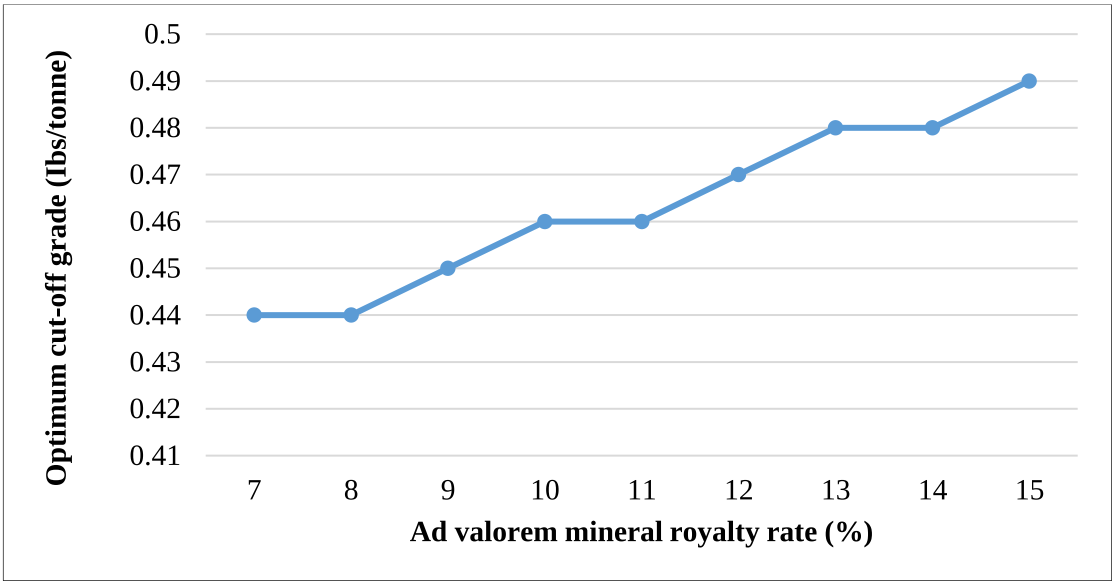

Figure 4: Relationship between optimal cut-off grade and mineral royalty rate

From Figure 4, it can be deduced that there is direct positive relationship between optimal cut-off grade and mineral royalty rate. |However, it must be mentioned that a higher than normal cut-off grade can lead to the use of high grading mining technique. This mining technique can lead to the reduction of economically exploitable reserves thus reducing the life of mine. A reduction in mine life can lead to future high unemployment levels. Secondly, high grading can lead mining engineers not to follow the echelon of mining subsequently leading to mining technical problems. In light of the above implications, it is important for governments to establish an ad valorem mineral royalty rate that generates a cut-off grade that is not excessively high.

5. Conclusion

Cut-off grade is an important parameter in any mining business setting. Thus, it is imperative that this parameter be optimised. However, achieving this is no easy task because of the complex nature of the problem. Lane algorithm is one of the earliest mathematical model to be ever applied in cut-off grade optimisation. In its original form it does not incorporate mineral royalty. Therefore, this paper presented a modified Lane's cut-off grade optimisation model that encompassed mineral royalty. Results based on the profit criterion and hypothetical data show that encapsulating the payable amount of mineral royalty increases the optimal cut-off grade and hence the average grade of material sent to the concentrator. This results in reduced amount of material sent to the concentrator and increased total amount of material sent to other destinations (waste dumps, leach pads and stockpiles). This leads to a reduced value of profit. The hypothetical mine results also show that there is a positive linear relationship between mineral royalty rate and optimal cut-off grade.

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Selection of a Suitable and Cost Effective Method for Dewatering Deep Orebodies(below 1040mL) at Konkola Mine-Zambia

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Abstract

Konkola Mine is one of the wettest mines in the world. It pumps about 350,000m^3 of water per day. From the inception of mining operations in 1956, water has always posed a challenge to mining despite the use of dewatering strategy involving dewatering crosscuts and dewatering boreholes.

The groundwater at the mine was mathematically modeled using MODFLOW software in 1989 and predictions of dewatering operations were made for periods of 1989 to 2020. However, in the last two decades, the implementation of the dewatering plan has lagged behind as a result of financial constraints faced by the Mine. The Footwall Aquifer at shaft No. 3 is behind by almost 3 years and the Hangingwall Aquifer at shaft No. 1 is behind by at least 13years. This has resulted in most of the reserves being underwater which poses a safety and mining challenge. The reserves below 1040mL cannot be mined until the Hangingwall Aquifer is dewatered below 1150mL Cave line. The dewatered reserves have almost 2 years to depletion hence this study was undertaken to determine a suitable dewatering method for reserves below 1040mL. The study also sought to establish whether the use of backfilling mining methods would reduce dewatering requirement for the mine.

Drawdown simulations were done using the MODFLOW-VKD software. The water table was generated using MicroStation and Geovia Surpac software. The seepage points identified are the Kafue River, Kakosa Stream, discharge canals, Lubengele Dam, and Lubengele Stream. The study has established that more than 194,170m^3/d of water could be excluded from seeping through the mine. An analysis of mines that use backfilling methods established that the use of backfill reduces hydraulic inflow paths into a Mine and also reduces the dewatering requirement.

The results of the study indicate that the existing conventional dewatering approach (using crosscut and dewatering boreholes) coupled with surface water exclusion methods are the most viable for the reserves below 1040mL as opposed to deep surface wells. It was also found that dewatering requirement could be reduced in the mine by the application of backfill. The approximate cost of Conventional dewatering method (19 crosscuts and 190 boreholes) implementation was less than the cost for the Surface deep holes (seven million seven hundred fifty one thousand four hundred eleven dollars).

Key words: Mine operations, dewatering, Hangingwall Aquifer, backfilling

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1.0Introduction

Konkola Mine (Figure 1) is located in the town of Chililabombwe and which lies 25 Km north of Chingola and 12 Km from Kasumbalesa border between Zambia and DR Congo.

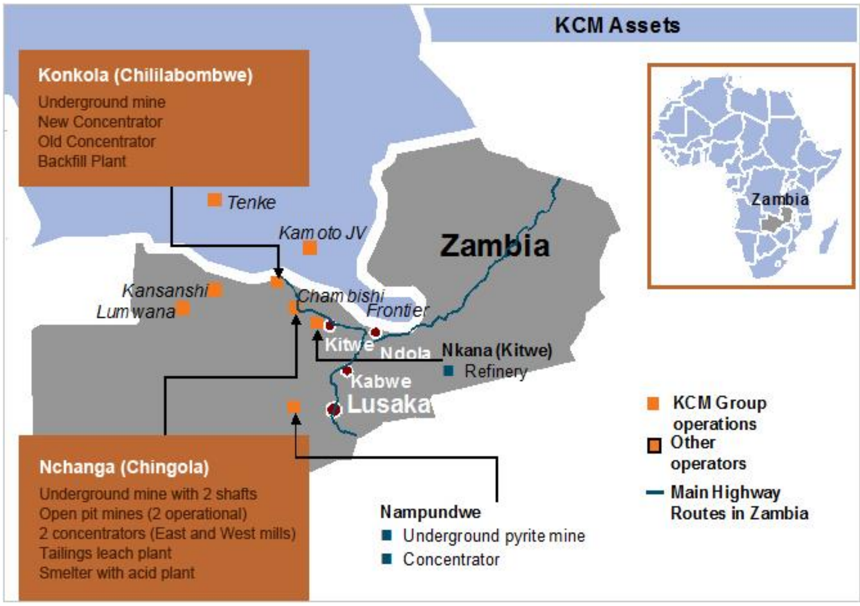

Figure 1: Location of Konkola Mine in Zambia

Production at Konkola Mine (then called Bancroft Mine) started at No 1 Shaft in January 1957. No. 3 Shaft started operating in 1963. While No. 4 Shaft commenced operation in 2010. Konkola Mine is one of the wettest underground mines in the world; it pumps about 350,000m^3/day of water to the surface.

There are three major aquifers in Bancroft Mining Area, where Konkola Mine is located. These are the Lower Porous Conglomerate (LPC) Aquifer- Footwall Quartzite (FWQ) Aquifer, Footwall Aquifer (FWA) and the Hangingwall Aquifer (HWA). The Footwall (20m-40m thick) and the Hangingwall aquifers (about 700m thick) all carry large quantities of water and are primary targets in dewatering. The HWA is immediately above the Hangingwall Quartzite which lies immediately above the Orebody. The Hangingwall Quartzite which is an aquitard lies too close to the ore shale, hence it is always a primary target for dewatering. The Ore Shale unit acts as a watershed and is underlain by the Footwall Aquifer (FWA). The FWA is composed of Porous Conglomerate, Footwall Sandstone and Footwall Conglomerate. About 250m away from the Ore Shale lies the Lower Porous Conglomerate/Footwall Quartzite Aquifer (LPCA/FWQA) which is composed of quartzite and conglomerate rock types. The HWA and FWA contribute about 80% to 90% of water inflows to the mine and only these two are dewatered for the sake of safe mining. The 10% to 20% of water is from the LPCA/FWQA which is not intentionally dewatered except in rare cases where the pore pressure is high. The Figure 1-2 shows the main aquifers in Konkola.

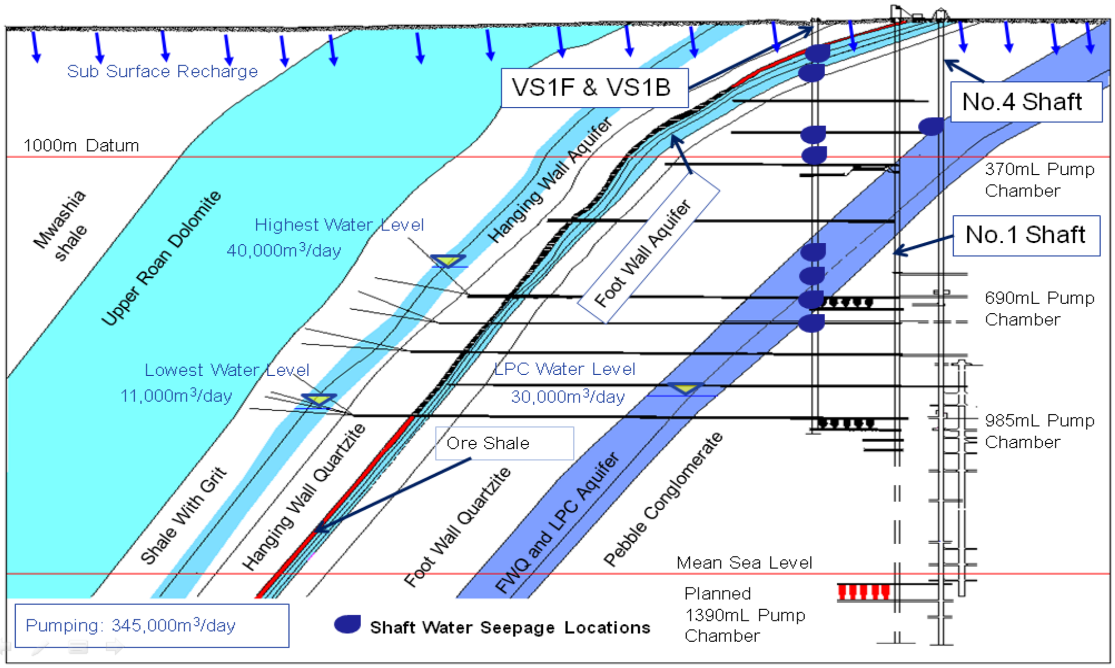

Figure 1.2: Konkola aquifers in Konkola Mine area, Chililabombwe, Copperbelt Province, Zambia

Since the mine started production in 1957, water has always posed a challenge to mining as a result the mine has always had a robust dewatering strategy to dewater the footwall and hanging wall aquifer. However, in the last two decades, the implementation of the dewatering plan has lagged behind by almost 15 years due to the financial constraints faced by the mine. This has led the mine to lag behind in dewatering development. As a result most of the reserves are under the water table and poses a safety and mining challenge.

Konkola Mine is currently mining the upper Orebody which is nearing depletion with about 2 years remaining. This being the case, the mine has invested over US$400,000,000 into the Konkola Deep Mining Project (KDMP) to expose the deep Orebody and increase the production of copper ore from 2 to 6mt/y at 3.2% TCu. The mine has about 49mt reserves. However, most of the reserves are below the water table. Extraction of reserves below the water table possess a great danger to mining operations in terms of safety and flooding. Only Konkola flats and Konkola extension are mining above the water table as shown in Figure 1.3.

In terms of the Footwall Aquifer at No.3 Shaft, it is behind by 3 years while the Hanging Wall Aquifer at No. 1 Shaft is behind by at least 13 years.

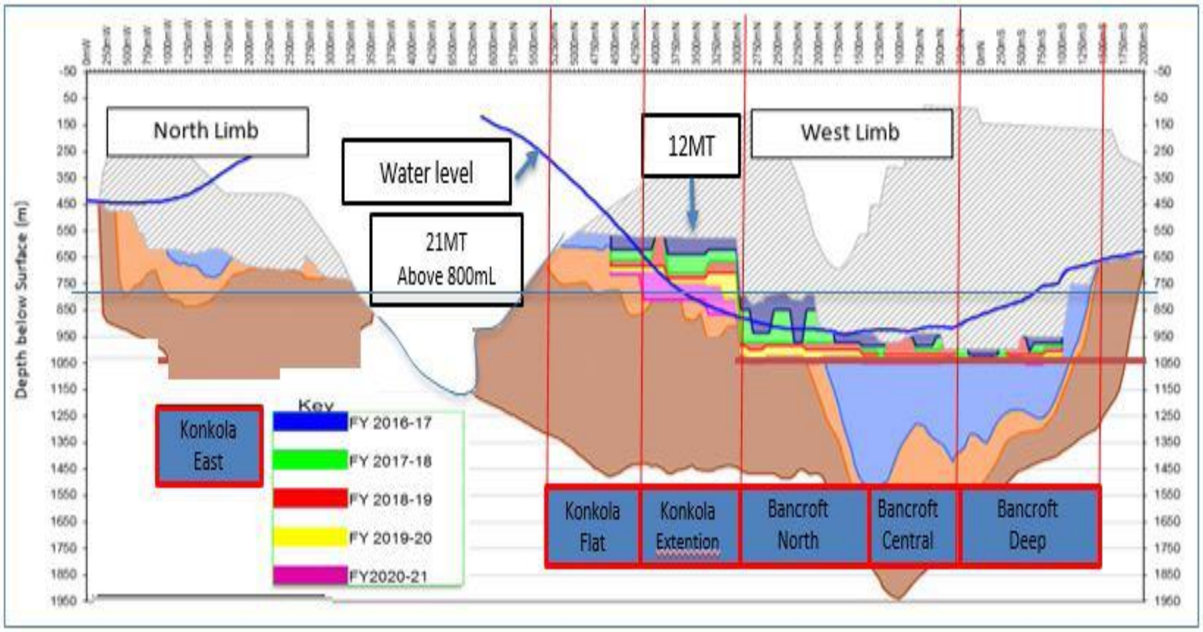

Figure 1.3: Konkola Reserves

Therefore, this study was undertaken in order to establish an accelerated dewatering plan for Footwall and Hangingwall aquifers for extra deep ore-body and to allow efficient mining operations to continue. The study compared different dewatering approaches and selected a workable approach for Konkola Mine based on cost and viability.

2.0 Methodology

Various methodologies were used to achieve the aim of the study. Desktop studies of the Konkola, Copperbelt and global dewatering approaches were reviewed. Various sites were visited for data collection and include: dewatering crosscuts, drain drives, settlers, sumps, pump chambers, discharge canals, Kakosa stream, Lubengele Stream, Lubengele Dam and the Kafue River.

Dewatering simulations were done using MODFLOW-VKD software and costs for both alternatives were compared and reviewed.

3.0 Data Collection and Analysis

3.1 Sources of Water to Konkola Mine

The study established three sources of water to the mine: rainfall infiltration, seepages from water bodies on the surface and water stored in aquifers (Mulenga, 1991).

3.1.1 Rainfall Pattern for Konkola Mine

The average rainfall for Konkola Catchment area is about 1283.5mm. The wettest months are December and January. The average rain water infiltration was found to be 20% (WMC, 2000), while 70% of the rain water ended up in run off and 10% evaporated. Table 3.1 shows the summary of rainfall pattern at Konkola Mine Catchment from 1953 to 2016/17 rain seasons.

Average infiltration from rainfall= 20%* Average Rainfall

= 20% *1274.3mm

=254.9mm/year

The average recharge to the mine is 254.9mm for year on the groundwater catchment area (estimated to be 240km^2)

The infiltration/d= (average infiltration*Area of catchment)/ 365days

= (0.2549m/year*240000000m^2)/365d/year

=167,605.5m^3/d

The presence of dambos in the Konkola Catchment Area also contributes to the high infiltration of rainwater into the mine. The dambos located at 2900mw near Lubengele Dam have made the area between 2200fL to 2900fL un-minable and the drives mined in this area were collapsing due to poor ground condition caused as a result of high presence of water (the entire area is sterilized). This area was not mined till present day. The mine has a lot of dambos which get filled up with water during rainy season contributing to high infiltration of rainfall into the mine. Figure 3.1 shows a picture of a dambo and Figure 3.2 shows the location of dambos in Bancroft Mining area.

Figure 3.1: Picture of a Dambo or Sinkhole at Konkola Mine

Table 3. 1: Rainfall Pattern Summary for Konkola Mine Catchment from 1953 to 2016/17 rainy seasons, Chililabombwe, Copperbelt Province, Zambia

Month	**September**	**October**	**November**	**December**	**January**	**February**	**March**	**April**	**May**	**June**	**July**	**August**	**Total Annual (mm)**
Average Monthly Rainfall (mm)	3.9	27.9	141.7	277.8	291.9	255.4	215.3	52.9	7.4	0.0	0.0	0.0	1274.3
Average Evaporation (mm/month)	195.2	215.2	167.2	130.4	122.8	119.4	131.6	134.2	136.0	118.6	134.4	157.8	

3.1.2 Water Contained in aquifers

The aquifers release water into the mine as a result of decompression of the rocks and dewatering. This is caused by lithostatic unloading as a result of under burden remover due to the extent of mining and new development. This leads to a decrease in total stress leading to a dilation and expansion of the overburden rocks thus leads to an increase in the permeability (K). Water also flows from other distant catchment areas into the mine catchment area as a result of hydrogeological connectivity and fractures, the Lubambe Water Table has been lowered to about 400mbgl or 973mamsl due to the pumping at Konkola Mine. Fractures play a major role in water inflow into the mine. The fractures act as water conduits or drainages, once they intercept a water source. The fault zones are areas of high hydraulic conductivity, Konkola Mining area is sandwiched by three major faults (Figure 3-2). The faults are connected to aquifers of the mine. The storage removal is estimated to be between 40,000-50,000m^3/d, this is controlled by the extent of mining and new developments but this however can increase to about 200,000 m^3/d depending on the extent of development.

The Hangingwall Aquifer and Footwall Aquifer contribute 80 to 90% of the water dewatered while the 10-20% comes from the Lower Porous Conglomerate and Footwall Quartzite Aquifer.

3.1.3 Concentrated recharge from surface water bodies

Konkola Mine has several recharge zones because of its unique surface hydrogeology and the effects of subsidence on the surface water bodies. The mining area is sandwiched between three major fault zones (Lubengele, Luansobe and Axial Fault zones) which are recharge zones in areas where they intersect a water source as these are connected to the mine aquifers.

The study identified the following possible seepage points into the mine catchment area, these are: Kakosa Stream, Kafue River, unlined canals, canal with damaged lining, Lubengele Stream and Lubengele Dam. Figure 3.2 shows surface seepage points in the Konkola Mine Catchment area.

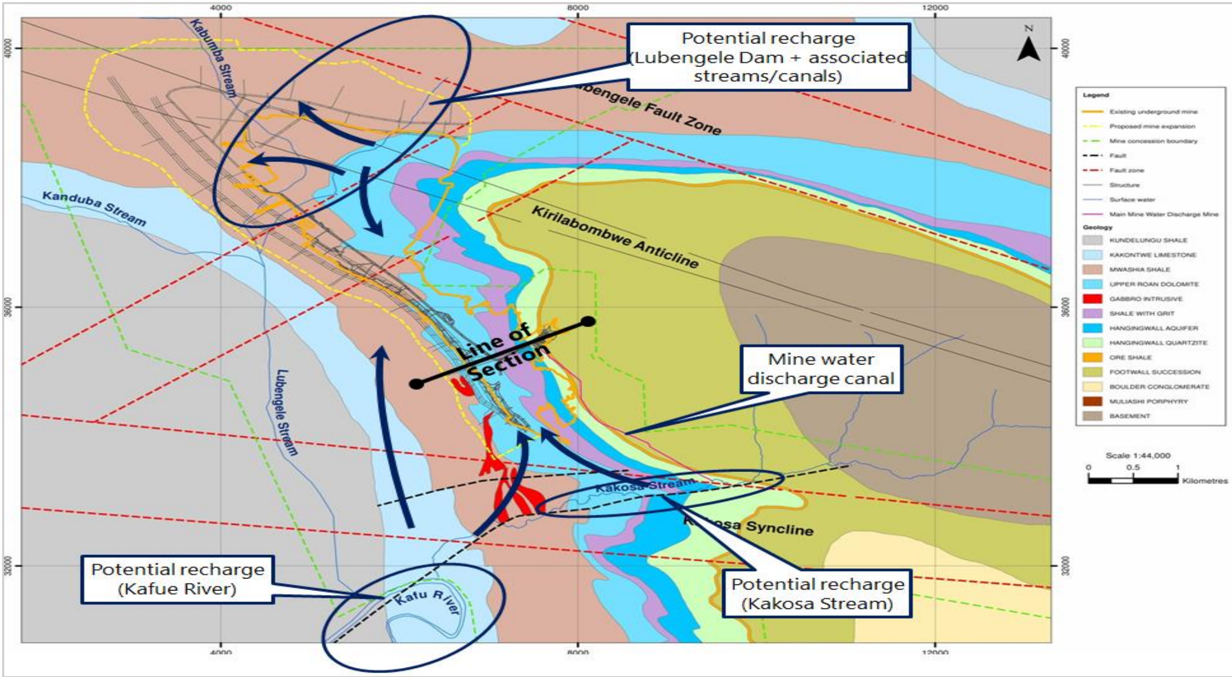

Figure 3.2: Seepage Points in the Konkola Catchment area, Chililabombwe area, Copperbelt Province, Zambia (WSP Parsons Brinckerhoff, 2017).

The recently developed groundwater model by Slumberger Water services indicates seepage of about 27,000m^3/day (possibly much more) from Kakosa Stream (reference). Figure 3.3 shows concentrated seepage at the Kakosa Stream.

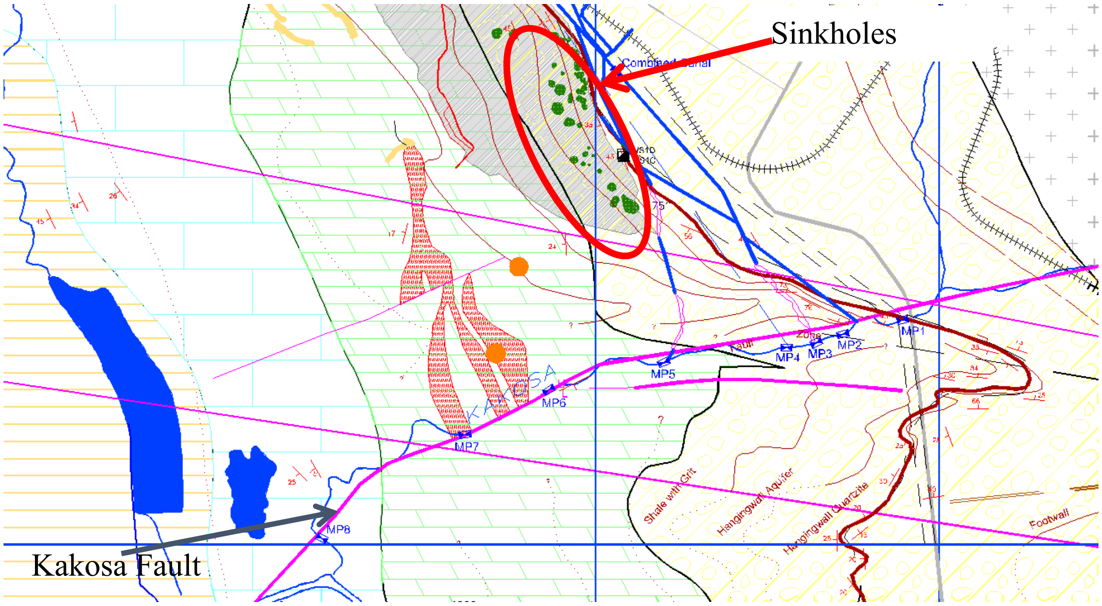

Figure 3.3: Possible Seepage at Kakosa Stream, Chililabombwe area, Copperbelt Province, Zambia

The seepage on the Kafue River was quantified into two seepage limbs (eastern and western). The Kafue River Western Limb intersects the Kakosa Fault which is connected to the mine aquifers and acts as a recharge conduit, about 58,840m^3/d of water seeps on this limb while about 93,230m^3/d seeps through the eastern limb which intersects the Kakontwe Limestone (Karstic). A river discharge measurement approach was used to determine seepage. Teledyne Acoustic Doppler Profiler was used to measure the discharge between 16 sites on the western and eastern limbs. Measurements were made in a run or where the stream flow was constant and straight. The gains or losses in discharged between sites gave the estimate net volume of water exchanged between the river and the groundwater (Banda, Phiri and Nyambe, 2017). Figure 3.4 shows the western and eastern limbs seepage points (References should be given for all Figures that you have NOT done e.g. maps).

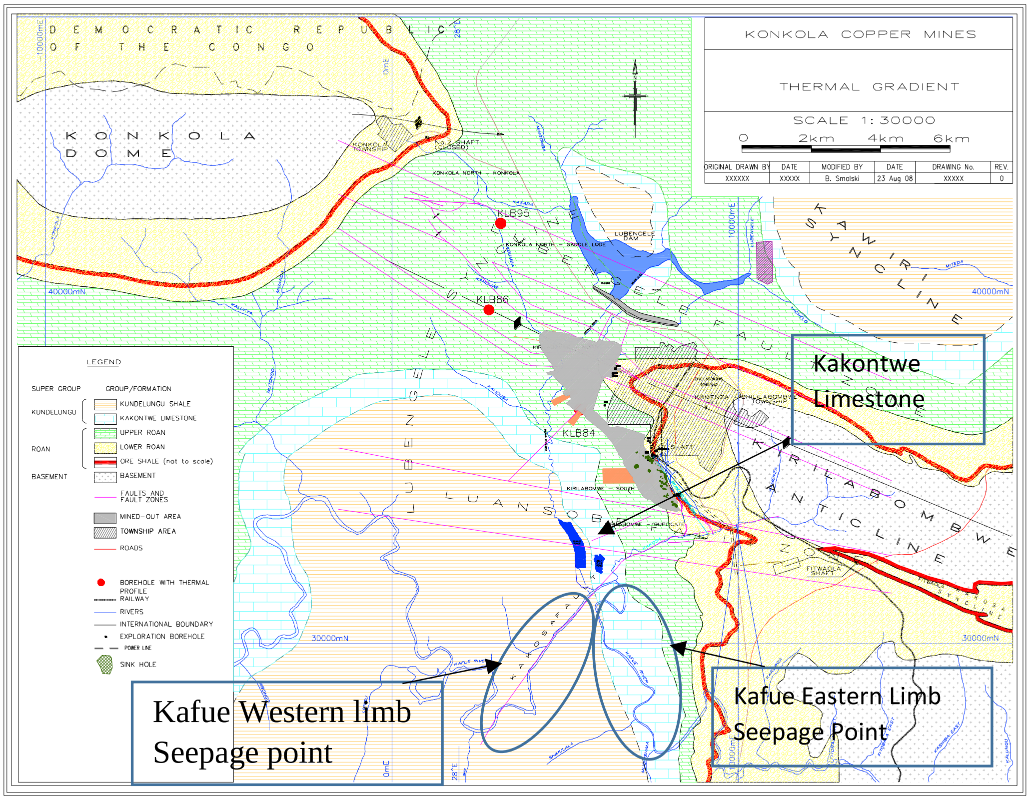

Figure 3.4: A map showing seepage points on the Kafue River, Chililabombwe area, Copperbelt Province, Zambia

Besides these, the other sources of surface water seepages to the aquifers are the cracked and unlined mine discharge canals. The canals are suspected to be leaking water into the mine as they pass through the caving areas on portions which are cracked and not lined. The Figure 3.5 shows cracked canal while Figure 3.6 shows extent catchment area of damaged canals due to settlement.

F igure 3.5: Water Discharge in Cracked canal

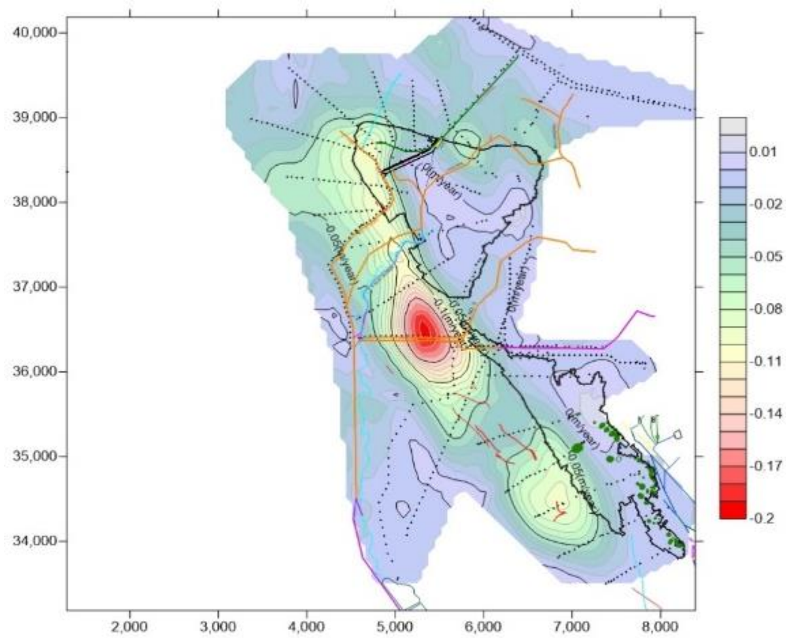

Figure 3.6: Damaged discharge Canals due to Settlements in the Konkola Catchment area, Chililabombwe area, Copperbelt Province, Zambia

Water infiltrates into the aquifers and the mines on the cracked portions of the discharge canal. The mine has got 22km of lined canals and 11km of unlined canals. The unlined canals have high hydraulic conductivity with the aquifers especially when passing through caving areas. Subsidence has greatly affected the canals and most of them crack as a result of tension cracks as shown in Figure 3.6. Ground settlement of as high as 0.2m/year and as low as 0.04m affects lined and unlined discharge canals. The high ground settlement areas have high hydraulic conductivity.

3.3 Methods of Dewatering

The Bancroft Mining area dewatered reserves has about two years life to depletion. Mining below 1040mL cannot take place before dewatering the aquifers that fall within the 1050mL cave line, this has led to the mine seeking an accelerated dewatering method that will lower the Hangingwall Aquifer water below the 1150mL cave line faster in order to avoid an ore gap. Three method have been identified, these are: Surface Exclusion (also known as Surface water control), Surface deep wells and Conventional underground dewatering methods also known as Breakthrough method with underground boreholes.

3.3.1 Surface Exclusion method

Surface exclusion methods as the name suggests involves the exclusion of surface water from the catchment area. The water from streams, rivers and other surface sources is prevented from recharging the aquifers. The only water that cannot be excluded from infiltrating into the mine is the rain water. Konkola Mine Catchment area has a significant number of surface water bodies that recharge the aquifers in the mine. Table 3-2 shows the amount of recharge into the mine from surface water sources.

Table 3.2: Recharge Quantities in KONKOLA Catchment area, Chililabombwe area, Copperbelt Province, Zambia

Source	**Recharge m^3/d**
Kakosa	27,000
Kafue Western Limb	58,840
Kafue Eastern Limb	93,230
Lubengele Dam	12,800*
Lubengele Stream	2,300*
Total	**194,170**

*Note: * No current recharge has been determined since the works done during the KDMP volume three report. The numbers used is as determined by the KDMP Vol. 3 report (KDMP V3, 1995). Though these numbers may not represent the current seepages as the area affected by mining induced* *subsidence has increased as compared to the time the study was done but can still be used as a minimum.*

Table 3.2, shows that by surface exclusion alone 194,170m^3/d of water can be excluded which accounts for about 56% of the daily pumped out water. This number does not include the seepage from the canal, thus more water can be excluded from the mine by this method. It is important to note that surface exclusion method cannot lower the water table but can prevents recharge of the aquifers and can reduce the amount of water being pumped per day, hence this method cannot be used as an exclusive dewatering method but needs to be coupled with another dewatering method.

There are various ways the surface water bodies can be prevented from recharging the aquifers, these are:

- Stream or river diversion from fault zones and caving areas; Kafue River and Kakosa Stream;
- Stream, river and discharge canal lining using flexible waterproof material like geomembrane or concrete lining: Kafue River and Kakosa Stream;
- Lubengele Dam Sealing;
- Lined canal refurbishment; and,
- Unlined canals sealing or lining.

3.3.2 Surface Deep Wells

The surface wells target to dewater the Hangingwall Aquifer that is above the 1150mL cave line in the Bancroft Mining area. The simulation of drawdown for this method gave a positive result in lowering the water table below the 1150mL Cave line and also in avoiding the ore gap in the Bancroft Mining area. This method involves sinking ten holes targeting the 1150mL cave line area.

This method requires comprehensive site investigations as a pre-requisite and are planned to take 6 months. The site investigations are critical for this method in order to acquire the geological, geophysical and hydrogeological data for the site. These studies will assist the drillers to accurately assess the drilling conditions of the area and associated risks. Besides this, the survey will also show a more detailed characterization of the lithology and groundwater flow condition. This will measure also the fractures and bedding planes of the area where wells will be located and also determine the approximate thickness of the bedrocks and location of faults, buried river channels, fissures and solution cavities. The studies to be done are listed below:

- Surface geophysics surveys for Characterisation of ground conditions and structural discontinuities at target drill sites;
- Diamond core drilling of pilot investigation boreholes;
- Downhole packer permeability (Lugeon) testing;
- Downhole wireline logging;
- Installation of multilevel vibrating wire piezometers;
- Installation of time domain reflectometer equipment; and,

Laboratory testing of geomechanical properties of main lithological units.

Three holes 333cm diameter are to be drilled during the site investigation at a depth of 1650m, one well is to be equipped with a submersible pump while the other two wells are to be equipped with piezometers. These wells will show the expected performance of the ten wells and any changes needed to be implemented before the ten dewatering wells are drilled.

Ten wells are to be drilled in the Bancroft area targeting the HWA. These wells are to be accurately drilled and positioned. The location of the holes plays a crucial role as they have to be located outside the caving area and in a position where they intercept the targeted portions of the HWA. The proposed location of the wells are shown in Figure 3.11.

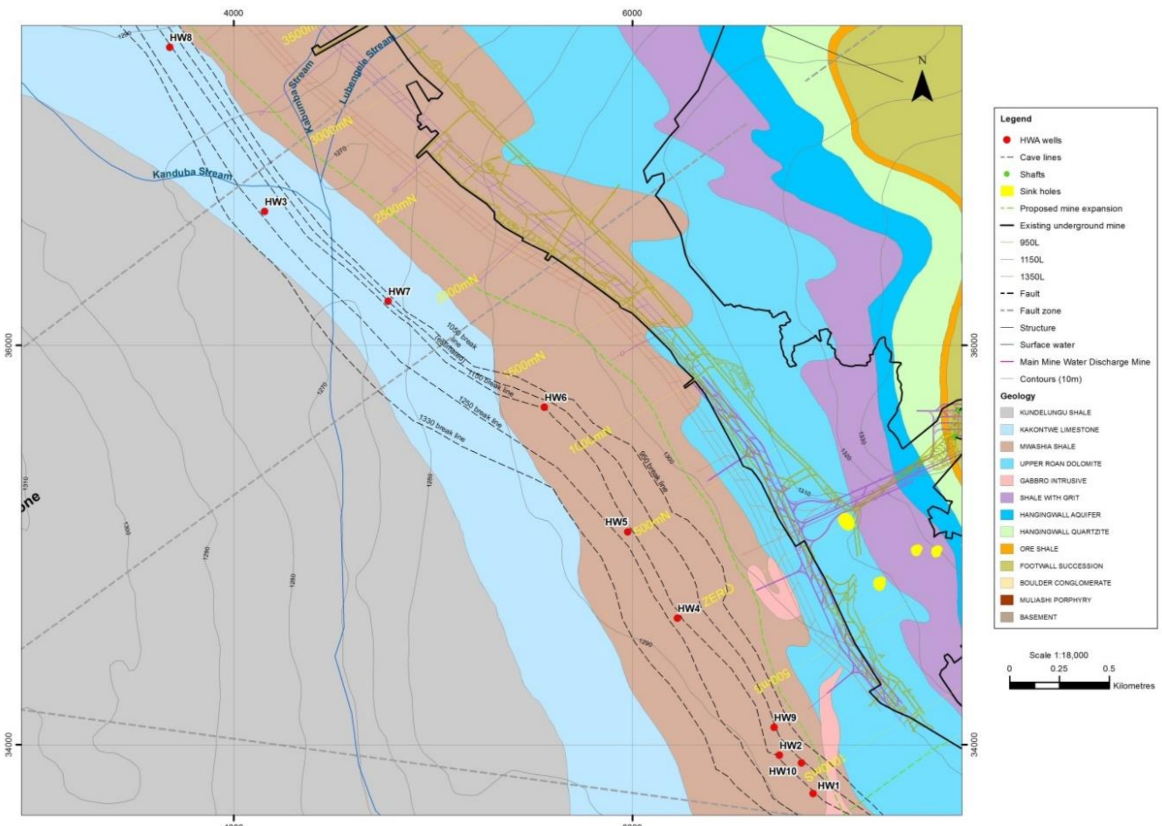

Figure 3.11: Map showing Proposed Surface well location (WSP Parsons Brinckerhoff, 2017)

The ten holes are planned to be drilled in 1 year six months by two drilling machines, drilling of one hole is planned to take about 3 months. The diameter of the wells is about 30cm and the expected flow is 50L/s. The depth of the holes are shown in Table 3.3.

Table 3. 3: Proposed Well depths in the Bancroft Mining Area, Konkola Catchment area, Chililabombwe area, Copperbelt Province, Zambia

Proposed Dewatering Well	HW1	HW2	HW3	HW4	HW5	HW6	HW7	HW8	HW9	HW10
Depth to base of HWA according to SWS model (m)	1121	1191	1337	1413	1603	1584	1386	1124	1043	1049

All the holes with an exception of HW9 and HW10, are located in the 1150mL breakline, HW9 and HW 10 are located in the 1050mL.

The expected flow rates simulated using updated groundwater model for the wells are shown in Table 3.4

Table 3- 4: Proposed Well Flow rates at Konkola Mine, Chililabombwe area, Copperbelt Province, Zambia

Well	Available drawdown assuming 30m water head in the well	Initial simulated pumping rate (L/sec)	Simulated pumping rate 3 months after commencement of pumping (L/sec)	Simulated pumping rate 6 months after commencement of pumping (L/sec)	Simulated pumping rate 12 months after commencement of pumping (L/sec)
HW1	153.1	50	50.0	50.0	33.7
HW2	89.7	50	41.6	37.9	31.6
HW3	112.7	50	38.2	32.3	26.9
HW4	83.7	46.7	33.2	28.8	24.0
HW5	266.0	50	45.4	39.1	28.8
HW6	626.9	50	37.6	29.2	21.4
HW7	107.4	50	35.1	27.6	19.2
HW8	52.8	50	38.7	37.0	35.7
HW9	126.3	50	50.0	50.0	8.6
HW10	152.4	50	50.0	50.0	16.5

Table 3-4 shows significant drawdown after just a year of pumping. The highest drawdown is achieved in HW6 (626.9m) and the lowest drawdown was found in HW8 (52.8m). The simulated water profile after about two years of pumping, is shown below in Figure 3.12.

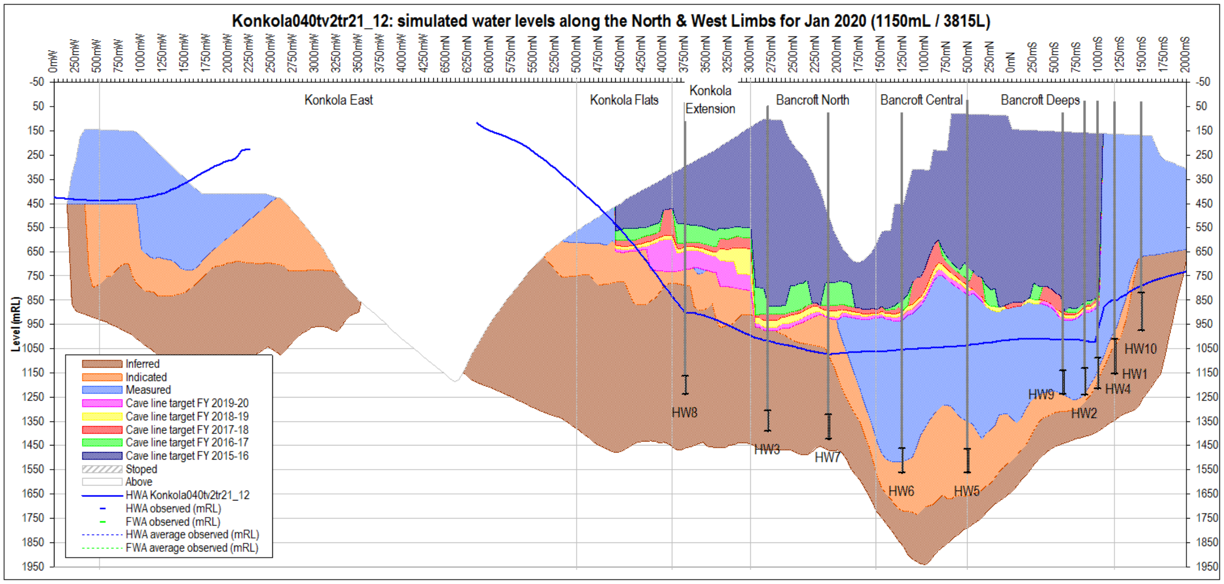

Figure 3.12: Water profiles after two years of pumping (WSP Parsons Brinckerhoff, 2017)

The dewatering target for the elevation of 1050mL area are achieved along most of the strike of the west limb and particularly Bancroft North, Bancroft Central and Bancroft Deeps sections. Once implemented on time, the method helps the mine avoid the ore gap in the Bancroft Mining areas. The drilling, completion, testing and commissioning of each hole will take 3 months, thus in those areas where the holes were commissioned early and the water table is below the cave line, mining can start even before completing the whole project.

3.3.2.1 Costs

The cost are of this method are illustrated in Table 3.5

Table 3.5: Cost of Surface Dewatering holes

Details	**Costs $**
Site Investigation and Management cost	4,140,523
Drilling Cost for the ten wells	35,000,000
Cost of Pumps (10)	5,500,000
Total Cost	**44,640,523**

3.3.2.2 Risks

Though the surface wells have shown great success in lowering the water table below the cave line, this method is a high risk method. The following are the risks associated with its implementation:

- Drillers experience in drilling in the Konkola Stratigraphy which is highly faulted and fractured. Drilling in this stratigraphy beyond a certain depth becomes problematic as experienced at Lubambe Copper Mine PLC when they attempted the surface holes. At depths of 500m to 600m the drilling become problematic, the driller experienced fluid losses and at 850m the drill bit became stack and hole was abandoned. The entire project was abandoned because of the stack drill steel and financial problem which were face by Lubambe;
- Missing the targeted points on the aquifers. This is a possibility as the holes are long and susceptible to deviation;
- Time sensitive: any delay in implementation would result in a gap in ore production in the Bancroft mining area;
- Long term security of associated surface infrastructure; and
- Some surface infrastructure will be located outside current mine concession area and will thus require negotiation for access and acquisition.
- Drying up of some hole after one year of commissioning.

3.3.2.3 Positives

The entire project needs about two years to implement and significant drawdowns are achieved within a year of operations. The method may be implemented well in advance of development of mine haulages on 1,150mL and 1,350mL. The method has shown successful results in mines where it has been used, at Barrick Gold Nevada Goldstrike mine, 520m drawdown was achieved as at 2014 (Zhan, J, 2014).

4.3.3 Conventional Dewatering

Conventional dewatering methods also known as breakthrough method has been used at Konkola Mine since inception of mining in 1957. The targeted area for dewatering is the area that falls within the 1150mL cave line. The method requires the setting up of the new main level, 1150mL, requiring 30,734m to be mined, which include a main level, drain drive, 19 dewatering centers from 200mW to 1200mS and associated dewatering infrastructure.

Before the LPCA is intercepted, the phase one of the 1390mL Pump chamber needs to be constructed, this will involve the mining of 2269m development which includes the establishment of sumps at 1350mL. Figure 3.13 shows planned 1150mL development.

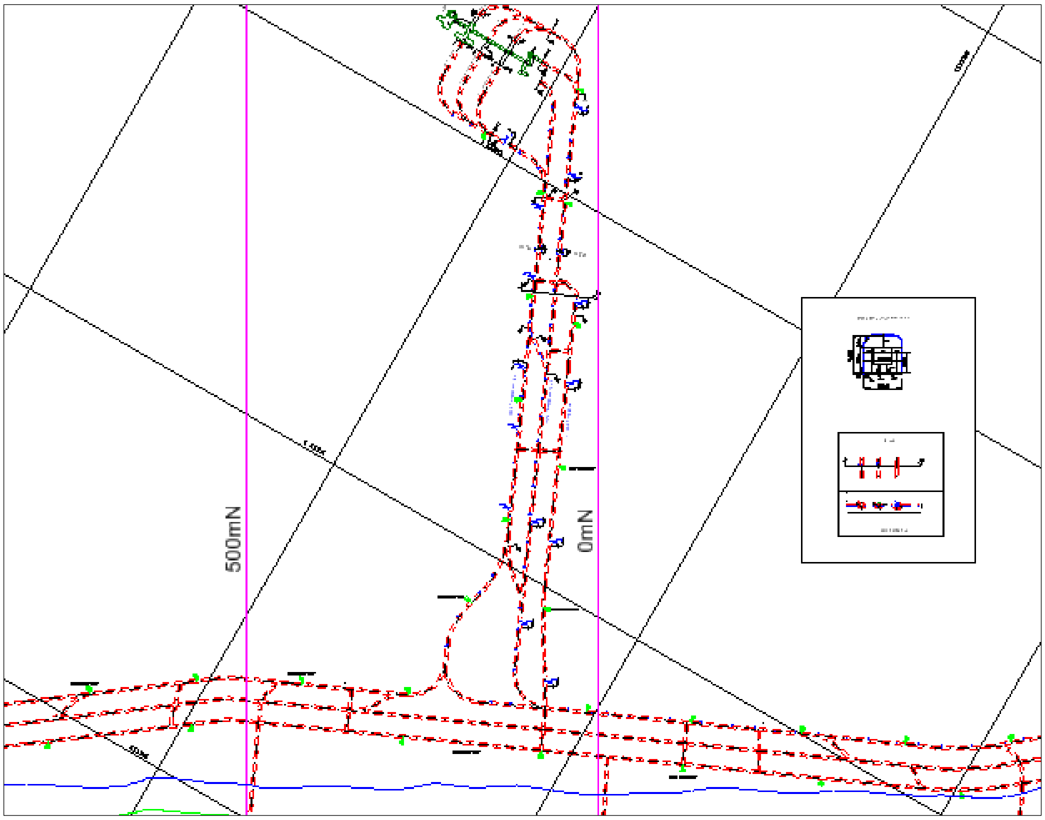

Figure 3.13: 1150mL Development Plan for 4 Shaft, Konkola Mine, Chililabombwe area, Copperbelt Province, Zambia

The 1150mL development will take 36months to mine and 18 to 24 months to dewater. The entire project needs approximately 60 months to mine and dewater that is 5 years. Historically 18 months of dewatering has been enough for dewatering of aquifers at Konkola. The development rate is 34m/day. The water tight door installation and associated developments will take 6months and this has been taken into consideration in the five years. The Gant chart shows the schedule of this project in Figure 3.14.

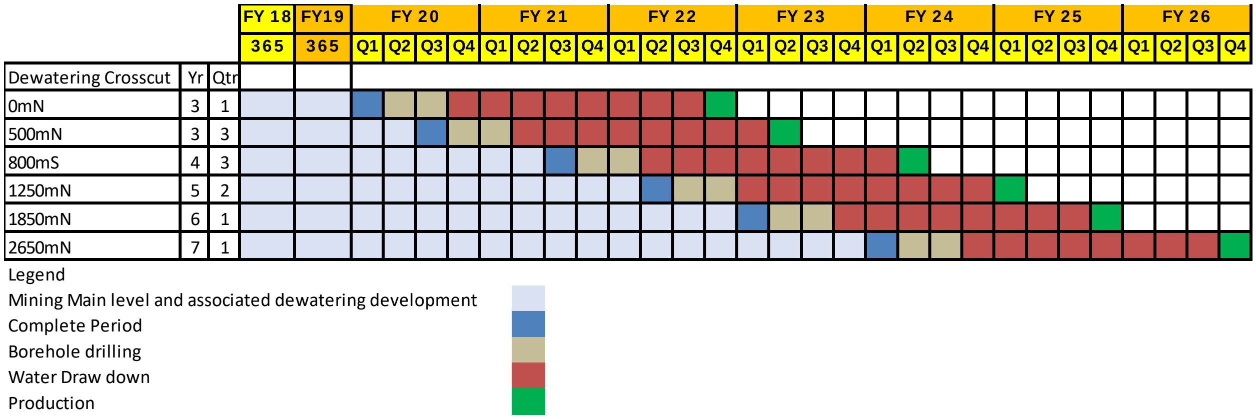

Figure 3.14: Schedule of development at Konkola Mine

In this schedule one hole is planned to be drilled and equipped in one month, the drilling rate is approximately 300m/month and the average length of hole is 200m. A dewatering crosscut constitutes 10 holes which are planned for six months using two drilling machines, the period for drilling has taken into account that it's not all holes that are drilled which reach the planned target and those that miss need to be re-drilled. This method offers a chance of re-drilling once the target is missed, hence offering flexibility.

The mining of 1150mL will start simultaneously with the mining of the 1350mL main Level and 1390mL pump chamber. Figure 3-15 shows the planned 1150mL and 1350mL

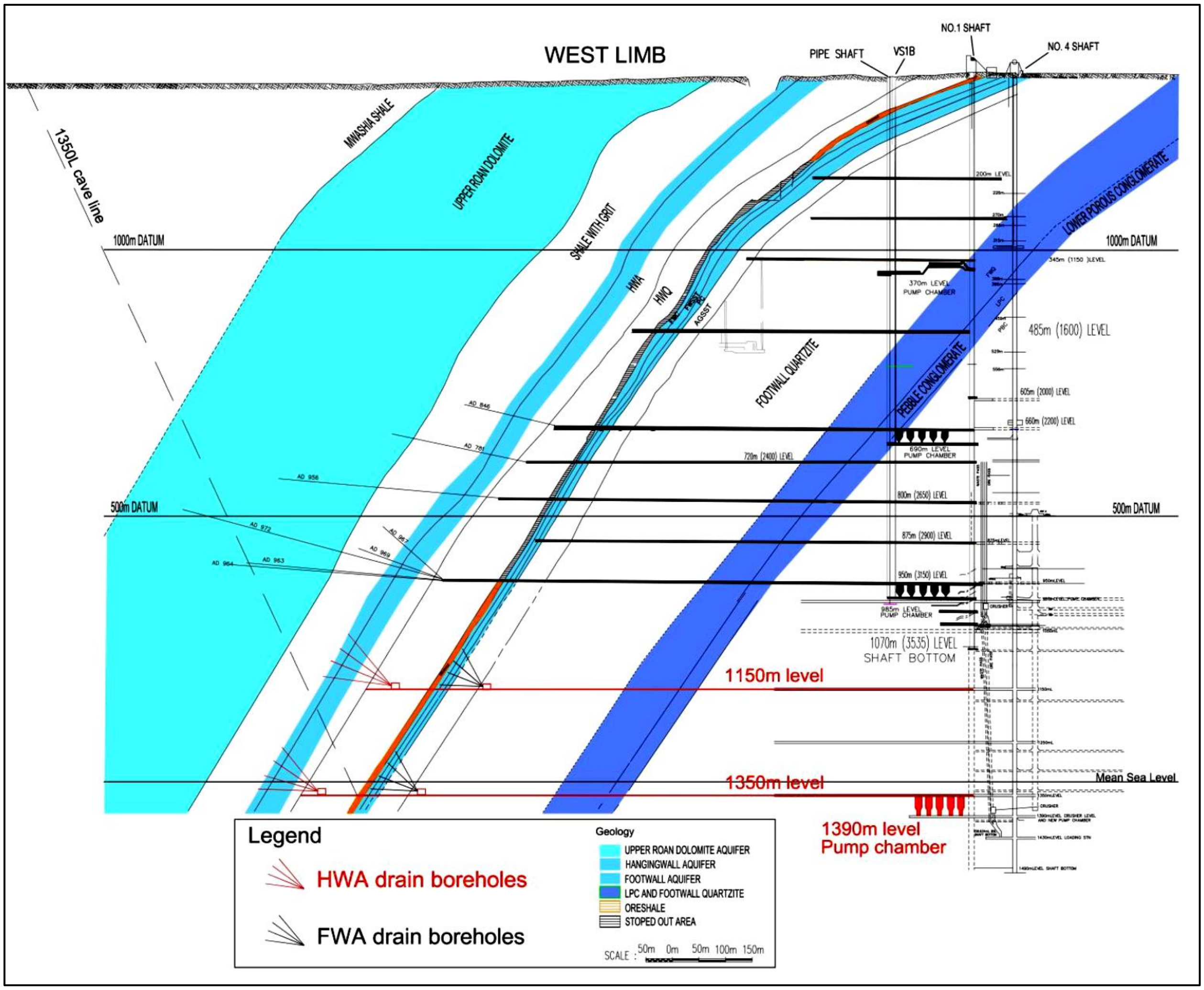

Figure 3.15: 1150mL and 1350mL development at Konkola Mine (WSP Parsons Brinckerhoff, 2017)

3.3.3.1 Cost

The cost considered for the project is the cost of dewatering crosscuts and not the cost of the main level and pump chamber, because these cost irrespective of any method used have to be incurred. This method will involve the mining of dewatering crosscuts and drilling of dewatering boreholes. The cost of mining the dewatering crosscuts and dewatering boreholes are as follow: The average linear meters for the dewatering crosscuts: 70m;

- The width and height are: 4.2m * 4.2m;
- The volume of the excavation: 1235.8m^3;
- Cost of the crosscut= 1235.8*$156 =**$192,629**;
- Average length of holes (HQ=96mm diameter): 200m;
- No of holes in a cross cut is 10, thus total meter drilled in one cross cut: 2000m;
- Cost of the 10 holes: 2000m* $107.67/m= **$215,340;**

- Total cost for a Cross cut: **$215,340+ $192,629= $407,969;** and
- **Approximate cost for 19 crosscuts: $ 7,751,411.**

3.3.3.2 Risk

The biggest risk to this method is mining the planned meters in the stipulated timeframe. The timeframes are highly sensitive as missing the targets means the ore mining gap becomes a reality. Figure 4-16 shows the decline in production associated with this method.

3.3.3.3 Positives

There are several positives to implementing this method, these are:

- Experience in method execution, this method has been used from inception of mining at Konkola Mine;
- Averts a gap in ore mining once implemented on time;
- The method offers flexibility: when a hole misses the targeted area in the aquifers, another hole is easily drilled at less cost; and
- Holes are short hence less susceptible to deviation.

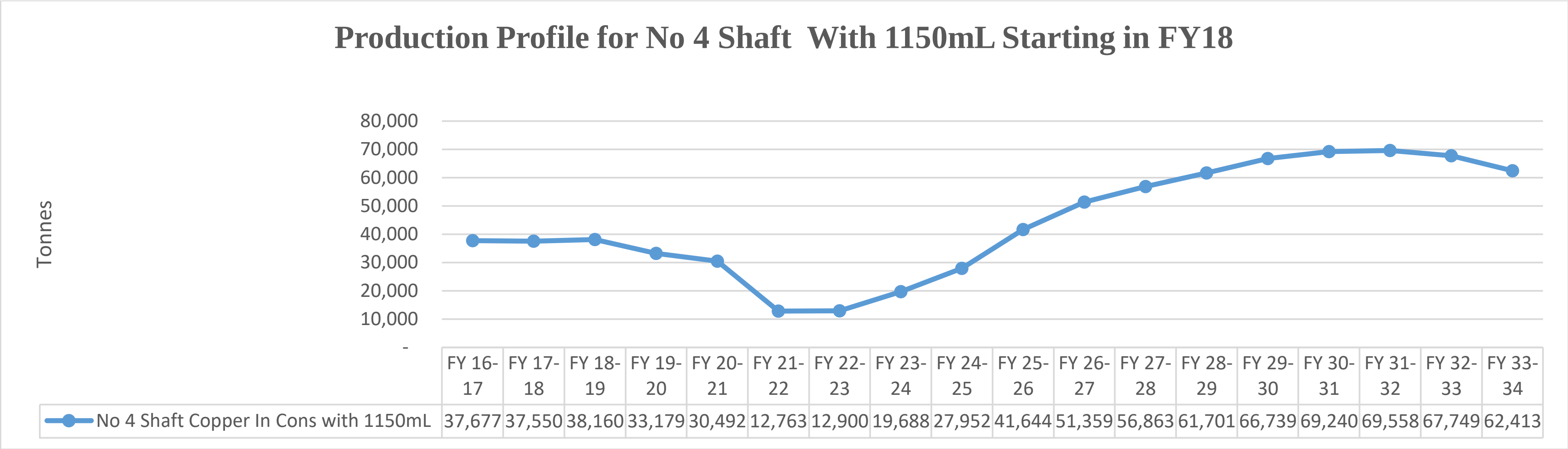

Figure 3.16: Production Profile for No. 4 Shaft at Konkola Mine, Chililabombwe area, Copperbelt Province, Zambia

3.4 Impact of backfilling mining method on hydraulic inflow paths

The main reason for dewatering is to lower the water table below the cave line because the area above the cave line is affected by subsidence hence it is a fractured zone (effects of subsidence discussed in section 2.8). Fractured zones are one of the main ground water flow channels to the mine (Mulenga, 1993).

Backfilling methods reduces the rate of subsidence by 80 to 90% in comparison to caving methods (Straskraba and Abel, 1994). The backfill fills the stope void forming an artificial support to the mine structure controlling both local stope wall behaviour and mine near field displacement, hence reducing the ground movement. The cave line displacement from A level to B level is **X** for a backfilled area (backfill shown in red) which is less than for an area that is not backfilled A level to C level is **Y** as shown in Figure 3-17.

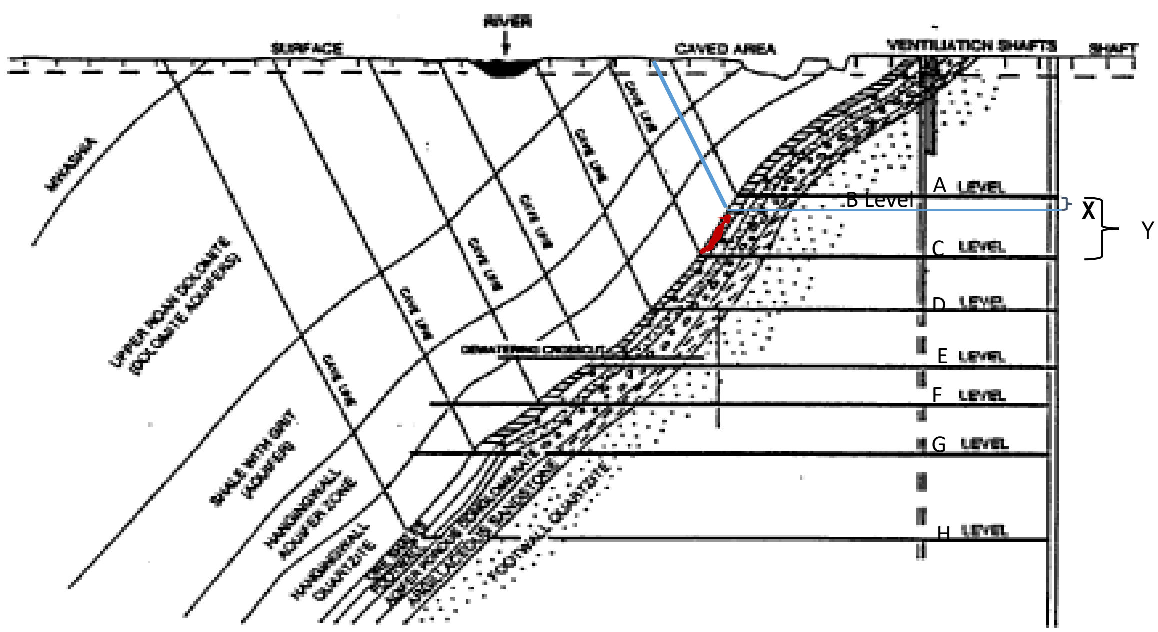

Figure 3.17: Cave line in mines with and without backfill (Straskraba and Abel, 1994).

The dewatering requirement is significantly reduced in mines that use backfill as shown in Figure 3.17 above. Typically, with backfill mining methods, it is necessary to dewater nearby Footwall and Hangingwall aquifers which are adjacent to the ore zone. However, Hangingwall aquifers approximately 40m or more above the mined ore would not probably drain into the mine (Straskraba, 1991). The need to drain these, is site specific and is determined by various factors: presence of faults, anticlines and fractures; pore pressure; and caved areas. Any substantial decrease in the dewatering should be based on geotechnical and hydrological studies and should be supported by rock mechanics and hydrogeological monitoring. A reduction in dewatering requirements would result in a significant reduction in costs, these are cost of:

mining water drives; mining dewatering cross cuts; drilling deep dewatering bores; and pumping. This can help in offsetting the cost of backfill.

Backfill infrastructure already exists at Konkola Mine, thus reducing the costs of migration from stoping methods to backfill as no new plant would be set up, though the only costs to be encountered are the costs of plants repairs and pipe rehabilitation/ relining. The mine has three backfill plants, these are KCM 1 BF, KCM 2 BF and KCM 3 BF plants. The KCM BF1 and 2 are located at No. 1 Shaft while the new plant KCM 3 BF is located at No. 3 Shaft. The oldest backfill plant is the KCM 1 BF followed by the KCM 2 BF built in 2002 and the KCM 3 BF built in 2012. The KCM 1 BF is designed to produce hydraulic fill while the KCM 2 BF is for cemented fill while the KCM 3 BF which is also known as the Waste Rock Crushing and Milling Plant is designed to produce coarse material. The hydraulic fill is to be mixed with the coarse material to form a high strength material with good permeability.

The mine once used Hydraulic Fill in the areas where Post Pillar Cut and Fill method (PPCF) was being used. However, due to the plant being un-available (broken-down), the mine is currently using waste rock backfill. PPCF is applied in areas with dips between 10° and 25°. The fill provides a floor for subsequent lift (Figure 3-18).

The suitable backfill blend for Konkola is a mixture of Hydraulic Fill and waste rock crushed at 2.83mm aggregate, this blend must be in the ratio of 3:7 by weight of hydraulic fill and the aggregate, respectively, with addition of 4% Portland cement. The mixture requires not less than 28 days to attain a required uniaxial compressive strength of 1MPa (Mutawa, 2011).

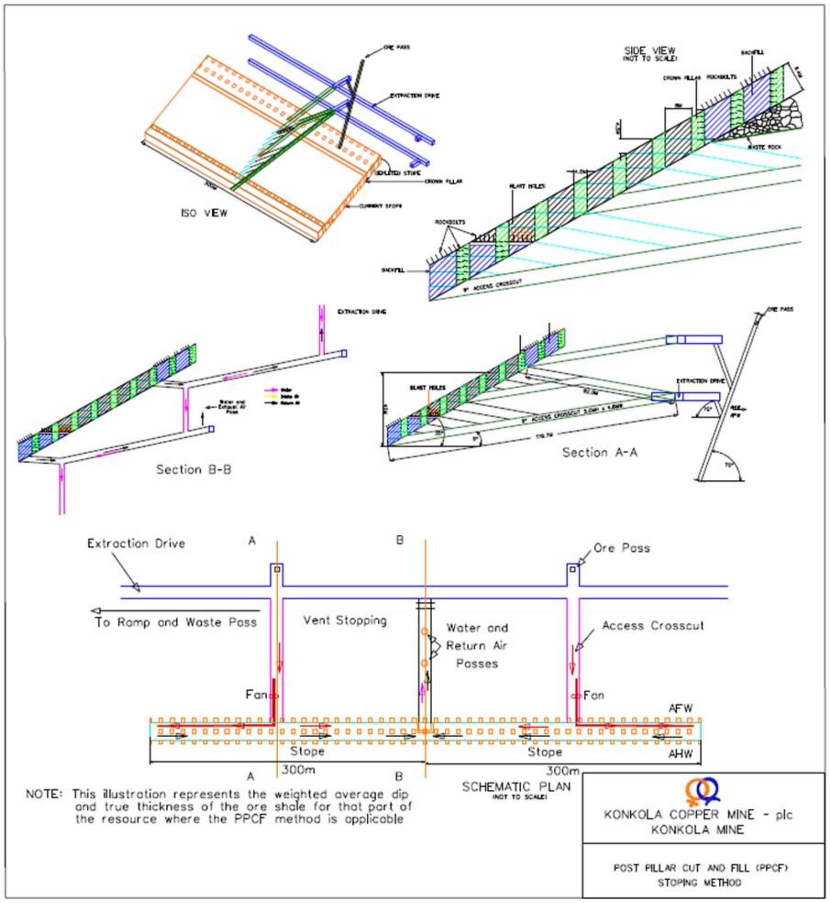

Figure 3.18: Post pillar cut and fill mining method used at Konkola Mine–

4.0 Discussion

The Konkola Mine Catchment area, which is about 240km^2 has three sources of water, these are rainfall infiltration, seepage from surface water bodies and water stored in aquifers. The rainfall infiltration is 20% of the average rainfall of 1274.3mm/year, which is 254.9mm/year and 167,605.5 over the entire catchment area which finds itself to the mine as a result of faults and fracture zones. The rainfall infiltration rate is high because of the presence of an extensive amount of saprolites in the Konkola Stratigraphy. A small amount of water from the rainfall can be prevented from infiltrating into the mine by the burring the dambos. The total seepage of the surface water sources is approximately 194,170m^3/d which is about 55.5% of the water pumped per day by the mine. The storage aquifer release is between 40,000and 50,000m^3/d.

There are three dewatering methods which were considered: surface exclusion, surface deep wells, and conventional method. The surface exclusion methods offers more than 194,170 m^3/d reduction in the water recharging of aquifers, which will significantly reduce the amount of water pumped out of the mine per day. The lining and diversion of streams and rivers can reduce the daily pumping rate by about 55.5%. The surface exclusion cannot be used alone but must be used with another dewatering methods.

The surface deep wells showed great success in lowering the water table though the method is high risk. The method's success will depend on the results from the site investigation. The cost of implementing this method is about \$44,640,523. Some wells will go dry after one year of being commissioned casting a shadow on the \$3.5m cost.

The other method reviewed was the conventional dewatering, this method is low risk and has been used since inception of mining. The mine management has enough experience this method. This method shows a decline in Copper in Concentrate production from 30,492t in the financial year (FY) 2020-21 to 12,763t in FY 2021-22. During this time the dewatered reserves will be at their lowest but however due to the completion of dewatering at 0mN in 2022, an upswing in production starts. From the year 2022 onwards, the production trend increased until it reaches 70,000t in 2030 as shown in figure 3-16. The approximate cost for this method is \$7,751,411 which is about 17.4% of the cost of the surface boreholes.

The surface deep wells when compared to the conventional methods offers an opportunity to mine to lower the water table faster and to mine without any slump in production. The major disadvantage of the surface deep wells is the high cost and the risk associated with this method, while the greatest advantage of the conventional methods is less risk and less costly bearing the fact that the mine is facing financial difficulties. The conventional methods having been executed at Konkola successfully from inception of mining give confidence to the mine management unlike the surface deep wells which have never been used at Konkola. The surface deep wells require site investigation (cost \$4,140,523) while the conventional method does not need any site investigation. In terms of practicality and cost the conventional dewatering methods in better than the surface deep wells.

Mines that use backfill have a less dewatering requirement compared to mines that use caving and open stoping methods. This is as a result of subsidence, subsidence is more pronounced in mines that use caving or open stoping methods than in mines that use backfill. Backfill reduces the area affected by subsidence hence reducing the dewatering requirement as the amount of required to drop the water table below the cave line is reduced. For Konkola Mine the suitable backfill is a mixture of Hydraulic Fill and waste rock crushed at 2.83mm aggregate, the blend being in the ratio of 3:7 by weight of hydraulic fill and the aggregate, respectively, with addition of 4% Portland cement (Mutawa, 2011).

5. 0 Conclusion and Recommendation

5.1 Conclusion

The study has established that the best and viable method of dewatering g Konkola Mine is the conventional dewatering method as it is easy to implement and cost-effective. The approximate cost for using the conventional method was determined to be \$7,751,411 while the dewatering method using surface deep wells was determined to be \$44,640,523. This presents a potential serving of about \$36,889,112. Surface exclusion method can exclude about 194,170m^3/d of water (which is about 55.5% of the water pumped out of the mine) from recharging aquifers. Besides this, about 176 605m^3/d of rainfall infiltrates into the mining catchment area. The use of backfilling method compared to caving and open stoping methods has the potential to significantly reduce the dewatering requirement.

5.2 Recommendation

The study recommends that the Mine:

I. strongly adheres to the dewatering plan and schedule of development of dewatering infrastructure
II. implements surface exclusion methods of water
III. moves from open stoping to method that employ backfill
IV. conducts more seepage test
V. consistently updates the dewatering model
VI. conducts site investigation to get more information on the Konkola stratigraphy
VII. conducts hydraulic tests in the selected existing and future drain holes to obtain true hydraulic parameters of the rocks.
VIII. rehabilitates all canals in the Konkola area

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Visible Light Communication: Enhancement of data rates and potential applications

Chamupunngwa Singwa[1], Sebastian Namukolo[2]

Abstract

Visible light communication (VLC), explores the utilization of the visible light portions of the electromagnetic spectrum as a means of addressing the shortage in the radio spectrum. The idea gained momentum due to the emergence of smart and energy efficient Light emitting diode (LED) and photo sensor technologies. A lot of VLC physical layer schemes that are focused on a particular modulation scheme have been proposed and some implemented. Additionally, hybrids of such schemes have been proposed. However, much of such works are focused on adapting the initial idea of communication using Visible light to the already commercially available LED bulbs i.e. using them for both illumination (their current use) and communication, and this limits the maximum possible data rates. The work presented in this paper is a literature review on the schemes that have been proposed and some basic evaluation experimental setup have been implemented at the Department of Electrical and Electronics Engineering and a proposal of a possible direction that could be taken in designing such systems aiming at maximizing data rates is proposed

Keywords: Visible Light Communication(VLC), Light emitting Diode(LED), Radio spectrum, Photo Sensor

1. Introduction

Visible Light Communication (VLC) is a subclass of wireless optic communication. It is the sending and receiving of information using visible light in a wireless channel. It uses a visible light spectrum (400-800 THz), K. Anish, R. Bute et al (2018).

VLC communication system, just like other communication systems, is basically made up of a transmitter, channel and receiver as depicted in the block diagram below. The transmitter section is implemented using artificial light sources such as Light Emitting Diodes (LEDs) A. Nath, et al (2015) A channel is basically air space and a receiver is basically a photo detector which can be Photo diodes, Nath, et al (2015) or photovoltaic cells, RF Wireless World (2012).

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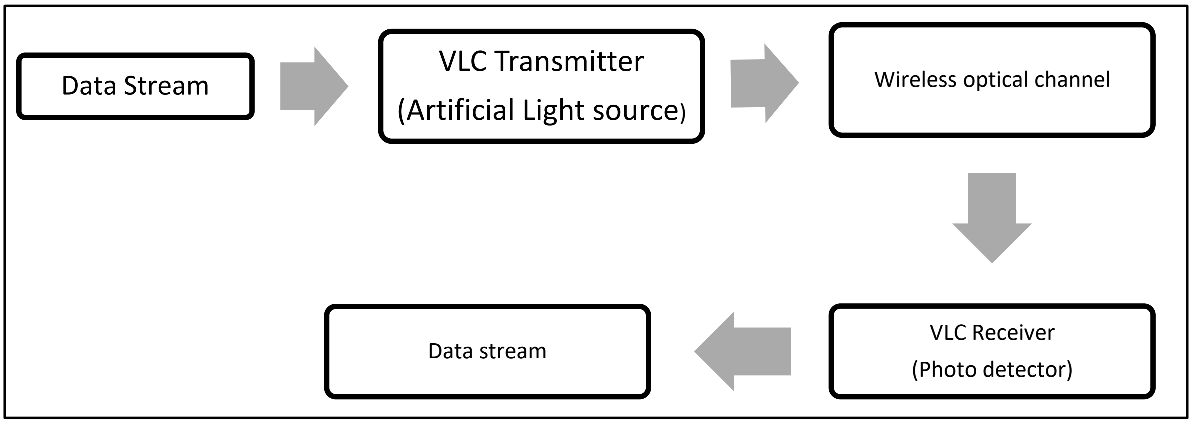

Fig. 1.1 A simple Visible Light Communication system

Research is being carried out in this area of communication as this promises to be, in some application, a better substitute or a compliment for communication systems that use radio waves. Additionally, this communication technology can be used where radio communication technologies are not possible or applied with a great amount of difficulty, such as undersea navigation, M Rhodes (20xx).

This paper gives a literature review on various VLC systems that have been proposed and implemented and then proposes a design of one such a system that optimizes data rates. It then concludes with potential applications of a visible light communication technology.

1.1 Problems of radio communication that need addressing

Radio communication has some challenges that cannot be ignored. The challenges include the following;

i. *Limited available bandwidth*

   The limited available bandwidth that the radio band has is now being seen to be one of the biggest challenges in radio communication. The number of devices that are getting connected to the internet, according to Cisco, Cisco (2015), is projected to reach 50 billion by 2020 with an average of 7.6 devices per person as shown in fig.1.2. This scenario may be worsened further by the Internet of Things (IoT), an issue that can be resolved by the complementing/supplementing of radio with visible light as visible light has a bandwidth that is 10,000x that of radio.

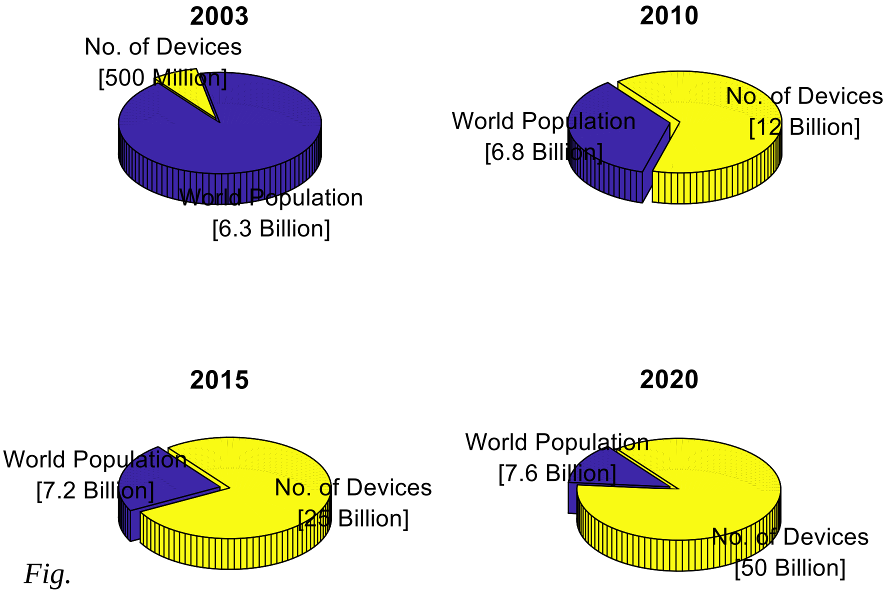

Fig. 1.2 Proliferation of radio communications devices (cisco)

ii. *Security Concerns*

Security concerns that the use of radio waves imparts into the user are due to the fact that no one can have a complete control on who can tap into the radio signals apart from the use of security techniques like encryption. This fact comes about due to the ability of these waves to penetrate through walls and other materials, Anurag et al (2015). Since it's not always that the coverage of such waves is within sight, unauthorized personnel with right decryption skills can sniff at the packets carrying information under transmission.

The use of VLC, in indoor wireless communication, can eliminate this concern as visible light does not penetrate walls making it impossible for sniffers beyond the confinement of a building to access the communication signals.

iii. *Interference among radio waves*

Interference among radio channels or radio waves is also one of the limitations and hence a challenge of using the radio frequency band. Additionally, some facilities such as petrochemical facilities don't need radio waves at closer proximity, R. R Sharma et al (20xx). Techniques to circumvent the earlier issue have been well developed. But in some applications, the problem still remains. The reason is that the techniques that have been developed are meant for channel interference mitigation and to some extent, radio signal interference mitigating. The latter problem of radio signal interference cannot be fully controlled because some sensitive device like some of the devices used in medical care systems which are implanted into patients and communicate to the outside world using radio waves are designed to work, at optimum, in free radio signal interference environments, Anurag et al (2015). Ergo, the use of radio signals for other means of communication is highly avoided in such environments.

Since visible light does not have a negative effect in such sensitive environment as its usage for illumination in these environments is a well-known fact, it can be used for communication in such environments

2. Modulation schemes for VLC systems

Various modulation schemes have been proposed and some have been implemented with the use of LED bulbs as transmitters and Photo diodes as receivers. Concordantly, the modulation techniques to be explored in this paper are going to be based on the same kind of transmitters and receivers. VLC transmitters make use of direct detection and/or intensity modulation schemes as it is not easy to modulate the phase of the carrier or the frequency, E. Monteiro (2013). This limitation is due to the way LEDs are made as it is only possible to emit a light pulse if a positive voltage of a certain level is present and only the intensity of the emitted pulse can be varied, R. Owens (2011). Therefore, the direct use of traditional radio modulation schemes is not possible, E. Monteiro (2013). There exist several modulation schemes that are under study for use in VLC. The following are some of the modulation schemes that have been proposed and the ones that are still under study.

On-Off Keying

On-Off Keying is the most basic of VLC modulation schemes. In OOK, data are sent via the presence or absence of light. Two forms of OOK are used. The first is OOK Non-Return-To-Zero (NRZ), where a binary “one" is a light pulse, and binary “zero" is the absence of a light pulse.

The other form uses Manchester coding in which a binary “one" is represented by an absence of light and then a presence of light and a binary “zero" is represented by a light pulse followed by an absence of a light pulse. Manchester coded symbols use twice the bandwidth of NRZ pulses. Manchester coded symbols offer flicker-free communication and a good synchronism property. The need for flicker-free illumination is due to some health concerns that have been raised about the effect of flicker on the human body, A. Wilkins, J. Veitch, and B. Lehman (2010), CIE (2014). At higher frequencies, flicker has no negative effects and so, OOK-NRZ can be used.

Multiple Sub-Carrier Modulation

In addition to single carrier modulation methods, such as OOK, VLC is also compatible with multiple sub-carrier modulation (MSM) techniques. MSM techniques are modulation schemes where information is modulated onto orthogonal sub-carriers. The sum of the modulated sub-carriers is then modulated onto the instantaneous power of the transmitter, A. Böcker, et al (2015). Since the subcarriers are orthogonal, as in OFDM, it is designed for parallel transmission of data from a single transmitter hub to several receivers.

Sub-Carrier Index Modulation Orthogonal Frequency Division Multiplexing (SIM-OFDM)

In this, an additional dimension is added to conventional 2D amplitude/phase modulation (APM) techniques. The key idea is to use the sub-carrier index to send information to the receiver end, A. Böcker, et al (2015).

Color-Shift Keying (CSK)

This is where multi-colour LEDs are used to transmit information with each colour band acting as a channel. The commercially available multi-colour LED bulbs are the RGB LED bulbs. The data symbols can be generated by a combination of these colours E. Monteiro (2013). The combination of colours can be done by making use of colour mixing schemes such as the CIE chromaticity chart.

2.1 Enhancement of data rates

The modulation schemes that act as base modulation schemes are OOK-NRZ, OOK-Manchester coding and CSK. The three basic modulation schemes and the comparison of their various figure of merits, E. Monteiro (2013). R, R Sharma et al (20xx). A. Böcker, et al (2015) are outlined in the table1;

Table 1: The comparison of the three basic modulation schemes

Modulation Scheme	Relative Complexity	Relative Data rates (Theoretical)	Relative Spectral efficiency	Flicker	Dimmable
OOK-NRZ	least	Slightly High	Slightly efficient	yes	yes
OOK-Manchester coding	Slightly complex	low	least efficient	no	yes
CSK	Complex	High	efficient	no	Slightly dimmable

- *The flicker parameter is there to cater for health effects flicker has on people.
- *The dimmable parameter caters for the ability of a system to communicate at any chosen light intensity, a property that makes it possible for such a system to be used in different conditions such as day and night times.

From the table, it can be seen that CSK has higher data rates at the expense of complexity, OOK-NRZ is least complicated though has less data rates and OOK-Manchester coding is slightly complicated and has least data rates. One advantage that OOK-Manchester coding offers is synchronism.

The choice of the modulation scheme highly depends on the application just like in all wireless communication systems. In this paper, an approach that aims at enhancing data rates is presented.

CSK modulation schemes uses three colour channels which are red, green and blue. This is so all because most of the commercially available multi-colour LEDs are Tri-Colour LEDs, Kitronik (2016). Since the approach of researchers has been to use commercially available LED bulbs, which are normally designed for illumination, for wireless optical communication as in E. Monteiro (2013). R, R Sharma et al (20xx), A. Böcker, et al (2015). This approach has resulted in the availability of only three colour channels limited by the tri-colour LEDs which are the transmitters of a VLC system. The approach proposed in this paper is to add more colour channels in the design of multi-colour LED bulbs. For example, a yellow channel may be added so that four channels are available. It must be emphasized here that the colour combination must be carefully chosen so that the final output colour is suitable for the given application. For example, a suitable colour scheme such as the CIE 1931 xy chromaticity chart, Ashdon (1931), can be used so that the different combinations of colours chosen in the design appear white in applications that involve illuminating humans as white light in illumination appears natural.

Colour channels may be further divided into smaller sub-channels as each colour is a range of frequencies. This approach and the earlier one would increase the complexity of the system by much but would also increase the data rates. The first approach of increasing the number of colour channels need a redesign of multi-colour LED bulbs so as to accommodate the additional colour channels. On the receiver part, multi-colour sensors that can detect more than three colours are already commercially available. Fig 2 illustrates a schematic of a four-colour channel VLC system where yellow is the additional colour channel.

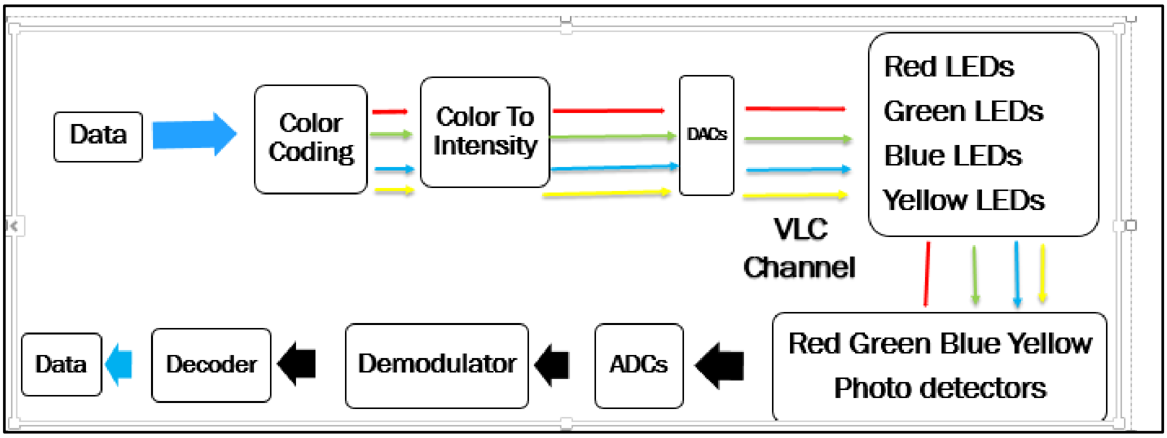

Figure 2. A CSK VLC system that uses four channels

3. Implementation of the VLC system

To test the workability of Visible Light Communication a system that uses OOK-NRZ was designed and implemented using development boards and two laptops that acted as severs. Various parameters such as Bit Error Rate (BER) and complexity were measured and recorded. []. The implementation block and circuit diagrams are as shown in figs 3 and 4.

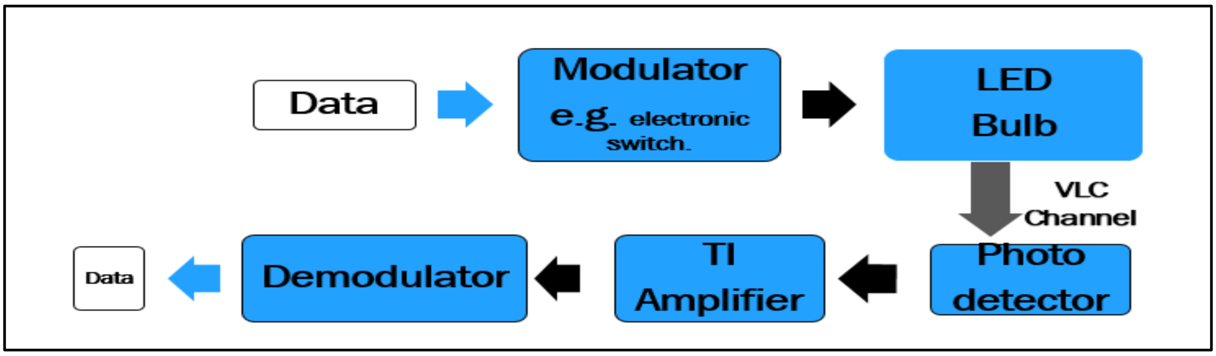

Fig.3. Block diagram representation of an OOK-NRZ system

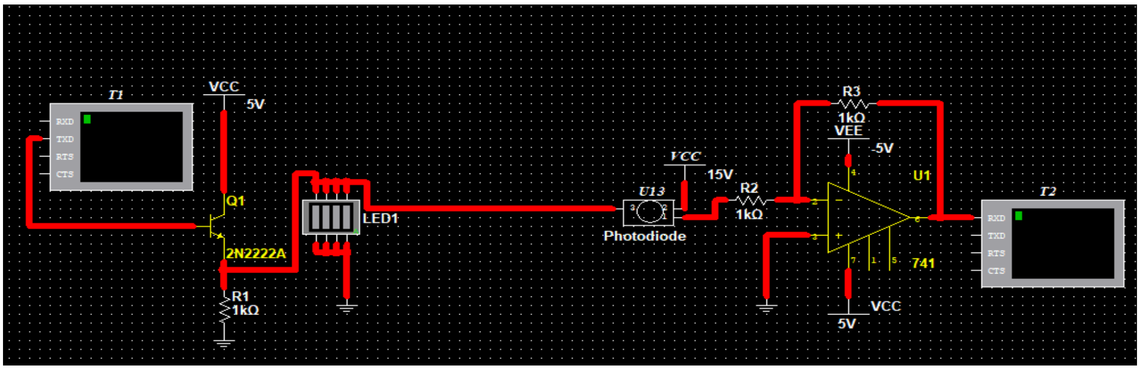

Fig. 4 OOK-NRZ VLC System circuit schematic created in NI Multisim 11.0.

3.1 Results and discussion

Bit Rate

As highlighted earlier, the fundamental limit to any communication system's bit rate relies on the modulation scheme used. In the implementation, the bit rate was also limited by the operational amplifier's GBP. The relatively low GBP of the amplifier (1.2MHz) acted as a figure to play around with in order to get a desired bit rate. The following were the recorded results. Varying the distance between the transmitter and the receiver, essentially had no effect on bit rate but the signal at the receiver could get too weak to be detected beyond a certain range of centimetres. Changing the gain of the receiver resulted

Table 2: results of Bit Rate vs distance for various amplifier

Distance (cm)	Amplifier Gain	Bit Rate(bps)
5	2.5	480,000
15	5.0	240,000
25	10.0	120,000
50	28.0	42, 900
100	265.0	4, 530

in the detection of the signal. But changing the gain means changing the bit rate and, therefore, it can be concluded that the bit rate varied with distance as given in table 2.

It must be noted that the operational amplifier that was used was LM234N. It was used in this experiment due to its availability during the time of experimentation. Therefore, the use of operational amplifiers with high GDP can yield high data rates.

BER

The bit error rate was recorded by comparing bits sent and bits received. The various sets of bits sent and received are shown in the table3. From the table, the BER for VLC system is easily deduced to be approximately 0;

Table 3: Transmitted and received bits

Transmitted Bits	Received Bits
10000000	10000000
00000000	00000000
01000000	01000000
11000000	11000000
00100000	00100000
10100000	10100000
01100000	01100000
00010000	00010000
10010000	10010000
10000000	10000000
01000000	01000000
11000000	11000000
00100000	00100000
11100000	11100000
11100000	11100000

3.2 Challenges

A major challenge in the experimental setup was the utilization of poor choice of components. Another challenge which is technical was that the transmitter and receiver could not be easily synchronized but this was expected since OOK-NRZ has a poor synchronism property.

Finally, the experimental set up must be done in the absence of other artificial light sources. This is because the transmitter of the VLC system should be offering both communication and illumination. Therefore, the presence of a light source other than ambient light in the same area is not possible in reality.

3.3 Recommendations

Future experiments should be focused on other modulation schemes such as Color-Shift Keying that enhances data rates. Additionally, Bidirectional Visible light communications that employ OOK-NRZ or other modulation schemes in different ambient light intensity should be carried out.

4. Conclusions

In this paper, an introduction to what visible light communication is was highlighted. Various modulation schemes were also given. The three basic schemes were compared based on flicker rate, dimmability, system complexity, data rates and relative spectral efficiency. CSK was seen to have a good spectral efficiency and highest data rates. A proposal of a possible direction to enhance data rates based on CSK was outlined.

The proposed approach of enhancing data rate is possible all because previous works on this issue have aimed at using commercially available LED bulbs and Photo sensors for communication. However, these were designed for illumination only and as such, there is a need for designers of LED bulbs to accommodate communication in their designs. The photo sensors that are commercially available can easily be used in visible light communication as they are well developed for communication.

The implemented VLC system proved the feasibility of such systems. From this simple setup, it was shown from the BER that the system was not prone to the erroneous effects of extraneous signals.

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A Perspective of Sustainable Development in Africa through the use of Cleaner Fuel for Locomotive Engines.

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Abstract

This paper reviews railroads and attempts to justify the need for railroads in Zambia with a focus on alternative cleaner fuel for locomotives. The paper reviewed literature across the world and highlighted a good magnitude of benefits associated with railroads and narrowed the discussion with an experiment on alternative fuel. It was found that railroads are associated with positive economic benefits such as massive employment, increased and enhanced business activities in the supply chain, reduced transport costs for motorists, reduced land requirement for parking and road construction and speeds up urbanization. It follows therefore that Zambia attempts to plan for railroads given the benefits that can potentially accrue. A reduction of emissions of thermal NOx is observed by correlation with the adiabatic temperature when a bomb calorimeter is used to simulate the combustion environment in internal combustion engines. The combustion chamber becomes cleaner and colder when comparing adiabatic temperatures between the alternative n-butanol (30% by volume)-gasoline (70%) blends and pure gasoline fuel. It is therefore recommended that Zambia undertake to plan for railroad and actually implement the projects. Additionally, the use of alternative fuel proposed can add to sustainable operation of the railroads.

1.0 Introduction

This paper discusses the current situation of railroad infrastructure in Sub-Sahara Africa and Zambia in particular. It further narrows down the discussion to the use of alternative cleaner fuels for locomotives. The paper highlights the inadequacy of this much needed infrastructure and stresses the potential and benefits that would accrue if Zambia adopted such infrastructure and the alternative fuel. Use of alternative cleaner fuels, illustrate a sustainable approach to rail road advances. In this case "sustainable development" is used in the sense that, advances in the use of clean fuels concurrently with the rail road development is sustainable. This approach does not affect public health to the same extent as would the use of fossil fuels in the long term perspective.

2.0 An Overview of Rail Infrastructure in Sub-Sahara Africa (SSA)

Railway efficiency is an important topic worldwide for transportation ministers of fiscally strained governments and railway managers operating in competitive markets. On the one hand, railways are under pressure to keep costs low, often because of market pressures or because of the unavailability of public funds as a result of competing national priorities (OECD, 2013). This is because railway systems are viewed positively by citizens and policymakers around the world because of their impact on mobility, their potential to improve land use and development in urban centers. However, SSA's

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transport-related infrastructure is limited, generally in a poor condition, and operating below design capacity, impeding development. Cross-border corridor transport in most of SSA is costly, slow and unreliable. This exacerbates transport challenges for landlocked countries in SSA with export potential (De Bod, 2008). The main response to these challenges has been increasingly ambitious programmes of foreign aid (Easterly, 2008). With just a fraction of the population of Asia, SSA receives more foreign aid, both multilateral and bilateral, than any other region (Holmes et al., 2008). Since the 1940s, more than US$1 trillion has been sent to Africa. Yet, while the debate continues on the many approaches to addressing SSA's development challenges, the fact remains that for SSA to achieve the 7% GDP growth rates in addition to reduce poverty, around US$20 billion per year infrastructure investment requirements was needed, which amount to, twice as much as the region invested historically (World Bank, 2005). Providing reliable, effective and efficient freight transport infrastructure is a key component of this investment (Njini, 2010).

3.0 The Need for Railroad Infrastructure-A Literature Review

In 2007, almost 20 percent of World Bank lending was allocated to transportation infrastructure projects, a larger share than that of education, health and social services combined (World Bank 2007). These projects aim to reduce trade costs. In prominent models of international and interregional trade, reductions in trade costs will increase the level of real income in trading regions. Related body of theoretical work argues that trade cost reductions can change the volatility of real income. This is a second welfare effect that may be especially important in predominantly agricultural, low-income economies.

The UK Rail Delivery Group reported in 2014 that the rail industry and its supply chain employ 212,000 people, contributing £9.3bn of GVA per year to the UK economy, and £3.9bn in tax revenue. The user benefits of rail for passengers and freight are between £1.3bn and £13bn per year. The benefits of reduced congestion also accrue to people travelling for leisure or commuting, the value of which is calculated to be up to £7.3bn per year. The rail sector delivers substantial environmental benefits, including between 0.7m and 7.4m tonnes of CO_2 in reduced greenhouse gas emissions from users making journeys by rail rather than by car and lorry (valued at £40m– £430m). In addition, rail delivers significant social benefits, including a reduction in accidents of between 95 and 950 serious casualties and fatalities a year, valued at between £33m and £330m annually.

Bitzan, *et al*. (2002) analyzed short railroad lines and found that in terms of preventing railroad abandonment, short lines provide reduced transportation costs to shippers, increased local business volume, reduced highway maintenance costs, decreased highway user costs, and increased economic development opportunities. In addition, short lines often provide improved service to shippers. Further, in examining the role of short lines in meeting the statutory responsibilities related to safety, there are substantial safety benefits from short-line rail operation in comparison to truck hauls, and that, for rail lines with moderate traffic levels and lengths of haul, continued short-line operation leads to fuel efficiency gains when the alternative is truck transportation. However, when lengths of haul and/or traffic levels are extremely low, continued short-line operation may lead to fuel efficiency losses when the alternative is truck transportation.

4.0 Operating an efficient Rail System with Cleaner Fuels

The Association of American Railroads (2011) reported that railroads are the most environmentally sound way to move freight. On average, trains are four times more fuel efficient than trucks. They also reduce highway gridlock, lower greenhouse gas emissions, and reduce pollution.

An efficient railway from a national perspective (including freight and passenger railways) maximizes revenues and minimizes costs while providing the desired level of service. A generic model of Swier's chart was derived to provide an overview of the relationship between costs, revenues and public subsidies for railway systems in general. The x-axis represent railway revenues/costs while the y-axis represents the track density.

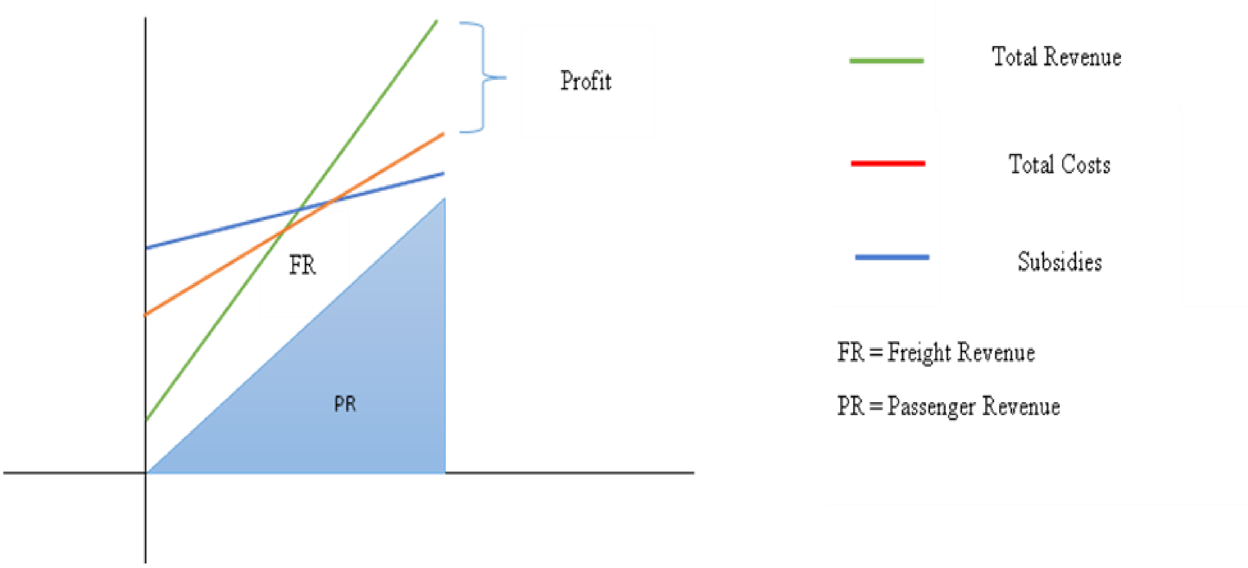

Figure 1: Profit determinants of rail systems
Source: Developed from Swier (2012) model.

The figure 1 above assumes that revenues are linearly related to track density which may not be so in reality, however, freight revenues may yield more and qualify the revenue/density linearity estimate.

Litman (2015) undertook a study in the US in which he concluded that rail transit does have significant costs. Rail transit requires about \$12.5 billion annually in public subsidy, which averages about \$90 additional dollars annually per rail transit city resident compared with Bus Only cities. However, these extra costs are offset several times over by economic benefits, including \$19.4 billion in congestion costs savings, \$8.0 billion in roadway cost savings, \$12.1 billion in parking cost savings, \$22.6 billion in consumer cost saving, and \$50 billion in reduced crash damages. From a household's perspective, rail transit provides direct transportation cost savings to an average of about \$450 annually per capita. Rail transit tends to increase regional employment, business activity and productivity. It can contribute to urban redevelopment. Property values increase near rail stations. Quality transit improves mobility for non-drivers, reduces chauffeuring responsibilities for drivers, improves community livability and improves public health. Section 6 deals with a study of the positive contribution that a blend of gasoline with alcohols fired in an internal combustion engine imposes on the environment.

Rossetti (2006) in a study on effects of weather on railroads in the United States of America found that rail roads are affected by a number of weather events such as high waters from flash floods, river floods, persistent heavy rains, and hurricanes have historically been one of the most prominent weather-related concerns facing the railroad industry..

5.0 Sustainable Railroads: An Opportunity for Africa and Zambia

The African Development Bank (AfDB) reported in 2015 that with a few exceptions (mainly in the RSA and Northern Africa), African railways clearly lag behind those of most other regions in the world. Rail transport has faced the same constraints and challenges as elsewhere. But, poor economic, technological and institutional conditions have further aggravated the situation in Africa. The result is outdated infrastructure, sometimes approaching a point of no return. The operations are clearly below international standards.

There are opportunities for railway development in Africa as a consequence of the following drivers postulated in Table 8 below:

Table 8: Opportunities for Africa to invest in rail infrastructure

a)	Growing urbanization and industrialization will pose new transportation challenges that railways are well suited to handle.
b)	Africa will produce large volumes of goods such as bulk minerals and commodities that are natural markets for railways.
c)	The huge continental mass of Africa and the existence of many landlocked countries will encourage the development of high-capacity and efficient transport corridors.
d)	Higher sensitivity towards environmental and safety issues will result in railways getting more public attention and social support.
e)	The reduction of the extremely high external costs (noise, pollution, congestion, accidents etc.) associated with the constant increase in the use and ownership of private vehicles.

Source: African Development Bank, 2015, Rail Infrastructure in Africa-Financing Policy Options

There are other opportunities for railways development in Africa because the growing urbanization and industrialization will pose new transportation challenges that railways are well suited to handle. Africa will produce large volumes of goods such as bulk minerals and commodities that are natural markets for railways. The huge continental mass of Africa and the existence of many landlocked countries will require the development of high-capacity and efficient transport corridors. Figure 3 shows the trans-Africa network. Greater awareness of environmental and safety issues will create a climate in which railways will get more public attention and social support. Railways may play a relevant role in the reduction of extremely high external costs derived from the use of road transport, in a context of constant growth in road vehicles in Africa.

Accordingly the AfDB (2015) reported that the areas deemed to be most appropriate to railway projects in Africa are

a) Major African metropolises Areas > Urban and suburban passenger railways.
b) Densely populated areas and corridors > High volumes for freight or passengers possible.
c) Corridors from ports to inland markets > Freight trains moving containerized or bulk materials from/to ports over long distances.
d) Major mining basins > Freight trains moving minerals and other raw materials to export ports.

Zambia has a total gazette Road Network of 67,671km with a core road network of about 40,454km necessary for economic development. This network is not adequate to facilitate good trade in the country and with its neighbors. Zambia has been experiencing a growing number of fatal road accidents involving passenger service vehicles a phenomenon that is insignificant with rail transport. This observation leads to the introduction of a statutory instrument that banned night travel for public passenger service. This led to evolving the Zambian economy to a sleeping economy which can only be countered by air and rail transport. It is against this background that greater potential remains with an establishment of a comprehensive railroad network to cover rural, per-urban and urban cities. A regional network is shown in figure 3 and Zambia ring network is potentially possible as indicated in figure 4

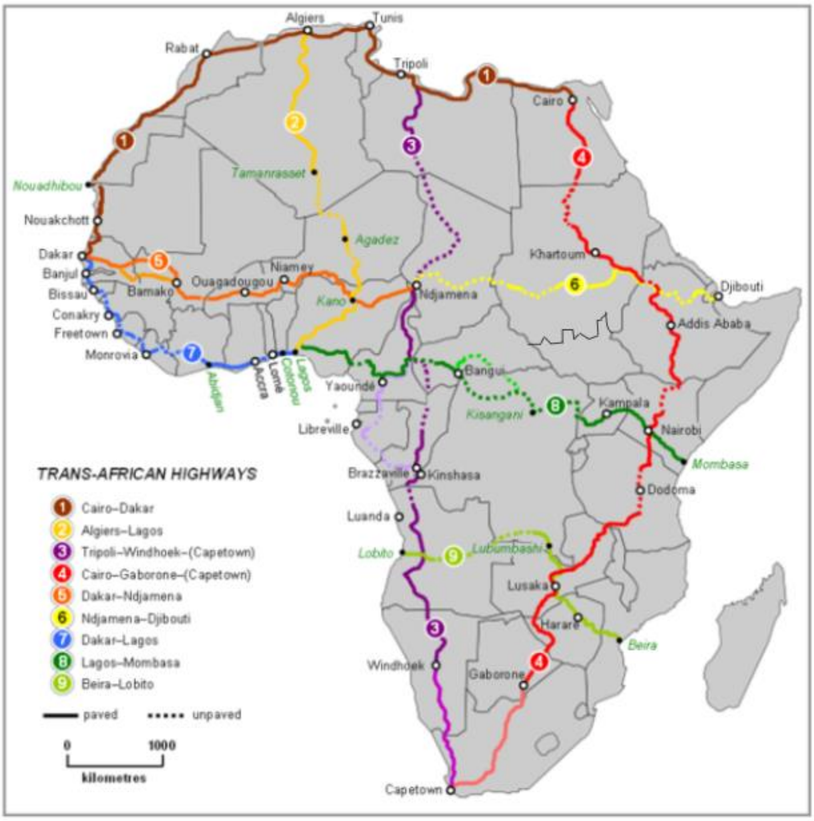

Figure 3. Trans-African highways
(Source:https://commons.wikimedia.org/wiki/File:Map_of_Trans-African_Highways.PNG)

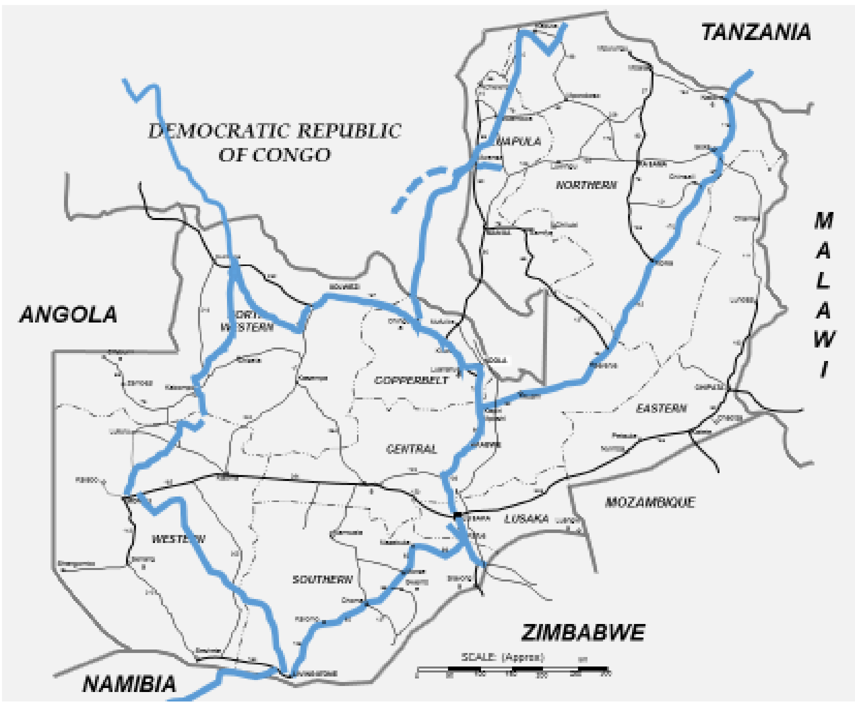

Figure 3: Potential railroad ring for Zambia.
Source: Developed from Zambian maps

This network if built would reach many rural places currently not covered by the road network yet accommodates a considerable number of farms and other grain producing areas. Further, it would add to the efforts to link to other African countries by rail.

6.0 A Case for Cleaner Railroad Locomotive Engine Fuel

The use of petroleum fuels in transportation, greatly contributes to the deterioration of the environment through the emission of pollutant substances. These emissions are usually incomplete combustion products such as nitrogen oxides (NOx), unburned hydrocarbon (UHC), carbon monoxide (CO), particulate matter (PM) and carbon dioxide (CO2). Consequently, the UN Climate Change Conference in 2014 set targets to achieve carbon emissions as follows:

a) Reduction in specific fuel energy consumption from train operations:

50% reduction by 2030 (relative to a 1990 baseline) 60% reductions by 2050 (relative to a 1990 baseline)

b) Reduction in specific average CO2 emissions from train operations: 50% reduction by 2030 (relative to a 1990 baseline) 75% reduction by 2050 (relative to a 1990 baseline

The use of alternative fuels to normal gasoline is a scientific research area that has been ongoing from the time scientists agreed that global warming and climate change were anthropogenic causes and hence would be a response to the climate change targets. The past research studies show that researchers have focused mainly on combustion and emission characteristics of single alcohol blends in an engine. The combustion characteristics of dual alcohol gasoline blends however, have few research case studies (Siwale, *et al*., 2012). It is against this background that the researchers undertook to research on the possibility of using alternative fuels for engines.

Hence, in this research, the combustion characteristics of the dual blends of methanol, butanol and gasoline were tested using a bomb calorimeter through the evaluation of the adiabatic temperature. The emission results obtained in an internal combustion engine are also reported in the author's past studies (Siwale *et al*, 2016).

Pukalskas *et al*. (2009) conducted engine performance tests and found out that the blend shared volume to gasoline fuel (GF) increases with the excess-air ratio (λ), or A/F ratio (based on mass fuel flow) but reduces as the speed is increased. This can be attributed to a better combustion quality at high loads than at a lower speed. This is illustrated in Table 1.0 for (n-butanol 30% by volume, gasoline 70%:Bu30) and Bu50. The slightly lower heating value of butanol increases the fuel-mass flow and λ value is leaner than that of GF.

Table 1: Operating conditions of n-butanol fuel blends

Blend	Speed (RPM)	Torque (Nm)	A/F air/fuel (lambda) ratio
Bu30	2500	7-21	1.1 -1.16
Bu50	2500	7-21	1.25-1.3
Bu30	3000	15-40	1.1-1.15
Bu50	3000	15-40	1.19-1.25
Bu30	4000	21-64	1.04-1.01
Bu50	4000	21-64	1.12-1.17

Source: (Pukalskas *et al*., 2009)

Further experiments conducted by Pukalskas *et al.*, (2009), showed that the carbon monoxide, emission is higher for Gasoline Fuel (GF) than for Bu30 and Bu50 for the entire load range although the CO when using GF is only slightly higher than that for the blends at 4000 RPM. Carbon dioxide is higher when using GF for the entire range than that for Bu30's and Bu50's. CO is expected to be reduced for oxygenate fuels than for GF on higher loads and temperatures. GF has a greater carbon to the hydrogen ratio (C/H) than oxygenate fuels and hence CO_2 is expected to be higher in high loads for GF than for the biofuels. The emission of UHC is highest when using Bu50 followed by GF and Bu30. In addition, Szwaja and Naber (2010), found that the ignition delay for n-butanol blends decreases with the increase of the alcohol fraction (B20 and B60) in experiments that were conducted on a single-cylinder engine.

6.1 Objectives

The objective of this research is to reduce the negative impacts of petroleum oil based fuels in reciprocating engines on the environment through the use of oxygenated (alcohol) blends, while not deteriorating engine performance. The focus of the paper is to determine the lower heating value, LHV of selected dual alcohol-gasoline blends; n-butanol-methanol-gasoline blends using a bomb calorimeter. This is followed by comparing the theoretical LHV of the blends with that obtained from the bomb calorimeter and simulating and estimating the adiabatic temperature under engine conditions using data from the bomb calorimeter.

6.2 Methodology

a) The adiabatic, temperature was evaluated in the bomb calorimeter at constant volume and at constant pressure in the engine.

b) The Lower heating value (LHV) was calculated by subtracting the water content from Higher Heating Value (HHV) as determined by equation (3).

c) The theoretical LHV was compared with the LHV calculated from the bomb calorimeter.

d) The blends, which met the vapor pressure requirement for gasoline fuel as proposed by Siwale *et al*. (2012) were tested.

6.3 Theoretical Perspective

The Low Heating Voltage (LHV) of the blend is given by equation:

$$LHV_{bl} = \sum_{1}^{2} \frac{vi}{v} * \rho_i * LHV_i / \sum_{i=1}^{2} \frac{vi}{v} * \rho_i \tag{1}$$

Where LHV_{bl} is the lower heating value (MJ/kg) of the blend; LHV_i is the lower heating value of component, i, of the blend, i=1 is n-butanol, i=2 is gasoline.

Density of a blend is calculated from the following equation:

$$\rho = \sum_{i=1}^{2} \frac{vi}{v} * \rho_i \tag{2}$$

Where ρ is the density of the blend (kg/m^3) and v_i/v is the volume fraction of component i of the blend.

6.3.1 Lower Heating Value (LHV) Determination

Balancing Of Bomb Calorimeter Combustion Equations

Example M10:B20 BLEND EQUATION

The mass was calculated by: Mass of each blend component= V×% purity ×ρ

Where;

i. V = volume of each blend component (ml)
ii. % purity = %purity of each blend component (%)
iii. ρ = density of each blend component (g/ml)

The number of moles was calculated by; Number of moles = $\frac{M}{Mr}$

Where;

i. M = Mass of each blend component (g)
ii. Mr = Molecular mass of the blend component (g/mole)

The total mass of fuel blend was calculated by;

M_T = sum of all blend components present in fuel

Where, M_T = total mass of fuel (g)

Table 3 indicates the value of the coefficients. The general combustion equation in the bomb calorimeter is as follows:

$$aC_8H_{18\,(l)} + bCH_3OH_{(l)} + c\,CH_3(CH_2)_3OH_{(l)} + dO_{2\,(g)} \rightarrow eCO_{2\,(g)} + f\,H_2O_{(l)}$$

Using the sample M10:B20, the value of coefficients a, b, c , d, e & f (number of moles), had to be calculated as an example

The balanced equation for M10:B20, after substituting coefficients is:

$$0.45C_8H_{18\,(l)} + 0.25\,CH_3OH_{(l)} + 0.22CH_3(CH_2)_3OH_{(l)} + 7.32O_{2\,(g)} \rightarrow 4.73CO_{2\,(g)} + 5.65\,H_2O_{(g)}$$

Coefficients determined are indicated in Table 2 and important parameters like density and molecular weight of each component in the blend are as shown in Table 3

Table 2: Combustion coefficients for reactants and products

a	b	c	d	e	f
0.45	0.25	0.22	7.32	4.73	5.65

Table 3: Properties of density and molecular weight for the fuels

Name	Density (g/ml)	% Purity	Molecular Weight (g)
Methanol	0.79	99.5	32.04
Butanol	0.81	99.5	74.12
En228 gasoline	0.74	-	114

The procedure used in the example for M10:B20 to determine the coefficients in the combustion equation was used for all the tested blends. The Bomb calorimeter mass calculation for each blend is given in Table 4 based on the volume fraction of the blend and its density according to equation 1.

Table 4: Mass of fuel used in the bomb calorimeter

BLEND	TOTAL FUEL MASS(Kg)	BLEND	TOTAL FUEL MASS(Kg)
M10:B20	0.0757795	M80:B20	0.079003
M15:B25	0.0763395	M0:B0	0.074
M20:B30	0.0768995	M70:B0	0.0772235
M25:B35	0.0774595	M53:B17	0.077561
M30:B20	0.0767005	M20:B0	0.074912
M40:B20	0.077161		

Engine Balanced Combustion Equation

With reference to the stoichiometric combustion assumption, the only difference between the bomb calorimeter and engine equations is the N_2 gas term, since an engine uses air as oxidizer.

Therefore the general Hewood (1988) combustion equation is written down below:

$aC_8H_{18\,(l)} + b\,CH_3OH_{(l)} + c\,CH_3(CH_2)_3OH_{(l)} + dO_{2\,(g)} + 3.76\,(d)\,N2_{(g)} \rightarrow eCO_{2\,(g)} + f\,H_2O_{(l)} + 3.76\,(d)\,N2_{(g)}$

$\Sigma n\Delta H_{fi(reactants))}$-($\Sigma n\,(\Delta H_{fi} + C_{pi}\,(T_{ad} - 298k))_{(products)}$- $\Sigma R(n_{reactants}298 - n_{products}\,T_{ad}$))= 0

Where,

n = number of moles for each individual component (mol)

ΔH_{fi} = standard enthalpy of formation at 25◦C (KJ/mol)

C_{pi} = average specific heat capacity (between the assumed final temperature of 2100k and 298K) @1200k (KJ/mol K)

T_{ad} = Adiabatic flame temperature (K)

R= 8.314×10^{-3}KJ/kmol (universal gas constant)

Heating Value (HHV) Calculation

Three sample trials of each blend were tested on each volume ration fuel blend. The higher heating value,(HHV or CV) for each sample trial was calculated by equation (3) as shown in Table 5

$CV_{fuel} = [M \times C_p \times T_4]_{water} - [M \times Cv]_{wire} / 1000[M]_f$ (3)

Where,

i. CV_{fuel} = higher heating value of sample trial (MJ/Kg).
ii. M_{water} = mass of cooling water (1.4kg).
iii. $C_{p\ water}$ = specific heating capacity of water (4.186 KJ/Kg ◦c).
iv. $T_{4\ water}$ = (final – initial) temperature (◦c).
v. M_{wire} = mass of used nichrome wire (initial nichrome wire mass – mass of nichrome wire remains)(Kg).
vi. C_V for wire = 1600 KJ/Kg.
vii. M_{f} =Mass of fuel sample trial blend(Kg)

Table 5: Example values are indicated for HHV.’

Blend Name	Average HHV(Mj/Kg)
M10:B20	38.32
M15:B25	37.45
M20:B30	35.63
M25:B35	34.2
M30:B20	35.31
M40:B20	33.51
M80:B20	26.16
M0:B0	45.3
M70:B0	28.03
M53:B17	31.67
M20:B0	41.54

Lower Heating Value from (HHV)

The lower heating value is calculated, Heywood. B (1988):

$$LHV = (HHV - 2.256\ (M_{water})/m_f) \qquad (4)$$

Where

i. LHV= Lower heating value(MJ/Kg)
ii. HHV= Average higher heating value(MJ/Kg)
iii. M_f =mass of fuel (Kg)
iv. 2.256= latent heat of vaporization of water at 25ºC (MJ/Kg)
v. M_{water} = mass of water (Kg) = No of moles of water (using the balanced combustion equation)× molecular mass of water.

6.4 Procedure for the determination of LHV from Bomb Calorimeter

Firstly 1g of the liquid fuel blend was weighed using the analytical balance.

Then the liquid fuel sample was weighed in a crucible and the nichrome wire was immersed in the liquid fuel sample. The oxygen delivery tube was attached from the oxygen cylinder to the bomb inlet and oxygen was allowed to pass slowly into the bomb. The bomb contents were transferred into the calorimeter with measured water of about 1400mls.The electrical terminals were connected to the bomb calorimeter. The stirrer was turned on to stabilize the temperature of the cooling water for about one (1) minute. After temperature stabilization, the finger was placed on the firing switch until the red signal showed. The initial temperature was noted down just after firing the bomb circuit. The temperature every after one (1) minute was recorded during the combustion process. The final temperature was recorded and the temperature drop recorded for at least 4 minutes. The apparatus was disconnected, and the excess pressure was released. The remains of the nichrome wire were weighed using an analytical balance.

6.4.1 Research Materials

Butan-1-ol of 99.5% purity, molecular weight of 74.12 g/mol and chemical formula, $CH_3.(CH_2)3.OH$ was used. Methanol of 99.5 % purity, 32.04g/mol molecular weight and chemical formula CH_3OH, to be used. Gasoline type EN228 of molecular weight: 114g/mol, and chemical formula, C_8H_{18}.

a) Tap water.

a. Pure Oxygen gas

6.4.2 Research Apparatus

The parr bomb calorimeter consisting of chromium steel cylindrical container with a lid, a cage, a crucible, a nichrome wire, gas cylinder, & calorimeter. The others were spatula, measuring cylinder, precision thermometer, stirring assembly, clamp and stand, power supply, bench-support,

Figure 2: bomb calorimeter apparatus

Table 6 Adiabatic Temperature results (Tad)

	Engine T_{ad}(K) constant pressure	Bomb Calorimeter T_{ad} (K) constant volume
	2378	7334
	2373	7265
	2360	7142
	2355	7039
	2357	7067
	2349	6892
	2398	7600
	2306	6483
	2322	6685
	2380	7328

Table 7 Lower heating value results

Blend Name	Bomb Calorimeter LHV (MJ/kg)	Literature LHV(MJ/Kg)	% Error
M10:B20	35.29	38.79	9.92
M15:B25	34.47	37.22	8.00
M20:B30	32.72	35.64	10.40
M25:B35	31.33	34.08	8.80
M30:B20	32.37	34.45	6.42
M40:B20	30.67	32.38	5.60
M80:B20	23.38	24.37	4.23
M0:B0	42.09	44.40	5.50
M70:B0	25.30	27.20	7.51
M53:B17	28.87	30.50	5.65
M20:B0	38.50	40.50	5.20

7.0 Results and Discussion

Cleaner options are now abundant, for instance, the total length of electrified route has slowly increased since the start of the time series in Britain. In 2013-14 5,268 kilometres of passenger and freight route was electrified, an increase of 0.1% on 2012-13 and 38.3% increase since 1985-86. Passenger train CO_2 emissions per passenger kilometre in 2013-14 were 44.8 grams, a 4.9% decrease on the 2012-13 value. Freight train CO_2 emissions per freight tonne kilometres in 2013-14 were 25.9 grams, an 8.1% decrease on 2012-13 (British National Statistics, 2014).

Many operators have installed energy efficient lighting at stations and in their offices, which is triggered by motion sensors. In the case of South West Trains in the UK, the fitting of energy efficient lighting has reduced station energy consumption by up to 40%. Merseyrail is harvesting rainwater and recycling it to use in lavatories and sinks. Many new or refurbished stations are using technologies such as photovoltaic cells on windows to harness solar power to provide a proportion of the electricity for the station.

In Table 6, the adiabatic temperatures from the bomb Calorimeter and simulated results in the engine are indicated. The higher adiabatic temperatures are due to the oxidizer used being pure oxygen in the bomb calorimeter. In the engine, nitrogen was included since air is burned in an engine. The adiabatic temperature values decreased due to the high latent heat of vaporization of alcohols causing evaporative cooling inside the combustion chamber. The adiabatic temperature reduces moderately with the increase of alcohol components in the fuel as alluded to as a result of the presence of the hydroxyl group components attached to alcohols. Therefore, a reduction of emissions of thermal NOx is observed by correlation with the reduction of the adiabatic temperature. Alcohols provide some of the oxygen molecules required for combustion internally within the fuel itself. The lower heating value indicated in Table 7 due to the blends of alcohol in gasoline, causes the burning temperature and the power output to reduce. However, this is compensated by increases of a high CH alcohol into the blend, in this case n-butanol.

8.0 Conclusions and Recommendations

This literature review was aimed at justifying the need for railroad by reviewing literature associated with economic benefits of the infrastructure. Literature indicates a great potential to achieve enormous benefits from railroad infrastructure. The economic benefits include access to far off rural areas hence allowing market access to farm produce from such areas, easy connectivity of districts and associated economic improvement in terms of employment, supply chain creation, enhanced business activities. In terms of urban centres, such infrastructure brings about decongestion and alternative transport options to motorists, reduced parking requirements, reduced road traffic accidents due to reduced vehicles on the roads and reduced emission or green-house gases. Further, road maintenance and rehabilitation requirements would reduce due to reduced tonnage carriage on the roads.

The experiment on alternative fuels revealed that blends which met the vapor pressure requirement for gasoline fuel as proposed by Siwale, *et al*. (2012) were tested to determine the combustion characteristics of oxygenated blends with gasoline in an engine. The Bomb calorimeter adiabatic temperature was used to simulate the adiabatic burning conditions of an engine. The temperature in the bomb calorimeter was higher than what was simulated for the engine. The differences between the two is due to the fact that combustion in an engine is not adiabatic and nitrogen is an additional component (air is the oxidant) where pure oxygen is the oxidant in bomb calorimeter experiments. The key indirect observation to this study is a confirmation of a reduction of emissions such as thermal NOx as the combustion environment in engines becomes cleaner and colder when the alcohol gasoline blends are used rather than pure gasoline.

It can therefore be recommended that Zambia undertake to plan for railroad and actually implement the projects. Additionally, the use of alternative fuel proposed can add to sustainable operation of the railroads.

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Electric and Hydraulic Power Utilisation in Modern Commercial Aircraft Primary Flight Control Actuators

Edson Lungomesha[1] and Ackim Zulu[2]

Abstract

This paper presents an overview of primary flight control actuator technologies used for safety-critical aircraft applications, with a particular focus placed on comparison between conventional hydraulic and the modern electric actuators being introduced on modern aircraft. Aircraft applications demand high reliability, high availability, and high power density while aiming to reduce weight, complexity, fuel consumption, operational costs, and environmental impact. The justification for incremental but cautious technological substitution of primary flight control actuators in commercial aircraft is due to the flight safety concerns. The comparatively simplified circuits of electric actuator technology make such system easier to maintain and inspect. Futher, the replacement of piping circuits with electric cables reduces the weight added to the aircraft resulting in lower fuel consumption. Therefore the new electric driven actuators provide significant technical and economic improvements over conventional mechanical, hydraulic, or pneumatic actuators. In addition the aircraft health monitoring system that can be incorporated in electric actuators helps in early detection of potential aircraft snags before they occur. This would consequently lower both maintenance time and cost. The reduced maintenance time would result in increased aircraft operational availability. The analysis suggests that the electric actuators are economical, safe and reliable and are gradually replacing the conventional hydraulic actuators in modern aircraft.

Keywords: Primary Flight control actuator, reliability, modern aircraft, mechanical, hydraulic, pneumatic actuators.

1. Introduction

Actuators are key elements of air and spacecraft. In the recent years, the concept of the more-electric aircraft pushed the development of electric actuation systems to substitute hitherto used hydraulic actuators in a broad range of applications such as flight control, landing gear and brake

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actuation (Jänker et al., 2008).

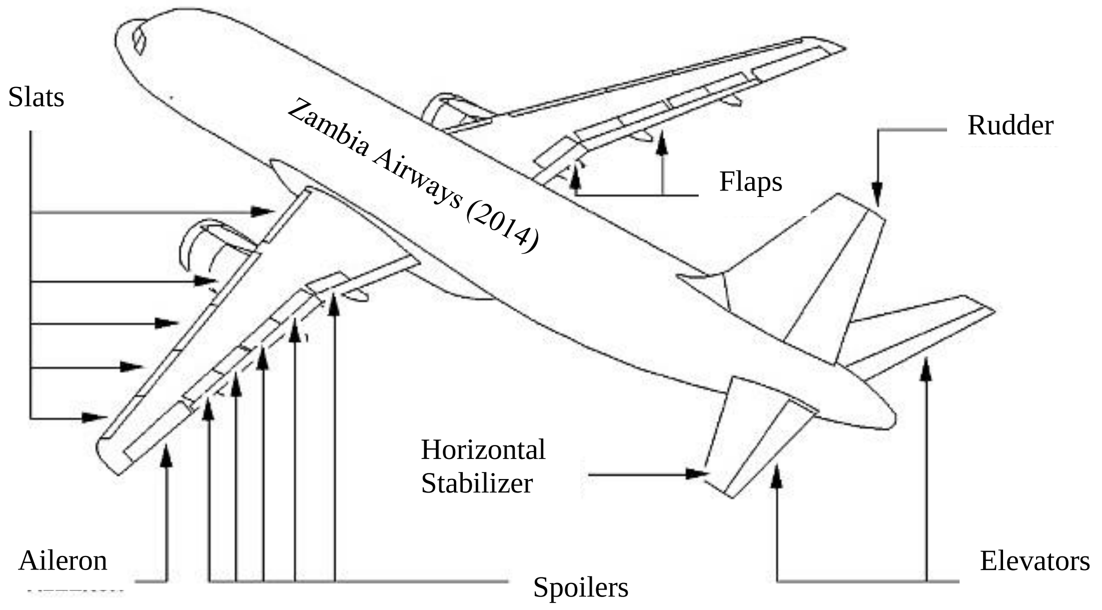

Fig. 1: Aircraft flight control surfaces

Hydraulic actuators are used to move the control surfaces in order to control the aircraft. There are three critical degrees of flight control; these are the controls for the roll, pitch, and yaw of the plane. The two types of flight control surfaces are the primary and secondary control surfaces shown in Fig.1. The primary control surfaces are the flight critical control surfaces and they include the rudder, ailerons, and elevators. Actuators that move these surfaces are referred to as primary flight control actuators. The secondary control surfaces are not critical to the flight as the aircraft can be flown without them; they are used for comfort and efficiency of flight and include control surfaces, such as flaps and slats. Therefore, the actuators that control these surfaces are referred to as secondary actuators (Wheeler, 2016). In conventional wide-body aircraft, the actuation system of the flight control surfaces is done by a centralized hydraulic system, constituted by a hydraulic pump and hydraulic motor drives positioned in the fuselage plus several fluid pipelines and hydraulic actuators positioned in the wings and tail surfaces (Bogliettin et al.,2009).

Mechanical or manually operated flight control actuators are the most basic method of controlling an aircraft. They were used in early aircraft and are currently used in small aircraft where the aerodynamic forces are not excessive. Very early aircraft, such as the Wright Flyer I, Blériot XI and Fokker Eindecker used a system of wing warping where no conventionally hinged control surfaces were used on the wing, and sometimes not even for pitch control as on the Wright Flyer I and original versions of the 1909 Etrich Taube, which only had a hinged/pivoting rudder in addition to the warping-operated pitch and roll controls (Wikipedia, 2019). However hydraulic actuators are inefficient due to high hydraulic loss and high maintenances cost. In addition, they are always active even when not in use (Behbahani, 2018).

2. Problem Statement

The commercial aircraft operational costs include both the direct operating costs (fuel consumption, etc.) and the indirect costs linked to maintenance and availability. Research has shown that 75% of late flights are due to system faults. Although electrical components may not offer improved reliability, they increase availability due to possibility of isolating a subsystem in case of failure. On the other hand, a hydraulic leakage in a device may lead to the isolation of the entire circuit, which provokes a “No Go” fault. The failure of a hydraulic circuit is equivalent to an electrical bus bar failure, whose probability is 100 times less. With modern advancement in technology, it may even become possible to anticipate faults in the near future through the health monitoring capacity. The use of reflectometry devices is making it possible to detect and localize cabling faults within 1 m of accuracy which is not possible in hydraulic systems (Roboam et al., 2012). Therefore the application of Hydraulic Actuator (HA) and Mechanical Actuater (MA) for controlling primary surfaces in commercial aircraft introduces numerous mechanical linkages and hydraulic power supply lines that result in increased linkages failure rates, complicates maintenance/inspection processes and undesirably increases the weight of an aircraft. Therefore, this study reviewed emerging electric and electronic-based actuator technologies that incrementally but cautiously are replacing HA and MA so as to reduce or eliminate the numerous unsafe linkages, simplify maintenance/inspection processes, ensure airworthiness integrity and significantly reduce aircraft weight.

The flaps and slat surfaces on the wings of the aircraft are used for lift purposes when taking off and landing. Generally, commercial aircraft use two hydraulic motors, mechanically summed via a shaft running the length of the wing span. The relative position of all flaps is monitored since their symmetry across both wings is critical to flight controllability. If a flap asymmetry occurs or a fuel pump fails, all the flaps are locked in position to prevent further instability. Therefore replacing these hydro-mechanical systems with electrical drives in the form of individual actuators at each flap surface can provide greater functionality and eliminate the need for centralized hydraulic pumps shafting, pipework, and other ancillaries, potentially improving system reliability, maintainability, and mass (Cao et al 2012).

3. Rationale

Actuators are critical engine and flight control components used in the aviation industry for motion and fuel controls. There are three types of primary actuators; electro-mechanical actuators (EMA), electro-hydraulic actuators (EHA), hydraulic actuators (HA). During aircraft powered ascent actuators control thrust vectoring of the main engines, movement of the aerodynamic control surfaces, and the positioning of propulsion system geometry and fuel/air control valves. EMAs comprise an electric motor and gear train to reduce speed, translate motion, and provide appropriate load torque while Electro-hydraulic actuators are self-contained systems that combine the merits of an electric system with those of the hydraulic systems. EHAs use an electric motor to drive a hydraulic pump which develops hydraulic pressure that acts on a cylinder to provide the mechanical actuation energy. On the other hand hydraulic actuators use a centralized hydraulic pump that supplies the required pressure. EHAs avoid the operability issues associated with a central hydraulic supply and distribution system. They also have weight and integration benefits. Aerospace actuation has historically been dominated by hydraulic and fluid power systems. Sales of hydraulic actuation systems today accounts for more than several billion dollars per year of business for the major vendors. These systems comprise about 19% of the cost of a commercial aircraft. However, as entrenched as hydraulics are in flight applications, the emergence and maturation of electrical actuation promises to encroach significantly on hydraulic technology over the next several decades. The EMA provides these benefits for applications where high-precision rapid actuation is desired. EMAs maintain the same high performance capability as hydraulic actuators and the potential for enhanced reliability and controllability in a compact package. The EMA will lend itself to high levels of diagnostics and fault prediction capability using algorithms in the engine/aircraft control system. Adaptive engines of the future will demand a variety of actuation solutions which may be a combination of EMAs, EHAs, and others. Their use requires careful system considerations to achieve optimal integration in the engine and air vehicle (Behbahani et al., 2010).Electromechanical actuators are gaining ground in the aerospace industry within the More Electric Aircraft (MEA) architecture. The aim of the All Electric Aircraft technology includes achievement of Electromechanical actuation for flight control systems (Papini et al., 2018).

There is a general intention in the aerospace industry to increase the amount of electrically powered equipment on future aircraft. This move is generally referred to as the“More Electric Aircraft” and brings with it a number of technical challenges that need to be addressed and overcome. High power, electric actuation systems are being proposed on many new aircraft with ratings up to 50kW. The role of actuators is to move flight control surfaces such as the rudder, aileron, spoiler etc. in order to control the speed and direction of the aircraft during flight (Trainer and Whitley, 2002).

Increasing demand performance and certification requirements by both the International Civil Aviation Organisation (ICAO) and local regulators for next generation aircraft flight control systems, together with a continuous development effort within the industry towards “all/more electric aircraft have fostered the need to replace traditional hydraulic actuating systems with electric and /or Electro-Hydraulic technology (Churn et al., 1998). Therefore aircraft flight quantities and success of the mission depend to a great extent upon the actuator performance, and design to achieve the specified criteria. Hence the reason why electromechanical flight actuators driven by electric motors have begun to displace hydraulic technology in advanced flight vehicles (Lyshevski, 1999).

4. Actuator Technology

4.1 Actuator Progression

Progression of actuator technology can be described using fig.2. According to the graph hydraulic actuator (HA) of fig.7 were installed on A320 and A340 aircraft between 1985 and 1995. The system is supported by piping network making it inefficient, difficult to maintain and has high risk of leakage. Conventional hydraulic actuators have been widely used due to their ability of generating high force, torque and high power to weight ratio (Rehman, 2016).

Higher reliability demands for modern aircraft cannot be achieved by conventional similar redundant actuation system. Fig. 2 shows the progression towards more electric aircraft (MEA) where hydraulic and mechanical actuators (fig.6 and 7) are gradually being replaced by electric actuators. In 2005 there was an introduction of servo-controlled actuators and 3 electrical backup hydraulic actuators (EBHA) shown in fig.8 incorporated in aircraft such as the Airbus A380. It has 8 spoiler panels which are controlled by 5 servo-control actuators and 3 electrical backup hydraulic actuators (EBHA) (Rehman 2016). From 2010 to 2020 Electromechanical actuators (EMA) have been applied in B787, A380 and A350. However, the probability of ball-screw jamming has not been completely eliminated in EMA systems. Therefore, an appropriate redundancy system should be put in place.

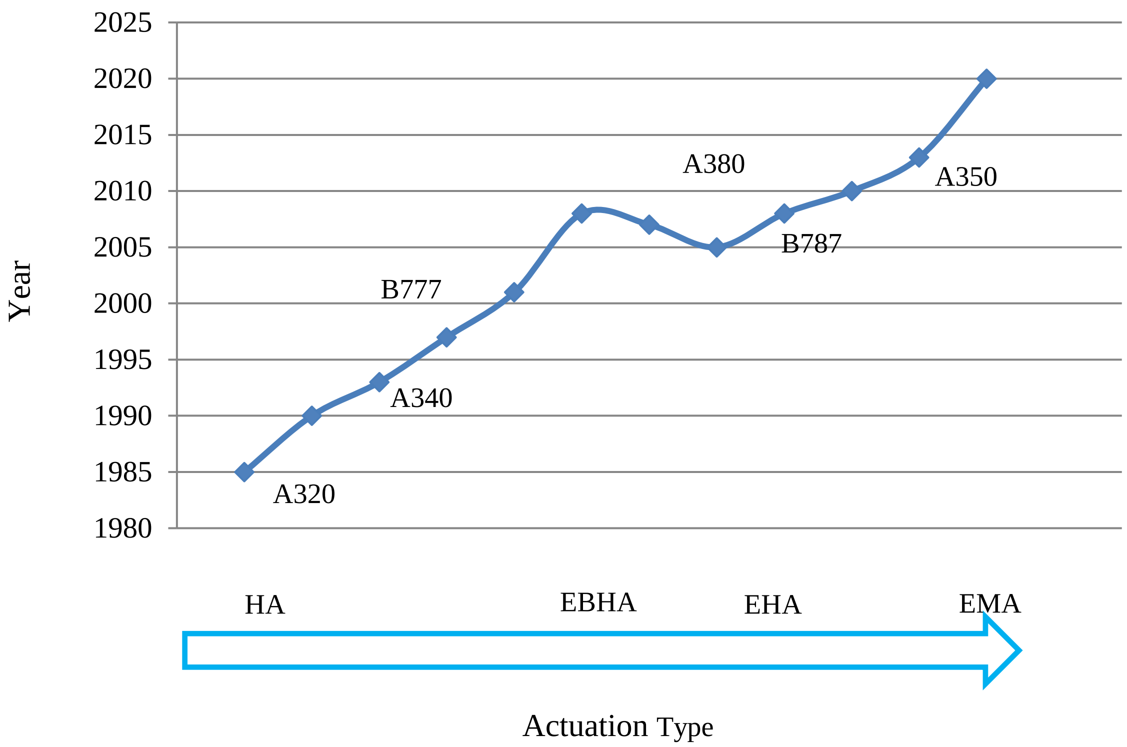

Fig 2: EMA introduction in aircraft flight control systems (Power source in the vertical axis on the left: M=Mechanical, H=Hydraulic,E=Electrical; Actuator type on the horizontal axis).

One of the reasons why the graph of fig.2 is wavy between the periods 2000 to 2005 rather than linear could be due to the occurrence of three air crashes within that period in which 369 passengers plus 6 people on ground lost their lives. The aircraft involved were McDonnell Douglas MD-83 on January 31, 2000. The other was the Beech BE-1900D on January 08, 2003 and the last one was the Airbus A-300-605R on November 12, 2001. The causes of the air crashes according to air crash investigators reports were due to malfunctioning of the flight control system. On the other hand only one air crash was recorded during the lower linear period 1990 to 1999. From 2010 to 2015 no related air crash was recorded.

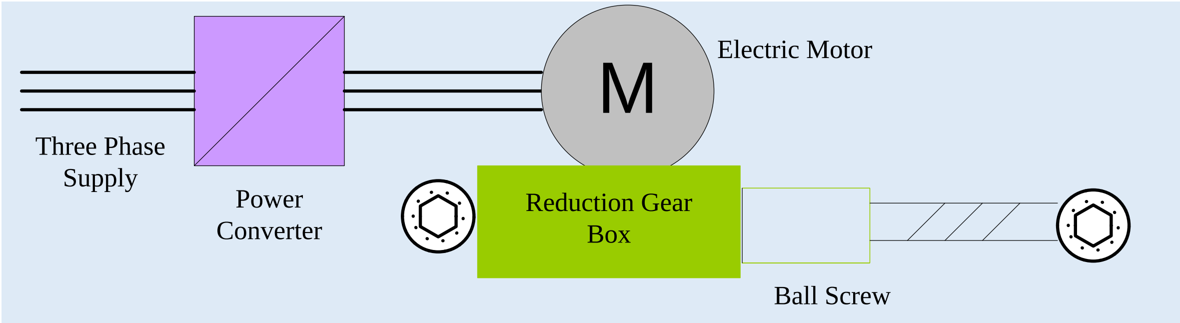

Fig. 3: A system diagram for an Electro-Mechanical Actuator (EMA)

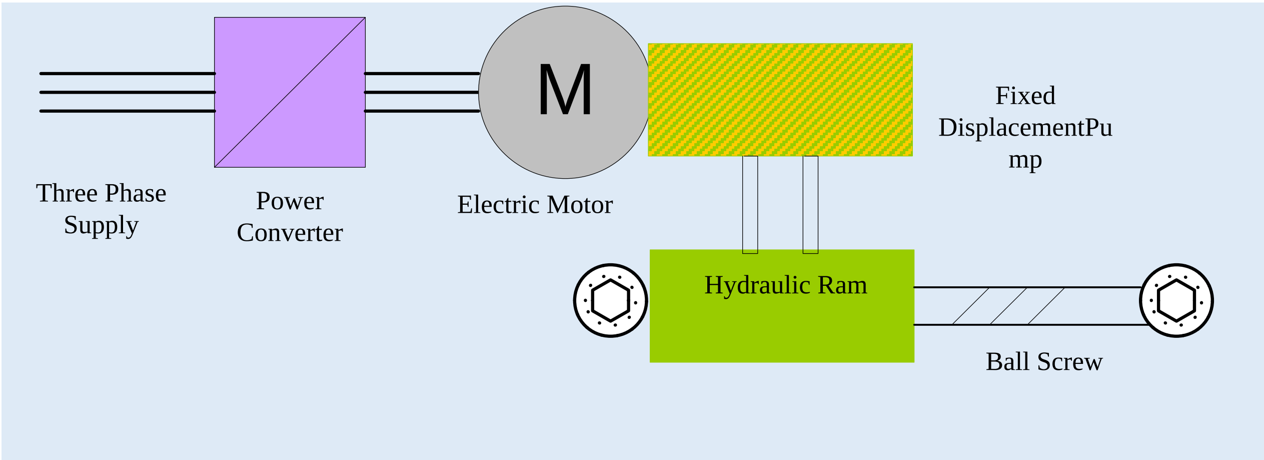

Fig. 4: A system diagram for an Electro-Hydraulic Actuator

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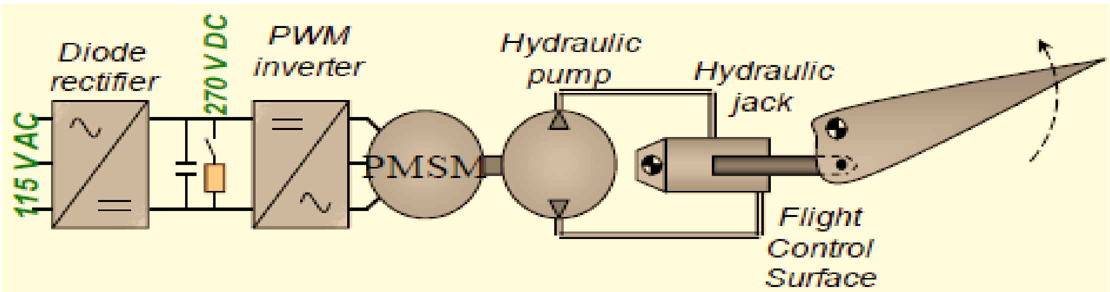

Fig. 5. An example of a power electronic fed actuator: A380

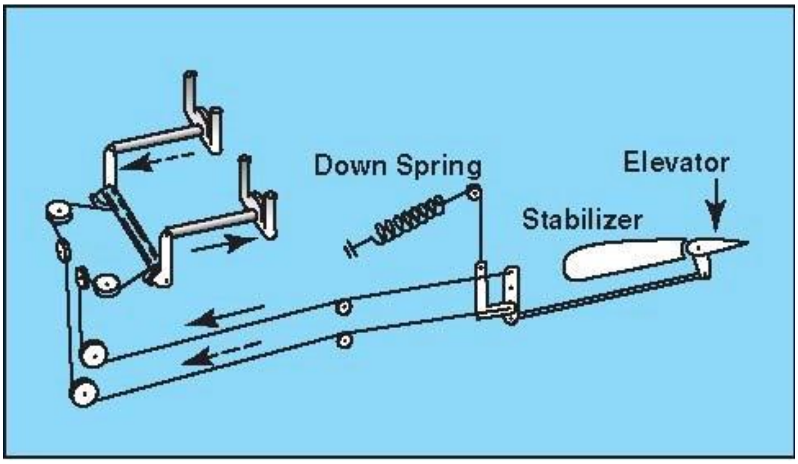

Fig. 6: Mechanical Actuator (MA) (Wikipedia, 2019)

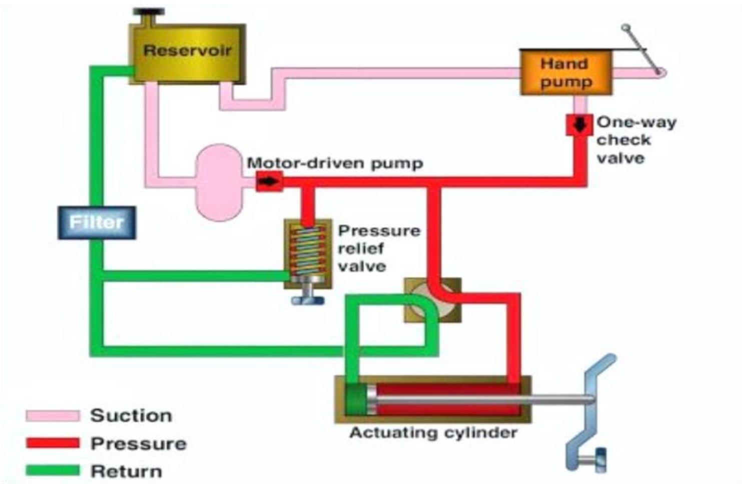

***Fig. 7: Hydraulic actuator (HA)* (Wikipedia,2019)**

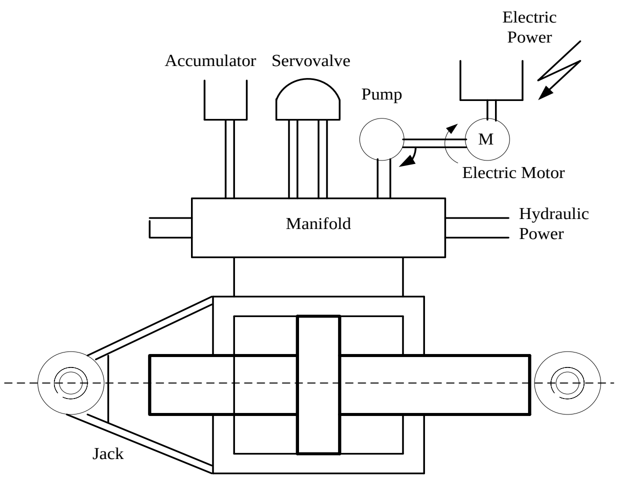

Fig.8. EBHA Electrical actuation principle for flight control system

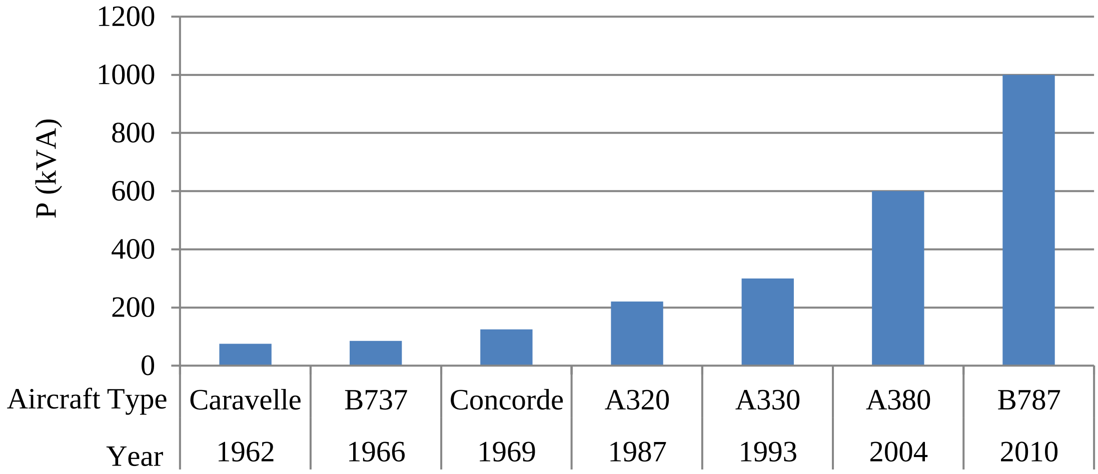

Fig.9: Evolution of electrical power needs

4.2 Actuator operation

The progression to more electric power has been moving correspondingly with gradual increase in commercial aircraft onboard power requirement of fig.9. In 1962 the Caravelle (using HA of fig.7) aircraft needed only 72kVA while in 1987 the Airbus A320 (also using HA) power requirement stood at 220kVA. The basic idea behind an EHA is to transfer electric energy into mechanical motion within one unit. Therefore, the typical elements of an EHA (Fig.4) are: the electric motor which transforms electric energy into rotational motion. The other is the hydraulic pump whose role is to transform rotational motion into hydraulic power. This is followed by the hydraulic actuator or hydraulic motor which transforms hydraulic power into motion. Finally there are the utility valves whose role is conditioning of the hydraulic circuit and mode switching e.g. active to damping (Gaile, 2016). The EHA, comprise a system driven by local hydraulics and controlled by a fixed displacement pump driven by an electrical motor. The actuator position moves by a fixed displacement for each revolution of the motor. There is no direct mechanical connection between motor and actuator arm, hence the EHA has benign failure modes, giving the system a significant advantage when compared to EMAs for primary fight control applications.

In an EMA system, the aircraft control surface is controlled by the control motor. The turning of the motor in Fig. 3 moves a ball-screw, often through a reduction gear box. As the motor is turning it displaces the actuator by a fixed amount because of the direct connection between the motor and ball screw. Nevertheless, one of the challenges of using EMAs for primary flight control applications on large aircraft is that as to date it has been very difficult to guarantee that the ball-screw will not jam. An actuator jam for a flight critical control surface would cause problems in the current design of aircraft as the surface would not be controllable unless a benign failure mode can be guaranteed. A jam in a ball-screw is not a benign failure as another actuator on the same control surface would not be able to move the surface if one actuator has jammed. Therefore when replacing hydraulic actuators with electrically powered actuators, the most obvious choice is to use an Electro-Mechanical Actuator [EMA], as shown in Fig. 4 (Wheeler, 2016). In EMA fig.3 the turning of the motor M through a reduction gearbox moves a ball screw connected to a control arm that moves the aircraft control surface. On the other hand in EHA fig.4 the Motor M through the fixed displacement pump and hydraulic ram to which a ball-screw and control arm are connected controls the aircraft controlled surface.

The All Electric Aircraft (AEA) concept deals with the distribution of electric power in the airframe electric system, replacing the existing range of secondary power distribution network. The engine system produces the secondary power in four forms: mechanical, hydraulic, pneumatic and electrical. The mechanical system supplies power to engine-mounted accessories such as oil, fuel and hydraulic pumps; and electric generators. The primary role of the hydraulic systems is to provide actuation of flight surfaces, landing gear and doors. The function of the pneumatic system is to derive pressure from a gas turbine off-take and provide heat and pressure for anti-ice protection, engine start and cabin environmental control. Electric power supply system provides power for: avionics, lighting and galleys etc. (Avery et al., 2007). For safety reasons a conservative approach has been taken in implementing new ideas and technologies in aircraft engineering. Nevertheless, there has been a trend in the aerospace industry to increase the proliferation of electric control and drives in aircraft in an incremental fashion. Mechanically driven actuators and hydraulic actuators are being replaced with electronic servo valve control “electro hydraulic actuation”. For instance in the Airbus A380 (fig.5) aircraft, electro-hydrostatic actuators provide hydraulic actuation from a localized pump and reservoir, allowing operation from an electrical power supply (Cao et al., 2012). Technological advances in the aircraft industry has improved aircraft efficiency and reduced the costs of air transport (Boglietti et al., 2009).

4.3. Comparison between Hydraulic, EMA and EHA

Electromechanical actuators (EMA) will become a key technology following the onset of power-by-wire flight control systems (FCS) in the next generation of commercial aircraft. Current technological advances in power drive electronics and permanent magnet synchronous machines have increased the power-to-weight ratio of EMAs making their introduction in the primary flight control systems (PFCS) of commercial aircraft more attractive. Research programs are contributing in raising the technology readiness level (TRL) of EMAs for such applications to TRL6. The next step will be to test full electromechanically-actuated active PFCS in flight. The use of EMAs gives rise to new types of faults and dynamics which need to be managed by robust control and monitoring functions. Due to the different physical characteristics of EMAs in comparison to state-of-the-art electro-hydraulic servo-actuators (EHSA), most control and monitoring functions need to be re-designed, verified, and validated. Only then, can the level of fault-tolerance required by airworthiness authorities be reached (Arriola, and Thielecke, 2015).

If the advantages, provided by the technology, fit with the needs of the application, EHA could be a very interesting technology. Nevertheless, they are not the best solution for every application. This means the technology is prime for safety critical application, such as on aircrafts, but could also be an interesting option for some specific industrial applications (Gaile, 2016). Table 1 outlines the comparison between central hydraulic system and EMA and EHA

5. Conclusion

The article has shown that although hydraulic power has dominated in commercial aircraft, the use of electric power is on the increase especially in modern aircraft.Thefore, the merits and de-merits of Mechanical Actuators (MA), Hydraulic Actuators (HA), Electro-Hydraulic Actuators (EHA), Electro-Mechanical Actuator (EMA) applications in commercial aircraft flight control systems have been considered.The comparison of these actuators reviewed that EMA was the best option for modern aircraft. The EMA was suggested to be the best option due to circuit simplicity, high efficiency, low maintenance and inspection cost, safety,airworthiness, weight and reliability.

Table 1: Comparison between Central hydraulic system and EMA and EHA

Actuator	Failure Management	Efficiency	Sensitivity To Wear
HA	Has a very low jamming probability and is very reliable. This is why the technology is still prime for use on aircraft, regardless of the technology being older.	Has its weaknesses in efficiency, assembly and repair (since the hydraulic losses are huge), and the exchange of individual components which means a risk of hydraulic fluid contamination.	
EMAs	The weak point of the EMA is about it's about 100 times higher probability of jamming compared to hydraulic actuators. This means that if it is used on an application where jamming of the actuator is hazardous, additional features must be installed to overcome this failure. Also, the passive behavior of the EMA is sometimes a problem, as it needs a lot of back-driving force. Also accurate damping in passive mode is a problem which is not simple to solve	Very good in terms of efficiency and redundancy, as an electric actuator can easily be isolated from the system in case of failure, while not influencing the other actuators supplied by the same power source. Also, EMAs are very good for highly dynamic applications, as they own a very stiff transmission, which allows for very accurate control of the output stroke. A further advantage is the easy exchange of such a unit after failure, as it is only plug and play.	
EHAs,	Low jamming probability and excellent behavior after failure (e.g. damping in passive mode, easy to switch in bypass).This means in principle the EHA technology combines the advantages of the central hydraulic system with those of a standalone electric driven unit. The weak points of the technology is the slightly higher risk of hydraulic leakage compared to EMA and the problem of oil degradation over time, which makes it difficult to build a EHA which is able to work for 30 years without maintenance.	The same advantages as the EMA regarding redundancy and maintenance	A further advantage of the hydraulic ram is the fact that it is less sensitive to wear under varying load conditions as would occur, for example, on helicopter primary flight control actuators.

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Septage Management through Decentralization & Community Participatory approach.

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ABSTRACT

Decentralization approach to sanitation has improved the whole Septage management. A project on Decentralization of septic tanks through a Community Participatory Approach yielded tremendous results in Solwezi; an urban area in the North western province of Zambia.. The decentralized system effluent was within the Zambia Environmental Agency specification of: turbidity-15 NTU, Total Coliforms-72 CFU/100mls, Ammonia NH_2 0.04 mg/l, Sulphate-34 mg/l. The key phenomenon was to reduce Non-point source pollution to point source pollution, which allows monitoring and management A City of approximately half a Million people at 95% utilization of septic tanks means 478,000 people require 80,000 septic tanks if we consider 6 persons per household. By decentralization through grouping of septic tanks, only 4000 septic tanks are required for the same population resulting in 95 % reduction in non-point pollution. In terms of Septage quality, decentralized plants that covered approximately 200 household's improved Septage quality by 70% (Jeannette and Sebastian 2017). Lessons from Solwezi show that Development of Septage Flow Diagram (SFD) to know exactly how many septic tanks are available in a given area will enhances targeted intervention. To resolve Poor performance of onsite sanitation systems, community engagements on integrated sanitation chain through effective Community Participatory Approach (CPA) should be part of the strategy. The community should know "the what, how, and why adequate and decentralized septic tanks are affordable and sustainable. Making households aware of the basic conditions associated with septic tank construction like, water tightness of septic tanks, surrounding soil structure, hydraulic consideration in tank sizing is very critical.

***Key words**: Decentralization, Non-point, community participation*

1.0 Introduction

Strong underlying ideology to any study are critical, the approach in this research focused on general and later the specific principles in Septage Management. Critical was the Solwezi Baseline Survey (2016) and the National Urban Water Supply and Sanitation Program feasibility on Solwezi. The baseline carried out a comprehensive survey on Solwezi the provincial capital of Zambia's North Western Province. For Solwezi urban, the population

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has grown by more than 138% between 2000 and 2010 (MLGH, 2014). This growth poses a challenge to all service providers in terms of keeping up with the growing demands of the population with sanitation being one of these areas.

2.0 Problem Statement

The increasing population of Solwezi has created additional demand for sanitation services. The sanitation coverage in Solwezi is only 21.6 %(NWASCO 2017) and mostly depends on individualized solutions such as the construction of “improper” cesspits, septic tanks and soakaways. There has never been any Centralized or offsite sanitation systems due to huge investment requirement. Solwezi alone requires 90 Million US Dollars (NUWSSP 2015). Local Planning Authorities are struggling to fulfil their mandate in the provision of planning and development control: Informal settlements have come up in several locations. In the absence of reliable and comprehensive spatial plans and related information such as plot ownership and use, the development of functional centralized systems can be extremely difficult. However, decentralization of septic tanks in Septage management and recognition of the benefits of onsite sanitation is a reality. Stakeholders should accept the fact that the majority of the population cannot afford the high costs associated with offsite sanitation.

3.0 Rationale

According to the National Vision 2030, 100 percent of the population should have access to clean water supply and 90 percent of the population should have access to sanitation by 2030.To actualize this vision, we require sustainable solutions that take into consideration the plight of the poor and marginalized community. Onsite solution that can reduce maintenance cost like decentralization of septic tanks can manage the Septage problem. In adopting this approach, we shall avoid the risk associated with majority of the wet systems that do not fulfil the standards associated with septic tanks and mostly inadequately designed and operated. By decentralization, the cost of grouped septic tanks shall be manageable as compared to individual units. In a City of approximately half a Million people at 95% utilization of septic tanks means 478,000 people require 80,000 septic tanks if we consider 6 persons per household. By decentralization through grouping of septic tanks, only 4000 septic tanks will be required for the same population resulting in 95 % reduction in non-point pollution. Introduction of decentralized plants that cover 200 households further reduces the pollution by 30%-50% as the reduction in multiple points of discharge has a corresponding reduction in amount of imminent pollution. In decentralization, bio digesters are utilized which ultimately improves Septage quality by 70% (Jeannette et.al 2017).Poor managed Septage can be a source of pollution (figure 3 and 4)

Fig 3: Dry and abandoned septic tank **Fig 4: poorly maintained septic tank**

4.0 Objectives

- Demonstrate the risk associated with poor Septage management
- Demonstrate that decentralization of onsite systems is sustainable in Septage management.
- Recommend a turnaround strategy in Septage management when adopting decentralization of onsite units.

5.0 Methodology

North Western Water and Sewerage Company received financing for a Pilot Project through Devolution Trust Fund (DTF) in collaboration with Bremen Overseas Research and Development Agency (BORDA) to decentralize 250 individual septic tanks through a community participatory approach.in Kandundu –C an urban compound in Solwezi. The area had disused dilapidated isolated septic tanks which were re-designed and joined to a new decentralized. Communal Anaerobic Digesters (ADs) of 40 houses each were also part of the intermediate primary settlers that produce biogas as a by-product of the system. The additional benefit was the Biogas production that helped households in cooking needs. Individual households were connected and gas stoves provided after a rigorous community sensitization.

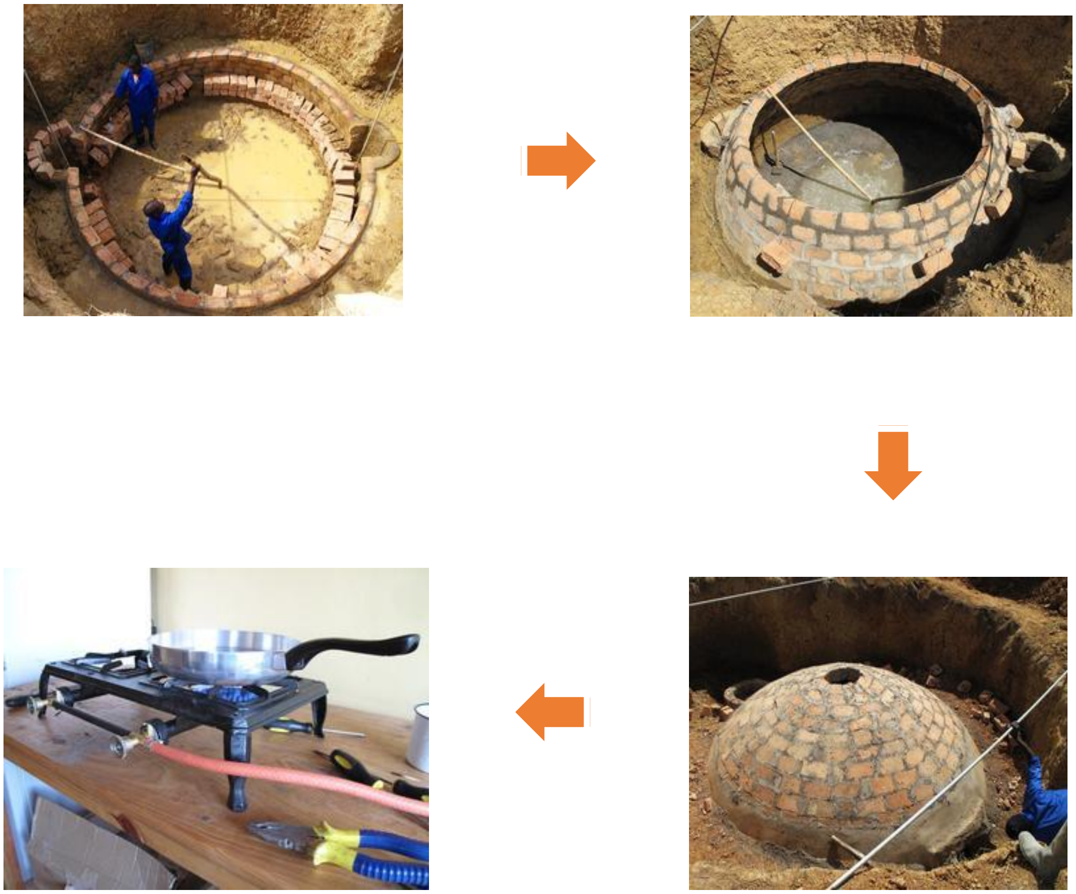

Fig 5: Communal Anaerobic Digesters (ADs) and ultimate benefit through biogas (courtesy: Pilot Project)

5.1 Adoption of Community Participatory Approach (CPA)

Over 99% of the population (SUBS 2016) currently rely on affordable onsite sanitation systems. In most cases not well constructed, operated and maintained. However, they provide the same health and environmental benefits as offsite systems at much more affordable costs and lower risks. There was need for community sensitization on what, how, and why of adequate septic tanks. For this reason, North Western Water and sewerage chose to engage the community during the process of decentralizing 250 individual septic tanks.

The process involved the following activities:

a) Sanitation mapping – a survey was carried out, to know what sanitation options existed in the compound. The rationale was not to impose any sanitation option.

Development of a shit flow diagram – the team had to:

- *Define the problem and state of septic tanks*
- *Measured and counted how many septic tanks existed.*
- *Analysed the data on the risk of pollution by onsite systems*

- *Proposed Improvements to the process of septic tank construction.*
- *Developed an in-house standard on Septage management*

b) **Task and skills available** –the project identified skills and personnel like bricklayers and plumbers and subsequently trained and employed them to perform masonry works during construction.

c) **Technical information and choices** – the Project technical staff introduced several sanitation options and allowed people to make informed choices at every level.

d) **Training on safety was important**- the team conducted a full training community on health and hygiene. Additionally all beneficiaries of biogas stoves which is a by-product of decentralized septic tanks were fully trained.

6.0 RESULTS

6.1 Comparison using effluent Quality

As compared to other sanitation options, Decentralization of septic tanks reduces the risk of imminent pollution and improves Septage quality by 70%. The effluent data show good removal of the Total Coliforms. The effluent does fulfil ZEMA standards in terms of Ammonia, Sulphate and Turbidity removal as seen below

Table 1: Decentralized onsite system effluent parameters

Parameter	***Units***	***ZEMA specification***	***Results***	***Compliance to Specifications***
Turbidity	*NTU*	*15*	*4.8*	*Satisfactory*
Total Coliform	*CFU/100mls*	*2500*	*72*	*Satisfactory*
Ammonia NH_2	*Mg/l*	*10.0*	*0.04*	*Satisfactory*
Sulphate	*Mg/l*	*1500*	*34*	*Satisfactory*

6.2 Comparison using the Financial, Institutional, Environmental, Technological & Social (FIETS) criterion

FIETS criteria forms generally accepted reference framework that incorporates multi facets of sustainability and makes this container concept ‘workable” the main reason for using it to analyze Decentralization of septic tanks and community participatory approach in Septage management.

- **Financial Aspects**: Decentralized septic tanks have low maintenance cost, hence poor and marginalized communities will not pay more for sanitation services.
- **Institutional aspects**: the system enjoys minimum regulation from regulatory bodies, apart from ZEMA.
- **Environmental aspects**: As seen from the results, positive or no adverse effects on the continued availability of natural resources and reduction of pollution.
- **Technological aspects**: Decentralized septic tanks can last for many years. It is favorable and affordable by the local community.

- **Social aspects:** The community in urban Solwezi have demonstrated that they socially and culturally accept this technology. Women are able to use biogas for cooking while the poor have reduce cost on electricity when cooking.

7.0 Conclusion

In this study, I found that urban areas like Solwezi Should not exclusively develop extremely expensive offsite sewerage solution while 99% of the population rely on affordable onsite sanitation systems. The focus should be on improving existing units through decentralization of septic tanks. This whole exercise should involve the urban communities affected. Results from the effluent parameters have shown efficient treatment of Septage through decentralization of septic tanks. Additionally, the FIETS criterion has illustrated clearly, why decentralization of septic tanks and community participation is a priority in urban areas that have similar geographical factors like Solwezi. The cost of a centralized of-offsite system for Solwezi is 90 Million USD due to the topography of Solwezi that will require frequent pumping and river crossings. However, decentralization of these septic tanks will only require 40% of this cost as estimated based on the Solwezi Pilot Project on Decentralization.

8.0 RECOMMENDATIONS

Based on results and comparison with FIETS criterion, stakeholders should recognize the benefits of onsite sanitation and accept the fact that the majority of the population cannot afford the high costs associated with offsite sanitation. Pay at least the same attention if not more to onsite systems as to offsite / sewerage system. Community participatory approach should be adopted in sensitization on “the what, how, and why adequate and decentralized septic tanks are affordable and sustainable.

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Future petroleum - experimental design and analysis of petroleum extracted from waste plastics/tires

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Abstract

This paper presents a first ever cutting edge designed pyrolysis pilot plant that converts all non-biodegradable polymers or used car tires direct into petroleum under green self-energy-sustaining process. This petroleum product is a hydrocarbon mixture of petrol, diesel and kerosene. In this paper, this hydrocarbon liquid mixture is referred to as polyfuel. Escalating demand on usage of plastics has led to a negative environmental impacts both on land and marine life. Worldwide almost 20% of the waste stream are plastic and over 30% of this waste plastics still ends up in landfill, recycled or at worst incinerated, a terrible wastage of resources that contains higher latent energy. Several successful experimental trials have been run on the prototype using two major raw materials waste plastics and used car tires. The petroleum recovery was above 84% of the total mass with a high calorific grade polyfuel product. The results show a variance of 20% more polyfuel recoveries from plastics than tires. The experimental design is based on pyrolysis technology that uses thermal catalytic depolymerisation of long chain polymer to short chain polymer molecules. This technology has huge potential to create future waste-to-energy petroleum source, climate change mitigation, solid waste management and acts as the lasting solution for all non- biodegradable materials that contaminates water bodies and kills marine life. The polyfuel could be used for all industrial thermal boiler requirements, domestic cooking, generators, cars and at the same time improves sanitation, creates jobs, combats deforestation and increases energy security. Developing nations especially in Zambia where solid waste management is a real challenge, root causes of drainage blockages that leads to a sanitation/health crisis due to water bone diseases outbreak.

Keywords: Polyfuel, Plastic waste, Used tires, Petroleum, Climate Change

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1. Introduction

For the past 5 decades, plastics have become at the centre of our civilisation and an important product in our daily life that is improving the quality of life Sharuddin *et al* (2016). Its competitive chemical properties such as inertness to corrosion and reaction, light weight and its insulation attracts wider application. It is one of the main products in medical healthcare, construction industry, car industry, manufacturing, electronic industry, electrical industry and product packaging industries. The demand for plastics is increasing despite most governments around the world enacting policies that discourages itself usage. Sharuddin *et al* (2016) summarised the outlook; the global production of plastic stands at 299 million tons in 2013 that has annually been increasing by 4% from 2012. The escalating plastic demand has led to increased accumulation of waste each year. About 33 million tons plastic waste in USA is generated, while Europe's 25 million tons of plastics find itself in waste stream Jambeck *et al* (2015). In Europe alone about 38% of the plastic waste goes to the landfill, of which 26% are recycled. On other hand there are about 1.2 billion (source: global green car report, 2018) cars on the road in 2018. On average, per every 40,000km a set of tires are discarded that has high latent energy for polyfuel raw material.

In Zambia alone, Lusaka is characterised by relatively large areas of unplanned or peri-urban settlements with a low-income population Walker, (2015), Mayumbelo *et al* (2007). An estimated 70% of the population of Lusaka lives in these areas. Although the formal employment rate in the peri-urban areas is low, there are a large number of informal businesses and workshops, which carry out different kinds of services. All these businesses, households and markets generate large amounts of solid waste. Households are by far the main generators of waste, contributing 233,000t/year, or more than 80% of the total amount (source: LSWSMP report, 2010). Especially the large peri-urban areas are the main source of all waste (approx. 66% of all waste). Industrial waste accounts for less than 8% of all waste. Household waste is mainly composed of organic waste and other types of waste (source: LSWSMP report, 2010) which constitutes 56.4% of household waste, constituting of leaves, soil and ashes from yard sweeping and cooking and whereas cardboards constitutes about half of commercial waste.

The stress on biomass energy source in Zambia is increasing, with 60% of the population without access to electricity, putting more challenge on deforestation and land use. With global call on climate adaption and mitigation the new methods and technology is a must for the survival of future human race Scheirs *et al* (2006). Though in recent years, governments around the world are enacting legislature to discourage plastics usage. These waste materials in Zambia has been a real challenge, most streets and back yard is littered with waste plastics, the municipalities has limited capacity to collect this accumulation of waste on daily basis Nchito *et al* (2004), Asomani-Boateng *et al* (2004). Hence these waste stream are washing into drainages and water wells during summer floods especially in high density areas. This floods acts as vector breeding grounds for water bone diseases such as cholera, dysentery and mosquitoes Guerrero *et al* (2013). The impact on the environment is high, loss of lives and contamination of water bodies.

Polyfuel is a hydrocarbon liquid mixture of petrol, diesel and kerosene. Pyrolysis process becomes an option of waste-to-energy technology to deliver recycled (renewable) fuel to replace fossil fuel Miandad *et al* (2016). Pyrolysis technique is the process of catalytic thermally degrading long to short chain polymer molecules, through application heat and pressure in the absence of oxygen.

The three major products of polymer pyrolysis are petroleum, gas and char which are valuable for industries especially production and refineries.

This paper presents first ever in Africa a feasible, practical designed working prototype with highest fuel recovery rates at lower temperatures. The polyfuel can be used in multiple applications such as furnaces, boilers, turbines, gas stoves and diesel engines in its crude form without further purification or treatment. Further fraction distillation of polyfuel yields pure diesel, petrol and kerosene that is the same grade as conventional fossil fuel and can be used directly into cars without blending. This will open up doors to real climate change mitigation, sanitation and sustainable energy solution for fourth industrial revolution.

This paper is organised as follows: Section 2 presents the process and experimental setup, Section 3 present the results and discussion. The last section 4 presents the conclusion

2. Polyfuel ProductionPolyfuel Process Design and Experimental setup

2.2 Process formulation

The production process of Polyfuel is generated by thermal catalytic de-polymerisation of plastic, tires, electronic body and rubber, waste oils under pyrolysis reaction Demirbas, (2004), Kaminsky *et al* (1999) and Williams *et al* (2007). Pyrolysis involves thermal break down of long chain to short chain polymer molecules in the absence of oxygen. Figure 1 shows the full process diagram. Plastics (Polyethylene, Polypropylene, Polyamide, Polystyrene) or used tires/rubber are collected from the environment/dumpsite and threaded into pieces, the raw material under goes preheat treatment in order to save energy and is fed into the reactor where pyrolysis takes place. Then the collected hydrocarbon liquid (polyfuel) is further distilled to get the actual components gasoline, kerosene and diesel. The by-product is carbon char and syngas which is mainly methane gas and mixture of moisture and other gases.

The syngas (flue gas) is high in caloric value and is used to provide heating for the reactor, making it energy self-sustaining. Polyfuel has many applications in industrial boilers, combined heat and power (CHP), furnaces, cooking stoves, diesel engines without need for treatment or upgrade like in the case of biofuels. Polyfuel production is more environment friendly that has no washing process in comparison to a recycling process that requires volume of water that contaminates the environment. The process requires less sorting process and lower labour intensive. It the process does not produce any greenhouse gases (GHG) and happens at green process.

2.3 Prototype Design and Experimental setup

The prototype is shown in Figure 2, with the major components. A closed loop water is circulated in the condenser for cooling purposes in the heat exchangers at room temperature. The bubbler is filled normally with water and the methane gas bubbles through the water into the atmosphere. In the big industrial setup it is collected for other application such as heating the reactor or sold for domestic cooking purposes.

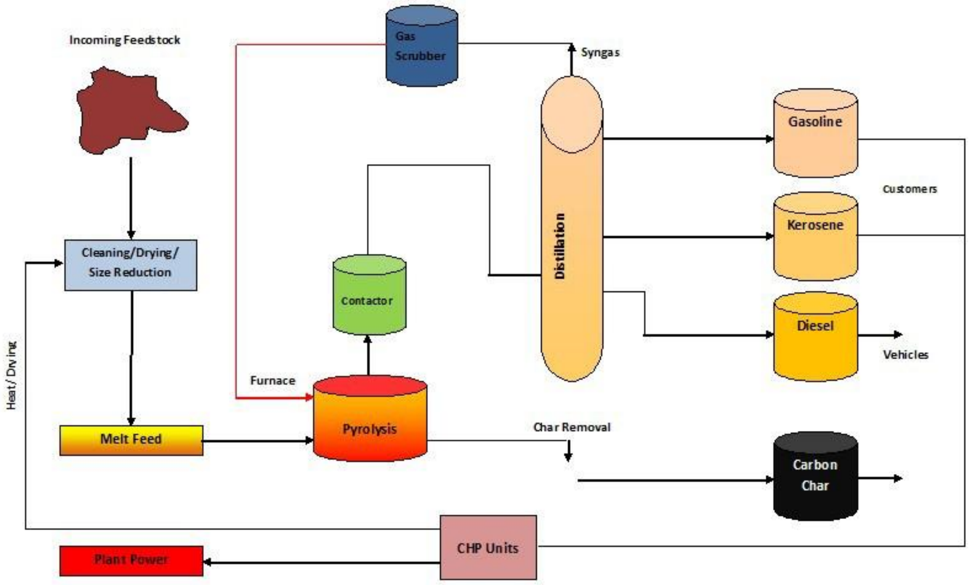

Figure 1: Process layout of polyfuel

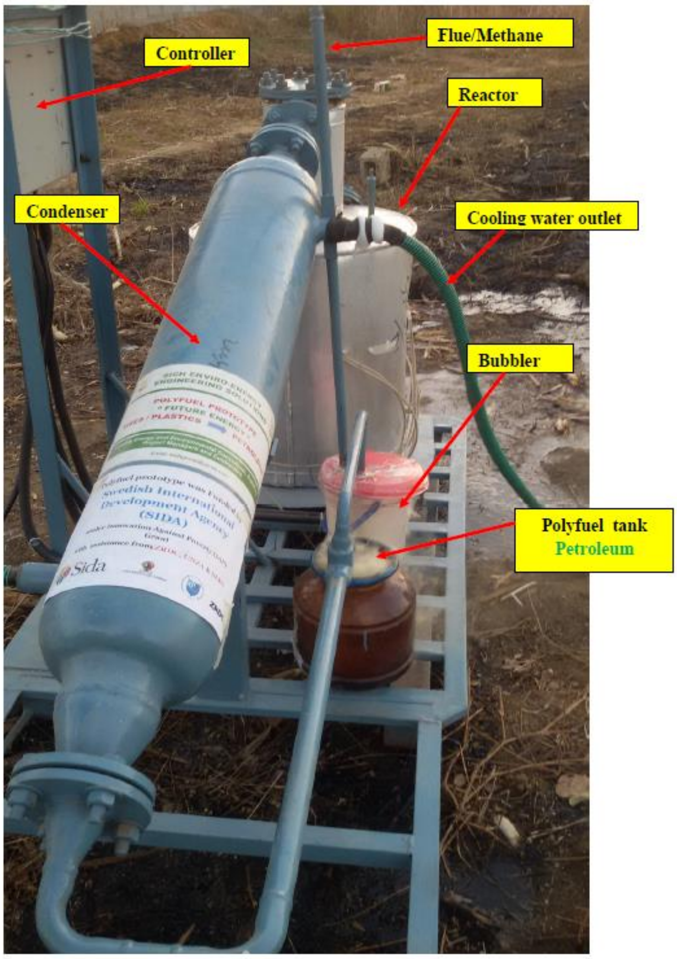

Figure 2: Experimental setup

The prototype has catalytic and material input powered by a single phase AC heating elements. Sources of reactor heating can be coal, biomass, electrical or methane gas any energy resources that might be available.

2.4 Raw materials for polyfuel production

There are several sources of raw material for this innovation, waste plastics such as Polyethylene terephthalate (PET), High-density polyethylene (HDPE), Low-density polyethylene (LDPE), Polypropylene (PP), Polystyrene(PS), ABS, Polyvinyl chloride (PVC), Polyethylene (PE), Acrylonitrile butadiene styrene (ABS), Polyamide (PA) or Nylons and Polybutylene terephthalate (PBT). Used car tires, Electronic scrap, Rubber and used oils. Figure 3 shows several sources of raw materials in Lusaka city. About 233,000 of tonnes of solid waste per year is dumped at Chunga dump- site (source: LSWSMP report, 2010) however a significant tonnage of this waste is not collected and find themselves in drainage, road side and city centres posing huge sanitation challenges.

(a) Soweto market

(b) Mixed plastic bottles

(c) Plastics palettes

(d) Used car tires

(e) Kamwala south ring road-Lusaka

(f) Tyre mending kiosks

Figure 3 (a-f): Raw materials and sanitation challenges in certain parts of Lusaka City

These valuable raw materials are treated as wastage but with this innovation, this waste can be turned into high grade fuel, producing energy from waste at the same time solving sanitation problem. Fewer plastics will reach rivers, wells or oceans to contaminate water bodies and destroy marine lives. This perceived 'waste' resource is often a challenge can be transformed into high value product and create thousands of jobs for collectors/distributor. Laws alone restricting/banning plastics usage will not remove them from the environment, neither stop its production. In every packaging and now construction material, plastics is replacing metal owing to its superior properties non corrosive and longevity. Paper bags that are being promoted have far more detrimental effects on climate change than imagined because it is proportional to tree cutting (de- forestation).

2.5 Results and discussion

2.5.1 Fuel production results

The fuel production and its quality principally is measured using proximate analysis due to their unique bond composition. Different types of polymers have distinct molecular composition, there- for this proximate technique measures the chemical properties of the plastic or tire compounds. Mainly four chemical elements are considered which are fixed carbon, volatile matter, moisture and ash contents. The quality of polyfuel recovery is proportional to the content of moisture and ash contents in the hydrocarbon liquid. The lower the contents the higher the yield and vice versa. Figure 4 summaries the proximate analysis results in terms of percentage yield from two materials involved in this experiment. Plastics proves to have higher polyfuel recovery, with ranges of about 70-90% petroleum and 5-8% char. The other by-product approximate 5-10% is methane and other flue gases. The flue gases doesn't do not contain carbon dioxide gases because the reaction happens in the absence of air containing oxygen for burning.

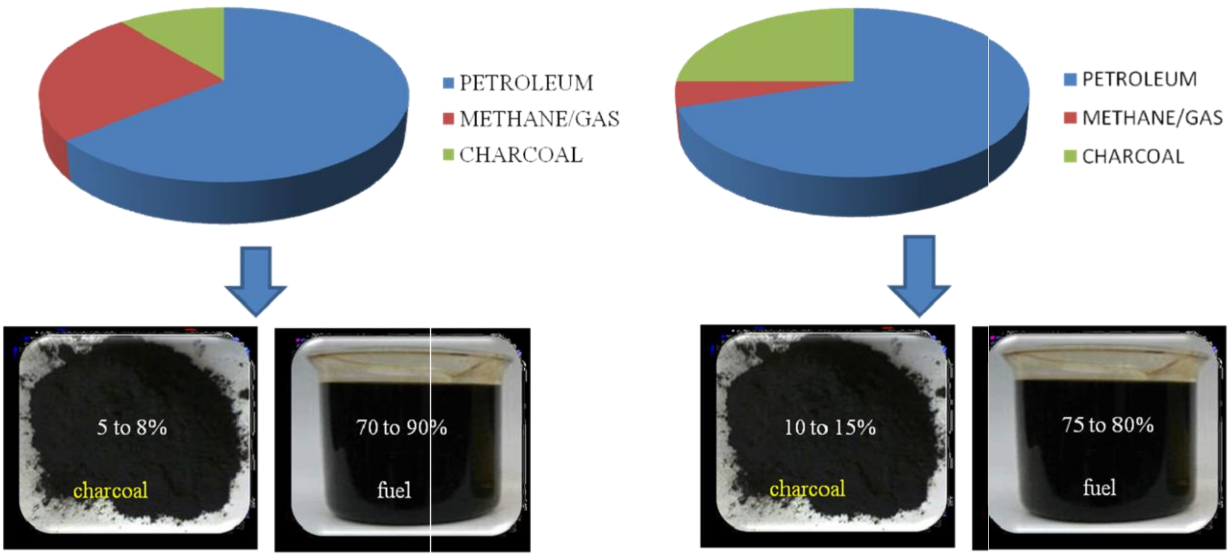

Figure 4: (a) Petroleum from Plastics *(b) Petroleum from Tires*

The tire on the other hand has significant higher percentage yield of char, the outputs are about 75-80% petroleum and 10-15% char. This can be attributed to the high rubber content that is non petroleum base coming mainly from rubber trees. While Table 1 shows the main chemical parameters of this petroleum. The polyfuel testing samples were taken to collaborating partners for testing, the results presents a huge potential of polyfuel of becoming next generation oilfields. Petroleum through plastics has higher percentage recovery at relatively low temperature starting from $60 - 71.0^{o}C$. This ’green’ energy is posed eventually to replace conventional fossil fuel.

Table 1: Recovery temperatures and quality indicators of polyfuel

SR. NO.	TEST	METHOD OF TESTING	RESULT
1	Density at @$15^{o}C$	ASTM D 4052:2002	0.7930 gm/ml
2	Acidity (mg KOH/gm)	ASTM D 974:2002	0.76
3	API Gravity @ $60^{\circ}F$	ASTM D 1298:1999	46.67
4	Flash Point COC	ASTM D 92-05a	<$40^{\circ}C$
5	Kinematic viscosity @ $40^{\circ}C$	ASTM D 445:2005	2.149 mm2/s
6	Colour	ASTM D 1500:2004a	D 8
7	Conradson Carbon residue	ASTM D 189:2005	0.010% (wt)
8	Asphaletene content	ASTM D 3279:2001	0.21% (wt)
9	Ash Content	ASTM D 482:2003	<0.01% (wt)
10	Calculated carbon aromatic index	ISO 8217 :1996	763.4
11	Pour Point	ASTM D 97-05a	$14^{\circ}C$
12	Sediment by extraction	ASTM D 473:2002	0.012 (wt)
13	Specific gravity @ $15^{\circ}C$	ASTM D 4052:2002	0.7932
14	Sulphur content	ASTM D 4094:2003	75 ppm
15	Water by distillation	ASTM D 95-05el	<0.05% (vol)
16	Distillation range	ASTM D 86:04 b	
	Initial boiling point		$52.0^{\circ}C$
	05% Recovery		$54.0^{\circ}C$
	10% Recovery		$55.0^{\circ}C$
	20% Recovery		$58.0^{\circ}C$
	30% Recovery		$59.0^{\circ}C$
	40% Recovery		$67.0^{\circ}C$
	50% Recovery		$68.0^{\circ}C$
	60% Recovery		$69.0^{\circ}C$
	70% Recovery		$70.0^{\circ}C$
	80% Recovery		$66.0^{\circ}C$
	85% Recovery		$65.0^{\circ}C$
	90% Recovery		$65.0^{\circ}C$
	Total Recovery		90% (vol)

In Table 2 shows the analysis of the by-product char or charcoal chemical properties. The char is chemically alkaline with pH-Value = 9. This char can be used to neutralise acidity in farming fields, enrich the soil and can be used in pharmaceutical industry as colorant absorbent. It shows high moisture retention rate that is better to reduce surface soil evaporation.

Table 2: Char chemical analysis from Tyres

Specifications	Analysis method	Charcoal from Tyre	Carbon black grades	
			HAF(N330)	SRF(N754)
Iodine Absorption Number, mg/g	ASTM D-1510	176	82	24
DBP Absorption Number, $cm^3/100g$	ASTM D-2414	78	92-102	50-65
CTAB surface Area, m^2/g	ASTM D-3765	68	74-90	23-39
pH-Value	ASTM D-1512	9	9-Jul	10-Jul
Heating Loss, % max	ASTM D-1509	1.5	2.5	1.5
Ash content, % max	ASTM D-1506	10	0.5	0.5
Average particle size, nm	Microscope Electronique	300 mesh		

3. Conclusion

The waste tires/plastic materials are converted into high grade petroleum (polyfuel) which is a hydrocarbon liquid mixture of petrol, diesel and kerosene with methane vapour. The prototype uses pyrolysis with a recovery rate of (60-90%) and a by-product ≈ 5% methane gases while the other is char residuals. This other char by-product is chemically neutral ≈5%. The crude polyfuel can be used for most of the energy needs and a further purification produces diesel, petrol and kerosene. The actual technical products in terms of percentage composition from the major two raw materials are as follows; From Tires: 45% Petrol + 40% Diesel + 15% Kerosene, Flash Point: 25 degree C, Pour Point: -21 degree C, Calorific Value: 10,150. From Plastic: 20% Petrol + 40% Diesel + 30% Kerosene, Flash Point: 40 degree C, Pour Point: -4 degree C and Calorific Value: 10,000.

Polyfuel has more advantages owing to higher calorific values, way above coal or biomass used as primary energy inputs/sources in Zambia. Since everyone uses energy and polyfuel will surpass conventional energy sources owing to its several added advantages, that it can be readily available for usage without preparation or blending. Polyfuel in its crude form can run CHP generators producing electrical energy that can fed into the main power utility grid system (off taker). With the abundance of raw materials, polyfuel becomes the cheapest form of energy in comparison to all other energy sources on the market. In addition, the following multiple benefits will be achieved upon implementation this innovation:

- problem of disposal of waste tires/plastic is solved,
- environmental and water bodies pollution is controlled,
- industrial and automobile fuel requirement shall be fulfilled to some extent at lower prices,
- greenhouses gases mitigation as there is no emission of pollutant during cracking of tires/plastics,
- polyfuel offers 100% solution for all non-biodegradables polymers which contaminates water bodies, blocks drainage and acts as vector breeding grounds,
- Superior climate change mitigation innovation, an energy recycling and permanent solution for all plastics than current solution under 3Rs, (Reduce, Reuse, Recycle)
- has potential to creates combined thousands of (direct and indirect) employment of low income people as raw material collectors and polyfuel distributors,

- is an alternative and reduces dependence on fossil fuels (petroleum) and increases national energy security,
- has positive impact on the reduction on deforestation owing to a relief on charcoal demand and,
- direct mitigation on sanitation and a source of cheaper energy.

The petroleum tested from waste tires/plastics complies with conventional fossil fuel by any internationally recognised standards anywhere in the world. The key indicators include flash point, viscosity, density etc. are within specified standards for fuel. The fuels are suitable for all engines including road engines such as trucks, buses, cars, motorcycle, as well as heavy machinery, generators, boilers and other stationery engines. The innovation has potential to be the future oilfields when all oil wells goes dry and save the dying marine lives that are swallowing plastics. It has potential of on long term transforming fishermen into fishing oceanic plastics instead of fish in our vast rivers and large water bodies.

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Industry perception of student industrial training as practiced by the University of Zambia

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Abstract

Industrial training, the activity of supervised practical training in real-work environment for defined periods of time before graduation, is an important component of the complete training in the undergraduate engineering programme as designed at the University of Zambia. In the recent past, the trend shows a general reduction in the opportunities for students to undertake industrial in Zambia. With the intention to improve this situation, this work has interrogated the factors that may contribute to the current situation. The Zambian industry is deemed to be one of the key stakeholders in the industrial training enterprise. By a process of surveys of the Zambian industry and following a statistical approach, the assumed perceptions of Zambian industry on the programme of industrial training as practiced by UNZA are tested. The results reveal more confirmations than rejections of the assumed perceptions. The components of the rejections as revealed by this study are unexpected, and are therefore to be specially considered in devising a corrective strategy for improving the industrial training experience.

Keywords: curriculum, engineering education, industrial training, work-integrated learning, dual-studies,

1. Introduction

Industrial training is generally understood to describe the activity of supervised practical training in a real-work environment for a defined period of time before graduation (Aziz et al, 2012; Meenaloshini et al, 2014). The general aim of this activity is to equip the engineering students with real-world experience in a supervised manner and in the process enhancing the internal university assimilation of both soft and hard skills. Tertiary institutions worldwide who adopt this practice maintain the term of industrial training, but others have morphed it in terminology and practice, such as *dual-studies*

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(mainland Europe, USA, Brazil) (Graf et al, 2015) and *work integrated learning* (WIL) (Australia, Canada, South Africa and Namibia) (Bowen and Drysdale, 2017; Agwa-Ejon and Pradhan, 2017).

Since its inception in 1969 the University of Zambia (UNZA) has applied this approach in the training of students in the discipline of engineering. Up until the early 1990s, which marked the beginning of the privatisation of state-owned enterprises, the execution of the industrial training activity, in the partnership between UNZA and the industry sector with engineering activities, had appeared to be smooth, achieving 100% placement of students in industrial situations and having a general level of high satisfaction by the involved parties. From this period, the apparent lack of sufficient positions for industrial training became noticeable, with this problem worsened by the increase in enrolment of students in engineering studies at tertiary level following the instituting of legislation which allowed more players in the higher education sector (GRZ, 1999). The current situation is that university administrations are struggling to fully place students in industrial training positions, resulting in delayed graduation of students and a degree of dissatisfaction in both the quality of the intended training of the students and the value expected by employers in fresh graduates.

This work has focused on examining the long-held positions and views on industrial training by the engineering industrial sector in Zambia. These views and positions are tested to determine their validity in the current environment and to form a basis for the factors that account for the formulation of a direction for remedies. It is understood that this situation stands also for other stakeholders such as students and university authorities, but the examination of their views and positions is expected to be done in another related study.

This presentation is organised to start with an introduction in Section 1, followed by a description of the features of the industrial training at UNZA in section 2. A method for the study is presented in section 3 while the results and a discussion appear in section 4. Finally in section 5, a conclusion is offered.

2. Attributes of industrial training at UNZA

In the five-year undergraduate engineering programme at UNZA, of which the last four are in the School of Engineering, the industrial training is conducted in the vacation periods at the end of the third and fourth years of study, as two non-credit courses or modules. Taken together, a minimum of 24 weeks of actual training in industry is required for satisfactory completion of the industrial training suite of courses.

While in industry on training, students are expected to keep a daily log of activities of their training and have a final technical report at the end of the training. The assessment of the students' industrial training is based on consideration of the duration of the training period in industry, the contents of the daily log book and the final technical report, together with a confidential report prepared by the supervisor of the student at the host industry. Despite these components being designated as non-credit, the industry training activity is compulsory for all students, and the activity produces a grade which is either satisfactory or unsatisfactory.

The annual required placement of students into industry from UNZA's School of Engineering is in the range 200-250, depending on the annual enrolment. The placements cover the five disciplines of the School of Engineering, as illustrated in Table 1 for a typical recent academic year. From 2013 to 2015 the rate of placement of students for industry training had ranged between 52-82%, when considered as segregations into the five engineering disciplines as well as into years of study.

Table 1: Enrolment and industrial placement figures for 2015 academic year

Discipline	AE	CEE	EEE	GE	ME	Overall
Enrolment	23	85	66	18	53	245
Placements	19	53	47	14	35	168
Rate of placement (%)	82.6%	62.4%	71.2%	77.8%	66.0%	68.6%

Legend: AE- Agricultural; CEE- Civil & Environment; EEE – Electrical & Electronic; GE – Geomatic; ME – Mechanical.

Even without considering the quality of the training and the value realised by the training institutions, the employing institutions and the students from the training activity, the placement rate typified by Table 1 puts the university in an unfavorable position. This is the current trend at the university, which may persist or worsen if measures are not instituted to address the situation.

3. Method of assessing the perception of industry

The population of study was the industrial entities in Zambia associated with engineering the disciplines and engaged with industrial training of students. The School of Engineering at University of Zambia has a database of 110 industrial entities with these attributes, who in effect form the population of study. Due to limitations in resources and time, the whole population could not be queried. Instead a non-probability sampling method of convenience sampling was used, in which the respondent entities were drawn from accessible regions in the major industrial centres. The impact of the restriction arising from applying this approach is covered in the discussion section of this paper. A sample size of 30 was reached in this approach, with the respondent required to complete a set of selections from a purposefully designed questionnaire. The respondent in each industrial entity would be one of the officials from the following units of the industrial entity: management, human resources, or training.

Thirteen statements or questions describing the state of the industrial entity towards industrial training in the questionnaire were used to evaluate the inquiry indicated in this work. The statements gravitated into assumptions of the industrial entities about industrial training, and were thus turned in hypotheses. Statistical data was collected and processed as binary outcomes of "agree" or "not agree". In each category of the binary outcome there would be two degrees of the state of the outcome in the questionnaire, such as plain "agree" and strongly "agree" or plain "disagree" and strongly "disagree". The test statistic for significance of the results is formulated on the position that the each assumption being examined was held by at least 75% of the population: thus, the hypothesis test statement has the character of:

'At least 75% of the industrial entities agree with position that …'

The confidence level selected for reporting the statistics is 95%, requiring to set the significance level at $\alpha = 0.05$.

In the next section, results for following this approach are reported.

4. Results of the assessment and discussion

In this section, the results of the assessment to determine the validity of the assumed positions of industry on student industrial training are presented. The outcomes of this assessment are also discussed.

4.1 Results

The distribution of the sample used in the study (industrial entities) arranged according to regions (Provinces) and according to the engineering disciplines are as shown in Figure 1 and Figure 2.

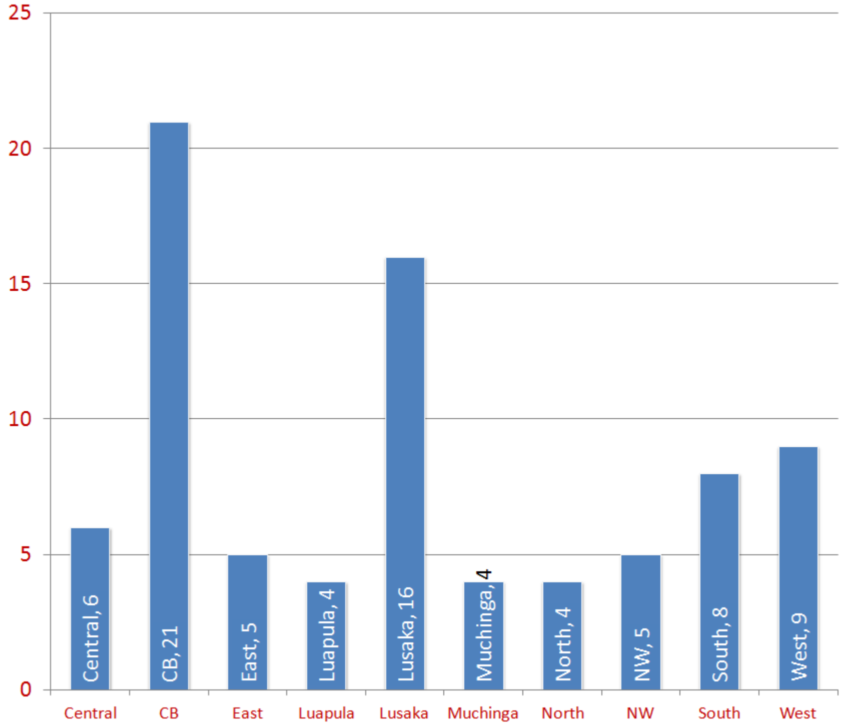

Figure 1: Distribution of the sampled-industry by region

In this sample, all the 10 provinces were represented but the dominant representations were for industries in Lusaka and Copperbelt provinces. When considered under engineering disciplines, the disciplines of civil, electrical and mechanical engineering dominate in the sample.

Table 2 is a list of the hypotheses presented to respondents to elicit the current positions of the entities on industrial training as practiced by UNZA.

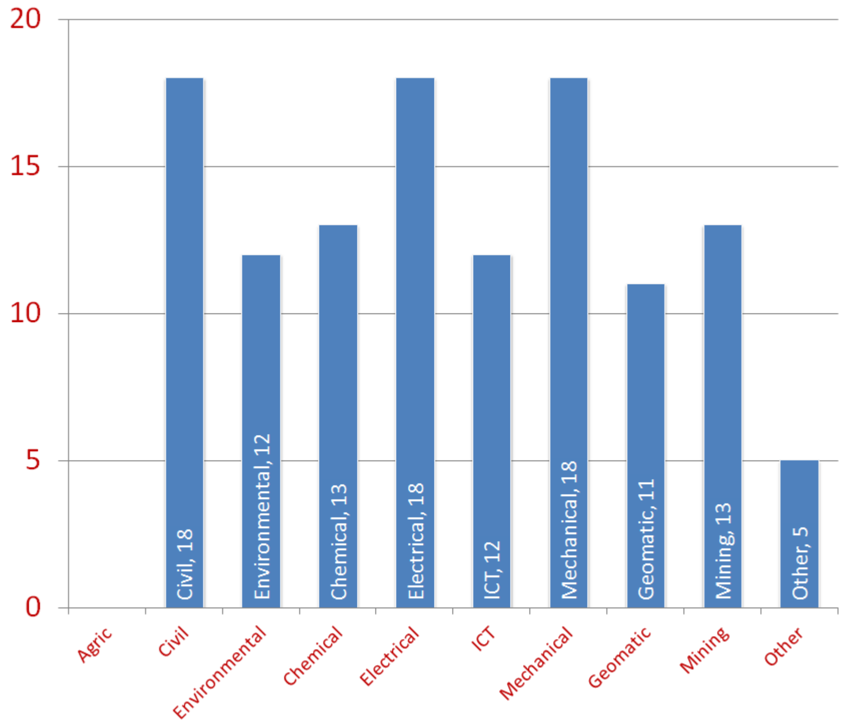

Figure 2: Distribution of the sampled-industry by discipline

Table 2: Hypotheses for surveying industrial entities

No.	Null Hypothesis - Industrial firm ...
1	values graduates with industrial training experience
2	has plans to admit engineering students for industrial training
3	admits engineering students for industrial training
4	has obligation to provide monetary allowance to engineering students on training
5	provides monetary allowance to engineering students on training
6	has guidance from the training institution on how to participate in industrial training
7	would value the feedback of the firm's participation in the industrial training of engineering students
8	has received feedback on the firm's participation in the industrial training of engineering students
9	has human and financial resources for industrial training of engineering students
10	has time for industrial training of engineering students
11	has arranged materials to support industrial training of engineering students
12	has qualified personnel to supervise engineering students on industrial training
13	has systems and processes to participate in training of engineering students

For each hypothesis, the test statistic is that at least 75% of the population hold the position claimed in the hypothesis. When tested for statistical significance, the hypotheses in Table 2 draw out the *p*-values as shown in Figure 3.

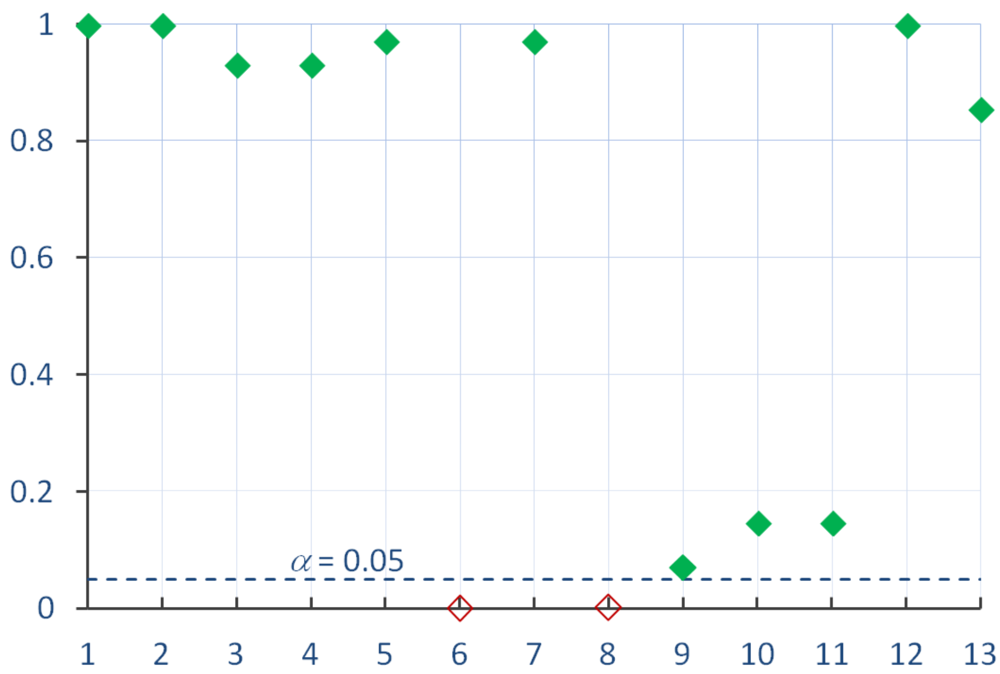

Figure 3: *p*-values for significance on test of hypotheses 1- 13

In all cases, we fail to reject the null hypotheses except in cases 6 and 8 (case 6: industrial firm has guidance from the training institution on how to participate in industrial training, case 8: industrial firm has received feedback on the firm's participation in the industrial training of engineering students).

Although the results of these 11 cases show that we fail to reject the hypothesis, the position for cases 9, 10 and 11 are not as strong as the rest of the cases, standing at what would be considered as marginal at 0.07, 0.146 and 0.146, respectively.

4.2 Discussion

Based on the findings as reported in the section on results, it appears that there are good grounds for the assumptions held by industrial entities on the industrial training as conducted by the university and as portrayed in the hypotheses of this investigation. As these assumptions had generally guided the policy and operating guidelines of both the industrial entities and the university on the industrial training, it can be accepted that the basis for the policies and operating guidelines is, to this extent, sound.

As it was encountered that two of the assumptions did not have supporting evidence to hold them, it is a fair position to face this challenge implied by results of the rejection of the hypothesis. It is noticed that the rejected assumptions allude to the operating relationship between the industrial entity and the university, where the role of the industrial entity should be defined and thereafter monitored. Further to this, attention could be paid to the weak support for assuming that industrial entities have resources (financial, material and time) for participating in industrial training activities as defined by the university. The rejection of the assumptions and the weak support as determined in the investigation stand as signals to re-examine the contra-indicated assumptions in drawing up new directions. In the desire to interrogate the factors that contribute to reduced opportunities for students to undertake industrial in Zambia, these results are not a revelation of such factors, since no correlation study has been attempted, but the results may be used as a starting point for self-examination.

It is taken that the findings in this work and the conclusions and recommendations arising from them are limited to the industry circumscribed in the convenience sample of 30 entities. While this sample size allows use of some inferential statistics, the extent of generalisation may be granted only up to the population of the 110 entities in the university database, and no further. This position is acceptable, as the study sought to only study the industry within the country.

5. Conclusion

This work has revealed that the generally held assumptions on industrial training by the industry in Zambia are largely applicable. However, in finding that a small percentage of the industry assumptions in the drawing of policies and operating guidelines for industrial training have an erroneous basis for holding them, it appears rational to re-examine such assumptions in defining a new direction for industrial training to be practiced by University of Zambia. As specifically revealed in this work, a different and more-engaging approach for the relationship of the university with industry on industrial training is required and so are innovative means for marshalling the spectrum of resources needed for this enterprise. This is recommended as a starting point for re-examination of the policies and operating guidelines for industrial training at University of Zambia.

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Importance of mining linkages development in promoting the growth of the Manufacturing sector in selected resource rich African countries: Study Review of Ghana, Botswana and Zambia

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Abstract

A critical issue in most mineral rich producing economies in Africa is the existence of poverty in the midst of abundant metals. This paper reviews the mining linkages with manufacturing sectors in Ghana, Botswana and Zambia with a view to highlighting extent of local content promotion in the supply of goods and services. The study review notes that Linkages between the African minerals industry and other economic and social sectors such as manufacturing in the three countries are generally not sufficiently developed, reflecting the industry's over-reliance on extracting and exporting minerals with limited value addition to overseas markets. The export of minerals in their raw forms means that opportunities for creating much needed jobs and transforming economies are lost. In Ghana, local firms are also largely unable to meet the supply needs of the mining industry in terms of quality, quantity and timeliness, while in Botswana the manufacturing sector remains small despite Government efforts to accelerate value addition to minerals. The picture is the same for Zambia with most of the goods and services required for the mining industry imported from other countries.

Therefore, there is need to strengthen local linkages between the mining industry and the local manufacturing sector in order to improve the socio-economic development of the countries by promoting value addition in industrial clusters, innovation, local procumbent and consumption. The Participation of local entrepreneurs in the local economies and supply chain should underpin the strategy.

Key Words: Mining Linkages, Manufacturing, value addition, local content

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1.0 Introduction

In 2009, the Africa Mining Vision (AMV) was adopted by African Heads of State and Government to offers a unique, pan-African pathway of reversing the old paradigm way of managing the African mineral wealth. It seeks to use Africa's natural resources sector to transform the continent's social and economic development to achieve better developmental outcomes. The Vision stresses the importance of a holistic approach to minerals development that adds value by: developing upstream linkages into mining capital goods, consumables and service industries; downstream linkages into mineral beneficiation and manufacturing; and side-stream linkages into infrastructure (power, logistics, communications, and water) and skills and technology development.

Linkages between the African minerals industry and other economic and social sectors are generally not sufficiently developed, reflecting the industry's over-reliance on extracting and exporting minerals with limited value addition to overseas markets. The export of minerals in their raw forms means that opportunities for creating much needed jobs and transforming economies are lost. A number of challenges inhibit development of economic linkages in the sector, such as externally-focused mining company procurement, significant skills gaps across Africa, and infrastructure deficits. With some exceptions, the African minerals sector generates little new knowledge in terms of mining-related products, processing technologies and services. There is limited domestic funding for technology and research, and generally weak partnerships between mining companies and local manufacturing companies. By enhancing linkages and diversification within and outside the extractive sector, resource-based industrialization has the potential to unlock more employment opportunities, higher incomes and increased socio-economic benefits in terms of development and poverty reduction in African countries.

This paper reviews linkages between the mining and manufacturing sectors in order to establish the state of linkage development in Zambia.

2.0. Problem back ground

African Resource rich countries such as Ghana, Botswana and Zambia continue to export minerals in raw form. These countries have very minimal linkages between the minerals industry and other economic and social sectors reflecting the industry's over-reliance on extracting and exporting minerals with limited value addition to overseas markets. The export of minerals in their raw forms means that jobs are externalized and opportunities for transforming economies are lost. This paper reviews the importance of mining linkages development in promoting growth of manufacturing sector. It highlights the challenges faced by the resource rich countries and ends by recommending the strategies for promoting enhanced mining linkages.

3.0 Inter-industry Linkage Development

Inter-industry linkages have been studied since the late 1950s with the purpose of identifying "key industries" that are central to economic development (Drejer, 2002). The linkage development bears its roots from Hirschman's Linkage theory. "Hirschman emphasizes the role of production linkages as a source of pecuniary externalities that come from input-output relations in two forms: Backwards linkages that lead to multiplier effects through inputs supply; and forward linkages that lead to multiplier effects through the processing of commodities" (Atienza, et al., 2018, p. 2). "Hirschman also acknowledges the existence of two other types of linkages but was skeptical about their potential contribution to the development of resource rich economies: consumption linkages, that lead to multiplier effects in local demand from the incomes earned in the commodity sector, are probably weak in the context of an open economy where workers import most products and services and fiscal linkages, related to the ability of states to tax the exploitation of commodities, whose contribution to development depends on the ability to invest productively" (Atienza, et al., 2018, p. 2).

The definition of a linkage effect is closely related to the discussion of how an input-output system emerges. Hirschman views the power of diversion as a measure of backward linkages based on mental experiment "assuming for every industry in turn that the country's development started with just that industry, so that all the industry's sales to and purchases from other domestic industries are imagined to have developed as a sequel to the foundation of the industry in question" (Drejer, 2002, p. 6). Hirschman's theory suggests that development is continuously evolving as long as effects are at play. However, he argues that misconception of the true character of linkages leads to claims that linkages are easy to make operational (Drejer, 2002). It is worth-noting that backward and forward linkages are not automatic and that the relationship between the market size and economic size of a plant that will trigger the private or public entrepreneurship needed to take up the opportunities for linkage investments. Variables such as technology and new economic activities in relation to the ongoing ones, as well as obstacles in the form of the need of large amounts of capital due to scale requirements and lack of marketing access and knowledge are also at work (Drejer, 2002).

3.1 Mining Linkage development in Ghana

Ghana is the second largest gold producer in Africa after South Africa and 11th in the world. While world-class mining supply and processing firms have emerged in South Africa, Chile and other mineral-producing countries, Ghana's mining sector inputs are mostly procured abroad, and its minerals are processed abroad. The Country importers and the mining Vision is guided by the African Mining Vision (AMV), adopted by African heads of state in 2009 to foster "transparent, equitable and optimal exploitation of mineral resources to underpin broad-based sustainable growth and socio-economic development." Ghana's structural weaknesses impede its drive for economic diversification and industrialization (ACET, 2017). Local firms are also largely unable to meet the supply needs of the mining industry in terms of quality, quantity and timeliness. Other issues include the strong presence of a class of business people who are essentially intermediaries between industry; a lack of political consensus on *how* to strengthen linkages between mining and industrialization; red tape and other difficulties in the business environment; and insufficient commitment from the mining industry itself to support local sourcing.

Gold accounts for over 96 percent of mined minerals in Ghana, and the minerals sector is responsible for roughly 37 percent of the county's GDP. The gold mining sector in Ghana employs over 17,000 people, and over 98 percent of them are native to the country. Over a third of production occurs in in scale mining operations and illegal mining remains a key issue in the country. It is estimated that USD $2.3 billion worth of gold was mined illegally in 2016 alone, with most of illegal gold being exported to India and China.

Lucrative natural resources, a stable political system, and a dynamic economy have enabled Ghana to become the second biggest economy in West Africa after Nigeria. However, Ghana faces economic challenges due to an excessive dependence on the mineral sector, and policy gaps in the extractive sector – notably with respect to beneficiation and value addition. (United Nations , 2016) The mining industry does not seem to connect sufficiently to other strategic areas of the economy, such as agriculture and industry. Ghana has recognised the need for diversification of its economic structure, with a stronger involvement of the local business sector, and by enhancing its human resource and technological capabilities to create more value and widen its economic space.

3.2 Mining Linkage development in Botswana

The economy of Botswana is highly dependent on the natural resources sector, mainly minerals. The country's nominal GDP has more than doubled over the last eight years, from P 73.5 billion in 2009, boosted mainly by prosperity in the mineral resources sector and more recently, by the expanding services sector. Despite the growing services sector (hotels and restaurants, banking and financial services), during the last few years and concerted efforts to diversify the economy, the highly capital-intensive mining sector remains the anchor for economic prosperity in the medium to long term; it accounted for an average of 19 per cent of GDP and 89 per cent of merchandise exports during the period 2012 to 2016 ((Botswana Ministry of Finance and Development, 2017). On the other hand, the small manufacturing and agriculture sectors, which hold the key to job creation and enhanced domestic linkages remain constrained because of a multitude of structural challenges, including low and unreliable rainfall (for agriculture) and supply problems related to electricity and water (for manufacturing). The manufacturing sector remains small in Botswana despite Government efforts to accelerate value addition to minerals (Figure 1). It accounted for 5.2 per cent of GDP in 2016, a decline from the share of 5.8 per cent in 2011. This decline can be attributed to challenges related to water and electricity supplies. The sector accounted for 9.2 per cent of total employment in 2016 (Bank of Botswana, 2016). Overall, mining continues to overshadow all other sectors of the economy in terms

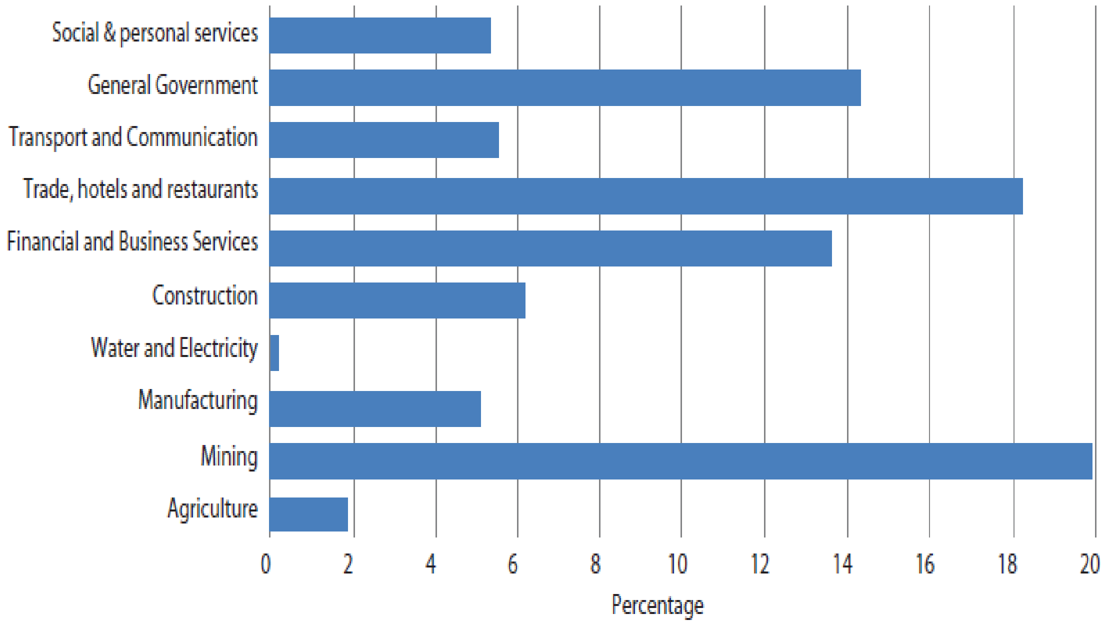

Figure 1 Sector shares and growth (2016) Percentage : Source (United Nations Comission for Africa,2017)

of contribution to value added. Consequently, the performance of the economy of Botswana remains intricately linked to that of the mining sector and to trends in diamond prices and export volumes.

Diamonds accounted for 92.1 per cent of the value of exports during June 2017. Overall, since the relocation of the De Beers aggregation and sales functions from the United Kingdom to Gaborone in 2012 and 2013 respectively, the share of diamonds in exports has increased on account of the re-export trade for rough diamonds. The relocation has generated business for the local diamond cutting and polishing sector, which despite the challenged competitive position, has provided local business opportunities in related activities. For example, De Beers's subsidiaries in Namibia, South Africa and Canada are exporting diamonds to Botswana for aggregation. This has also helped to improve the country's terms of trade, buoyed by the increase in foreign exchange. However, to further accelerate value addition and strengthen the international competitiveness of value-added diamonds from Botswana, a holistic approach to deal with the shortage of skilled personnel in the sector and the high cost of local labour is needed. Otherwise, the local Figure 10: Principal import commodity groups, June 2017 (percentage) Diamonds, 92% Machinery & electrical equipment, 3% Meat & meat products, 2% other, 3% Source: (Statistics Botswana, 2017e) sector will remain uncompetitive compared to the cutting and polishing centres in India, China, Belgium and Israel, for example.

3.2.1 Local beneficiation and manufacturing

Efforts to increase local beneficiation and manufacturing in the diamond sector has been undermined by lack of competitiveness, a direct result of poor skills and the high wages compared with other established diamond polishing and manufacturing centres, in Belgium, China, India and Israel, which have developed and perfected diamond value addition. This calls for a strategy to improve the skills base in diamond cutting and polishing for higher productivity and development of further downstream value addition into jewelry manufacture.

The creation of special economic zones is one of the strategies to promote export-led development, diversification and industrialization. Broadly, the objectives of this effort are to maximize use of local content in goods and services; promote technology transfer and innovation and management skills; develop a small and medium scale enterprises sector and attract FDI. Special economic zones include export processing zones, industrial parks, eco-industrial parks, technology parks and innovation districts. Furthermore, the Economic Zones have been targeting the development of mineral commodity value chains within the special economic zones and ensuring that domestic enterprises occupy specific positions along those value chains.

3.3. Linkages Development with the Manufacturing Sector in Zambia

Few studies have been carried out on manufacturing linkages in Zambia with the main focus being around mining (Miller, 2018). Kragelund (2017) looked at institutional impediments to resource led development and Miller (2018) talks about the significant advantages that could be accrued by mines in Zambia through buying manufactured products and maintenance services from local companies. Central to the discussions was the ability to promote local industrial development, a sector that has failed to take-off since its decline in the 1990s. Highlighted is how local procurement could be the new solution to the challenge and could also help in dealing with other economic challenges such as unemployment.

3.3.1 Local beneficiation and Manufacturing

The aspect of local content has become a major concern not just in Zambia but the world over. More recently we heard that the Australian Government expressed concern as regards the limited input by Australian entities into mining contracts. The Zambian Government is aware that the determination of what constitutes local content has proved elusive. But the Government knows that just as the mining companies seek innovative ways and means to explore, identify and develop mining projects they must also seek ways and means of engaging local business in a meaningful manner. Vertical integration in the mining sector is therefore something the Government is actively addressing. Significant opportunities exist to create value adding backward and forward linkages in Zambia's mining sector. Currently, the framework that is managed under the Zambia Development Agency (ZDA) Act provides for a number of incentives that support business developments.

The study conducted by (ICCM, 2014) on behalf of the Chamber of Mines of Zambia indicated that although the majority of goods are procured from Zambian companies, many of these goods are imported by local agents who then supply the goods to the mining companies; thus, very few of the goods procured by the mining sector are actually manufactured in Zambia. In contrast, most of the services needed by mining companies are procured from Zambian businesses and provided by Zambian nationals. During the period until 1997, when ZCCM operated the mines in the Copperbelt, a policy of local procurement was strictly adhered to. This policy, combined with strict foreign exchange controls and measures to protect domestic industry, led to the establishment of a significant manufacturing sector and a relatively diversified local economy. However, in the 1990s, two things happened to change this:

- ZCCM's management challenges combined with a decline in the copper price from the 1980s led to poor supply chain management and eventually to deterioration in suppliers' capabilities, as well as considerable rent seeking by local suppliers – thereby contributing to ZCCM's loss of profitability
- Protectionist policies were dismantled, and domestic manufacturers struggled to remain competitive. After privatization, new mine owners introduced higher standards in their procurement that many local suppliers failed to meet, in particular when competing on similar terms with international producers. Despite these challenges, some supplier firms have survived and the Copper belt's local economy remains more diversified than other provinces outside of Lusaka.

However, in attempting to address the failure to take off, the Zambian government launched a local content strategy that suggests that multinationals and transnationals ought to procure 35 percent of their inputs from the domestic economy.

Identification of key industries is of key importance in the analysis of linkages using input-output tables. The mining sector has been identified as a key sector in the Zambian economy, because of the country's high dependence on the sector for economic growth. Hence, the paper measures linkages between the mining sector with other sectors of the economy, and pays attention to the manufacturing sector.

On the other hand the study undertaken by (UK/AID and world Bank , 2011) observed that manufactured goods, equipment and consumables are expensive and/or difficult to obtain in Zambia; hence mines rely heavily on imports from South Africa and elsewhere. Due to the logistics costs, trade facilitation fees and markups associated with imports, equipment and spare parts in Zambia can cost more than twice what they would in other countries. Motivated by profit, mines are keen to source from the least-cost providers that can meet their standards of quality, quantity and reliability. The greater use of local manufacturers could theoretically reduce the import and logistics-related costs that mines currently incur. Local manufacturers, however, lack the capacity to deliver the more complex, high value-added products that account for the majority of mines' spending at a sufficient quality to meet the needs of the mines. International mine suppliers, who can produce the required quality, have thus far not located in Zambia due to its lack of attractiveness for manufacturing and, until recently, insufficient demand from mines. As a result, the industry buys only low-value items (such as food, clothing, and non-critical services) locally, often from traders rather than local manufacturers. Therefore, developing a high-quality, high-value-added manufacturing base in Zambia that is capable of supplying reliably a number of key products to the mines, will take time and will likely not be feasible for all types of mine supplies. Nevertheless, a more efficient local manufacturing industry could ultimately reduce input costs for the mines, improve industry competitiveness over the longer term, raise the incomes of local producers, and, potentially, help create markets for the copper fabrication industry.

Despite the various structural changes that have occurred in the mining sector beginning 2008; such as the introduction and reversing of the windfall tax among others, the examination of recent National Input-Output table (Central Statistics Office , 2017) indicates that 3,769.7 million kwacha was expended on the mining sector on goods and services, with domestic suppliers supplying goods and services worth 3,005.1 million kwacha and with import sources supplying 764.4 million worth of goods and services from different sectors (Table 1).

Table 1: Input Payments by the Mining Industry in Zambia (k' Millions)

	Locally Produced	Imported
Mining and quarrying	432.3	392.6
Manufactured product	154.3	270.5
Manufacturing products	332.2	13.8
Electricity, gas, steam and air conditioning supply	415.1	0.0
Water supply; sewerage, waste management and remediation activities	110.8	
Construction work	150.3	7.4
Wholesale and retail trade; repair of motor vehicles and motorcycles	240.2	
Transportation and storage services	241.2	12.3
Accommodation and food services	55.4	37.8
Information and communication services	55.6	0.7
Financial and insurance services	24.0	0.2
Real estate services	47.0	
Professional, scientific and technical services	199.4	10.3
Administrative and support service	464.2	19.0
Education	59.7	
Human health and social work services	23.5	
Total Basic Price	3,005.1	764.6

Source: Central Statistics Office, 2017

Of the total 3,769.7 million consumed by the mines from 16 suppliers as indicated by table 1 above, 3,197.4 million (approximately 85%) was supplied by the 8 top suppliers, with 6 of these supplying

71% of all domestically produced goods and services procured in the mining sector. It is worth-noting that this does not entirely indicate inter-industry dependence, it however, reveals the low levels of which mining is integrated with the rest of the economy, and the extent which the integration is limited to only a few productive sectors. From the above it could clearly be noted that Zambia just like many developing countries lacks substantive and complex inter-industry connections with most domestic payments having been done for mining and quarrying (432.3 million); manufacturing products (332.2 million); electricity supply, sewerage and air conditioning supply (415.1 million) ; and administrative and support services (464.2 million). This clearly demonstrates how constrained the spread of multiplier effects of mining activity are, and generally demonstrating how regardless of the majority of payments having been done towards domestic suppliers the sector the economy is still struggling to diversify and reduce on the commodity dependence that has occurred since independence.

To effectively impact the economy, unlike the case observed above regarding the top suppliers in the mining sector, significant contribution by the mining sector would be facilitated by manufactured products. However, the table above highlights that manufactured products are mainly sourced from foreign markets with import value for period under review amounting to 270.5 million kwacha and 154.3 million kwacha from domestic markets. Expressed as percentages of the value procured in the sector shows that 36% was procured from domestic sources and the 64% was procured from import sources. This therefore, demonstrates that the state of imports of manufactured goods in the mining sector is high, therefore demonstrating how desperate Zambia has been in trying to reduce the import dependence and justifying the need for enhanced local development linkage.

3.3.2 Multi Facility Economic Zones / Priority Sector Incentives

In order to encourage manufacturing in the country and promote mining linkage development, (ZDA, 2013), a number of incentives have been provided for such as:

- Machinery for manufacturing qualifies for wear and tear allowance of 50% per annum on the first two years.
- Customs duty exemption on most capital machinery and equipment for manufacturing
- Reduced duty on, or duty free imports of certain raw materials

The priority sectors as related to mining linkages are: the manufacture of: machinery and machinery components iron and steel products, electrical and electronic products and components and parts thereof; chemicals and petrochemicals, pharmaceutical and related products, wood and wood products, transport equipment, component & accessories, clay-based, sand-based and other non-metallic mineral products, plastic products, scientific, and measuring devices/parts, rubber products, leather & leather products, packaging & printing materials and cement. Hence there are massive investment opportunities in processing copper, iron ore and steel, cobalt and other minerals into intermediate and finished engineering products. Engineering products have a ready local market from the mines (for the supply of mining equipment), construction companies, and other industries. The manufacture of engineering products includes metal items such as window frames, doors and roofing materials, as well as nuts and bolts, as well as light engineering products such as;

cable carbon brushes, switchgears, pipes and rail sleepers. Furthermore Zambia imports all major spare parts required for machinery and therefore investment opportunities also exist in the local manufacture of spare parts for various industrial machines.

4.0 Discussion

Ghana, Botswana and Zambia are resource rich mineral countries in Africa. These countries continue to mine and export minerals in raw form with very little value addition. In both countries, the linkage with the manufacturing sector is very weak due to a variety reasons: in Ghana the structural weaknesses impede its drive for economic diversification and industrialization (ACET, 2017). Local firms are also largely unable to meet the supply needs of the mining industry in terms of quality, quantity and timeliness. Other issues include the strong presence of a class of business people who are essentially intermediaries between industry; a lack of political consensus on *how* to strengthen linkages between mining and industrialization; red tape and other difficulties in the business environment; and insufficient commitment from the mining industry itself to support local sourcing. In Botswana, the small manufacturing and agriculture sectors, which hold the key to job creation and enhanced domestic linkages remain constrained because of a multitude of structural challenges, including low and unreliable rainfall (for agriculture) and supply problems related to electricity and water (for manufacturing). The manufacturing sector remains small in Botswana despite Government efforts to accelerate value addition to minerals.

The case of Zambia is similar, Local manufacturers, lack the capacity to deliver the more complex, high value-added products that account for the majority of mines' spending at a sufficient quality to meet the needs of the mines. International mine suppliers, who can produce the required quality, have thus far not located in Zambia due to its lack of attractiveness for manufacturing and, until recently, insufficient demand from mines. As a result, the industry buys only low-value items (such as food, clothing, and non-critical services) locally, often from traders rather than local manufacturers.

5.0 Conclusion

Governments in these countries need to:

i. Establish and strengthen local manufacturing firms to meet the supply needs of the mining industry in terms of quality and quantity;
ii. Attract international firms to set up manufacturing business while at the same time discourage trading in imported goods and services;
iii. Continue to provide incentives in form of tax rebates for attracting foreign investments
iv. Encourage integration of the local manufacturers and suppliers with existing economic Zones

6.0 Recommend

Governments should consider:

i. Address structural weakness that imped economic diversification and industrialization
ii. Rewarding mining companies some form of incentives that are willing to support local manufacturers

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The Significance of Earthquake Incidences on Structural Design in Zambia

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Abstract

Design and construction practices in Zambia have tended to neglect the risk posed by seismicity on structures, and no earthquake design and construction guides exist. Studies of present and historical seismic activity in Zambia and other places of similar tectonic setting suggest that the effects of earthquakes on structures in Zambia should not be ignored.

The study advanced the hypothesis that there was sufficient evidence to warrant factoring in the risk posed by seismicity in design and construction practices, and hence the need to develop an earthquake design and construction guide. While Zambia's location in the interior of the African plate may suggest low seismic risk, the influence of the East Africa rift valley cannot be underestimated. Even with limited history and records of earthquake activity in Zambia and the sub-region, there are sufficient records of major earthquakes resulting in damage to structures and even loss of lives.

The research methodology included literature review of works related to seismicity in Zambia and the sub-region, analysis of raw earthquake data obtained from earthquake monitoring agencies, review of available seismic reports on Zambia and a questionnaire survey targeted at practicing structural engineers and related professionals. Recorded earthquake events and their physical effects on structures, as well as the prevalent structural design practises amongst engineers in Zambia, were reviewed.

The research indicated evidence of physical damage resulting from earthquakes. The limited seismic consideration prevalent in design practises in Zambia is largely attributed to the lack of an earthquake design guide and the preconceived notion that Zambia and the sub-region have insignificant earthquake activity. It is recommended that the Engineering Institution of Zambia (EIZ) should spearhead development of a seismic design and construction guide for Zambia.

Keywords: Earthquake, Seismicity, Structural Design

1. Background

Earthquakes and seismic activity are a vital consideration in the design of structures throughout the world. However, the lack of localised design standards addressing earthquake and seismic

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activity in Zambia suggests that structures are not consistently designed to resist the effects of earthquake and seismic activity. The impact of earthquakes on structures may range from minor deformations of structural elements to complete catastrophic collapse of the entire structure. Although most earthquakes are moderate in size and destructive potential, a severe earthquake occasionally strikes a community that is not adequately prepared and thousands of lives and billions of dollars in economic investment are lost (FEMA, 2010). One of the key ways a community protects itself from potential earthquake disasters is by adopting and enforcing a building code with appropriate seismic design and construction standards (FEMA, 2010). It is therefore imperative that accurate local information and prediction of seismic activity is obtained and design codes made therefrom.

Seismic activity in Eastern and Southern Africa is controlled by the East Africa Rift System, an intraplate fault line on the Africa plate. The East Africa Rift System forms two main lines, the Eastern and Western Branches. The Eastern Branch extends from the Afar triangle in the north to Northern Tanzania in the south. The Western Branch extends from Lake Albert in the north to the south of Lake Malawi in the south, encompassing lakes Edward and Tanganyika. Zambia's seismic activity is mainly influenced by the Western Branch which passes close to the northern and eastern region of the country. Given the remoteness of the region from the African plate boundaries, seismicity in Southern Africa is largely attributed to intraplate tectonics that globally account for a very small percentage of annually recorded earthquakes. In spite of the very low frequency of occurrence, seismicity associated with intraplate tectonics is complex and will occasionally reach critical values. In Malawi, the Salima earthquake (Ms=6.1) of 10 May 1989 killed 9 people. In Tanzania, the Kasanga earthquake (Ms=7.3) of 13 December 1910 caused significant damage in southern Tanzania (Midzi et al 1999).

1.1 Problem Statement

With increased investments in infrastructure by both private and public institutions, there arises an inevitable need to formulate and enforce localised building codes. While many other aspects of structural design and analysis can be adopted from internationally recognised codes of practice, aspects of structural design addressing seismicity need to be locally formulated. It is common practice to ignore the effects of earthquakes in the absence of local design standards. This, however, could be a risky omission especially with sensitive structures such as bridges, dams and high rise buildings normally built at high costs. The research assessed the significance of earthquake incidences on structural design in Zambia.

2. Regional seismicity

Earthquake activity in the eastern and southern Africa region is characterised by the occurrence of destructive earthquakes which are controlled by the well-known regional tectonic feature, the East Africa Rift system (Midzi, et al., 1999). The East African Rift system (EARS) is a 3,000-km-long Cenozoic age continental rift extending from the Afar triple junction, between the horn of Africa and the Middle East, to western Mozambique. Sectors of active extension occur from the Indian Ocean, west to Botswana and the Democratic Republic of the Congo (DRC). It is the only rift system in the world that is active on a continent-wide scale, providing geologists with a view of how continental rifts develop over time into oceanic spreading centers like the Mid-Atlantic Ridge (Hayes, et al., 2014).

The East African rift system shows up at the surface as a series of several thousand kilometers long aligned successions of adjacent individual tectonic basins (rift valleys), separated from each other by relative shoals and generally bordered by uplifted shoulders. Each basin is controlled by faults and forms a subsiding graben or trough, near one hundred kilometers long, a few tens of kilometres wide, empty or filled with sediments and/or volcanic rocks (Chorowicz, 2005).

Traditionally, an Eastern (including the Ethiopian Rift) and a Western Branch are distinguished (Ring, 2014). The Eastern Branch runs over a distance of 2200 km, from the Afar triangle in the north, through the main Ethiopian rift, the Omo-Turkana lows, the Kenyan (Gregory) rifts, and ends in the basins of the North-Tanzanian divergence in the south (Chorowicz, 2005).

The Western Branch runs over a distance of 2100 km from Lake Albert in the north, to Lake Malawi in the south. It comprises several segments: the northern segment includes Lake Albert, Lake Edward and Lake Kivu basins, turning progressively in trend from NNE to N–S; the central segment trends NW–SE and includes the basins of lakes Tanganyika and Rukwa; the southern segment mainly corresponds to Lake Malawi (Nyasa) and small basins more to the south (Chorowicz, 2005).

A third, south-eastern branch is in the Mozambique Channel. The south-eastern branch comprises N-striking undersea basins located west of the Davie ridge (Chorowicz, 2005).

Seismicity in the East African Rift is widespread, but displays a distinct pattern. Seismicity is characterized by mainly shallow (<40 km) normal faults (earthquakes rupturing as a direct result of extension of the crust), and volcano-tectonic earthquakes. The majority of events occur in the 10–25-km depth range. This pattern is widespread throughout the EARS, and provides insight into the relationship between depth of earthquakes, the deformation of continental lithosphere, and magmatic processes in many sectors of the rift (Hayes, et al., 2014).

Besides the well-known tectonic zones of the East African Rift System (Fairhead and Girdler, 1971), there is a zone of seismicity in and around Zambia. This consists of two main branches. One strikes southwest from the southern end of Lake Tanganyika through Lake Mweru and the border region of Congo DRC and Zambia before terminating at about longitude 22°E. The other is parallel to the first and is located along the Zambia-Zimbabwe border, striking southwest from about 15°S, 30°E through Lake Kariba into Northern Botswana before again terminating at about longitude 22°E. These two zones are joined by an area of diffuse seismic activity northwest of Lake Kariba (Fairhead & Henderson, 1977). Figure 1 shows the East African rift system.

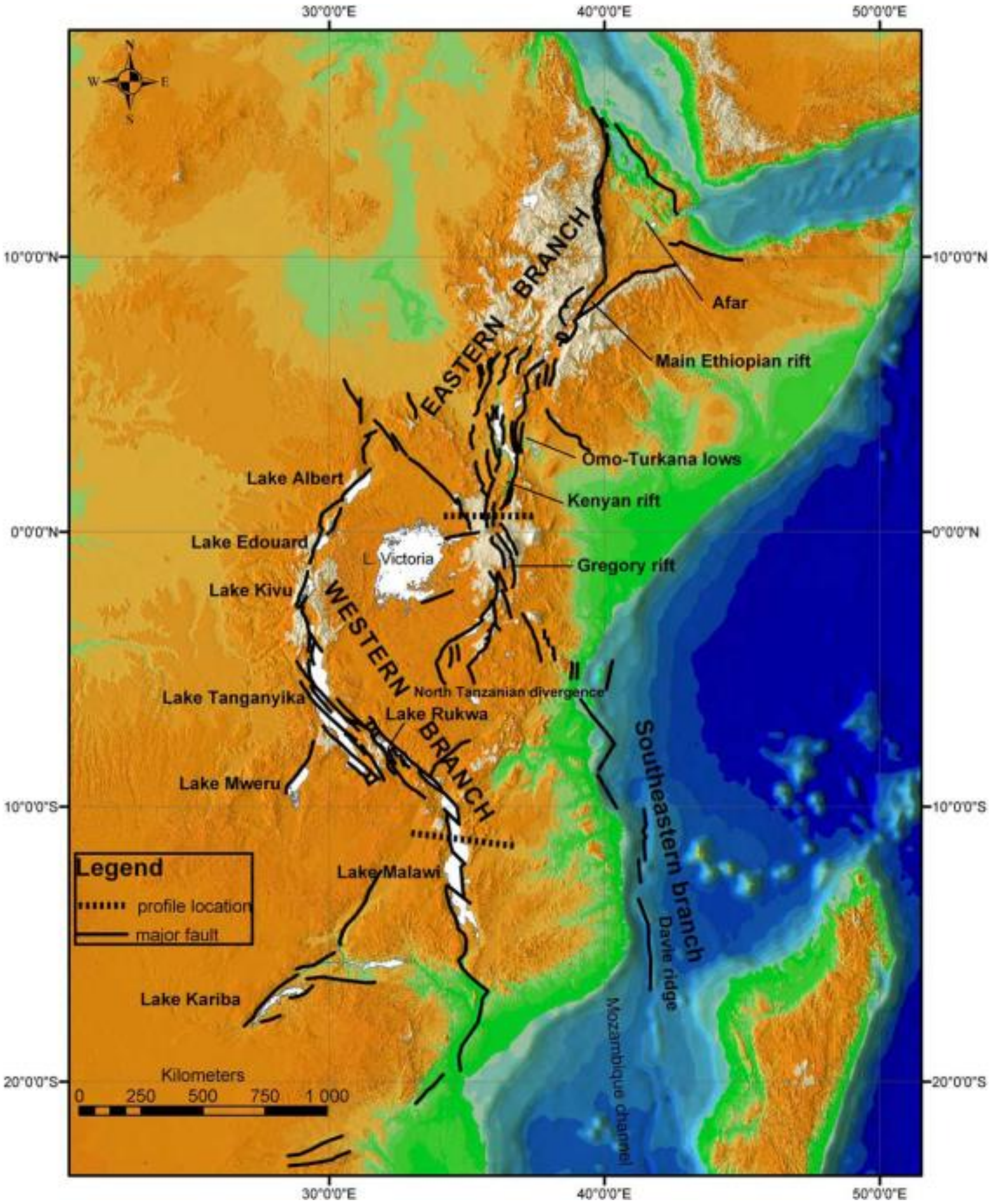

Figure 1: Ilustration of the East Africa Rift System (Chorowicz, 2005)

3. Previous Seismic Hazard Assessments Related to Zambia

The following studies, also highlighted by Studio Pietrangeli (2016), provide insight into seismicity of Zambia and the surrounding areas:

a) 1997, Hlatywayo D.J. Seismic Hazard Estimates in Central-Southern Africa;
b) 1999, Midzi Et al. Seismic hazard assessment in Eastern and Southern Africa;

c) 2007, OCHA. Earthquake Risk in Africa: Modified Mercalli Scale;

Hlatywayo (1997) performed a seismic hazard study for Central-Southern Africa (centred in Zimbabwe). The study followed the conventional probabilistic hazard analysis procedure, defining seismic source zones from seismicity based on instrumental records from a catalogue that spans a period of 83 years. Results were presented as a seismic hazard contour map related to a 100 year recurrence period. The map, presented in Figure 2, shows the peak ground acceleration (PGA) in the southern part of Zambia extending to Lusaka and Central Province ranging from 0.025g to 0.1g in the Mid-Zambezi basin, Luangwa rift and Itezhi-Tezhi.

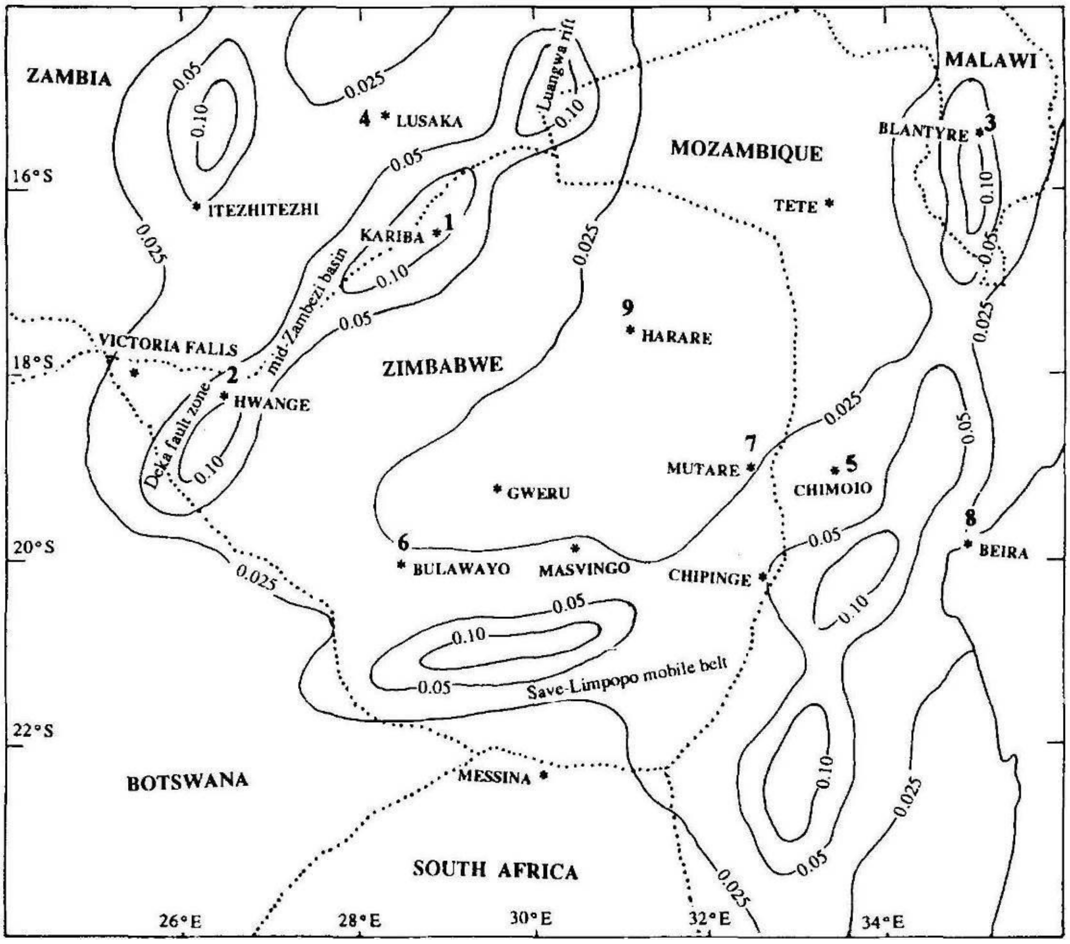

Figure 2 Map of Central-Southern Africa showing, in units of g, PGA contours for a 100 year return period (Hlatywayo, 1997).

Midzi et al. (1999) conducted a probabilistic seismic hazard assessment for Southern and Eastern Africa. Seismic hazard maps for 50, 100, 475 and 945 years return periods were prepared. Figure 3 shows the seismic map developed by Midzi et al (1999) for a 475 year return period, corresponding to a 10% probability of exceedance in 50 years.

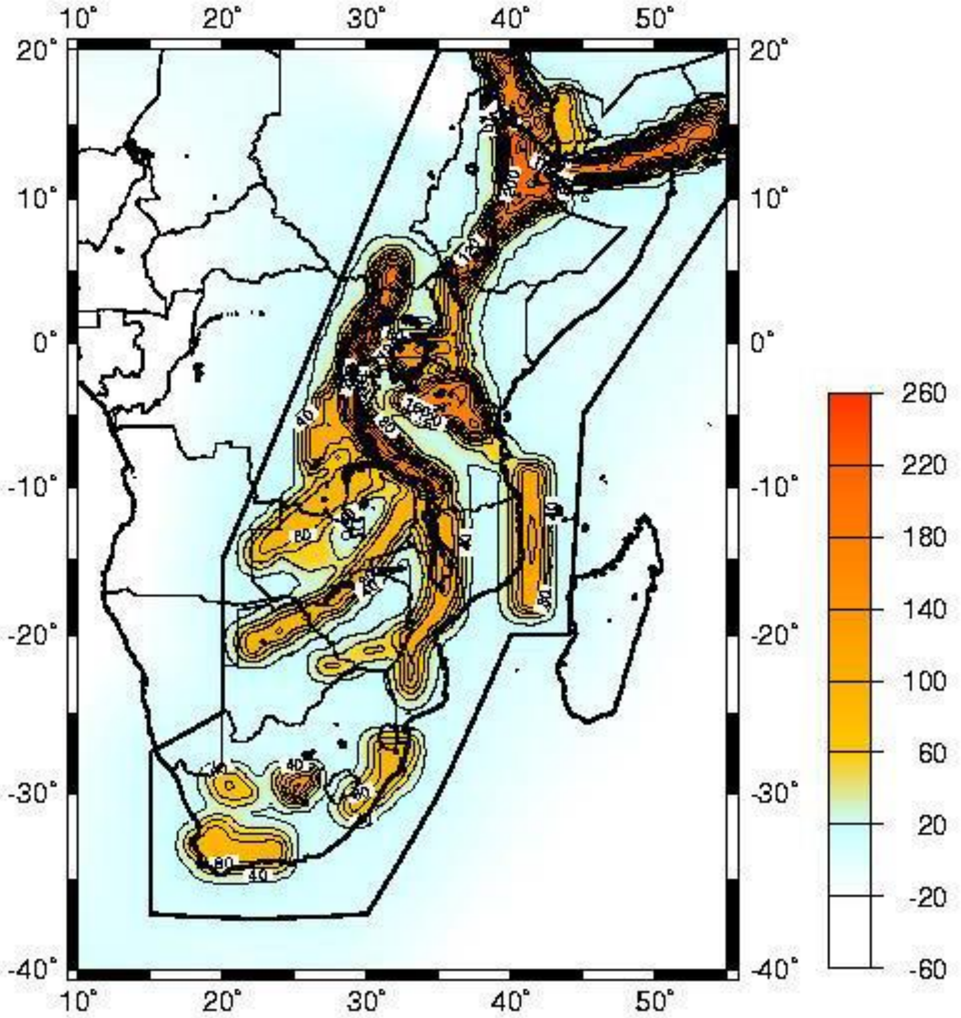

Figure 3 Map showing PGA values measured in gals in Eastern and Southern Africa for a 475 year return period, corresponding to 10% exceedance in 100years, (Midzi et al, 1999)

Figure 4 provided by **United Nations Office for Coordination of Humanitarian Affairs (OCHA, 2007)** shows earthquake intensity zones, for a return period of 225 years, in accordance with the 1956 version of Modified Mercalli scale (MM), describing the effects of an earthquake on the surface of the earth and integrating numerous parameters such as PGA, duration and soil effects as well as historical earthquake reports. The map shows degree of intensity VII (Very Strong) in three zones in Zambia; two parallel NE- SW oriented zones and another connecting the two Zones. The rest of the country lies in degree of intensity zone VI (Strong).

According to the Modified Mercalli scale, an earthquake of degree of intensity VII would cause negligible damage in buildings of good design and construction, slight to moderate damage in well-built ordinary structures but considerable damage in poorly built or badly designed structures.

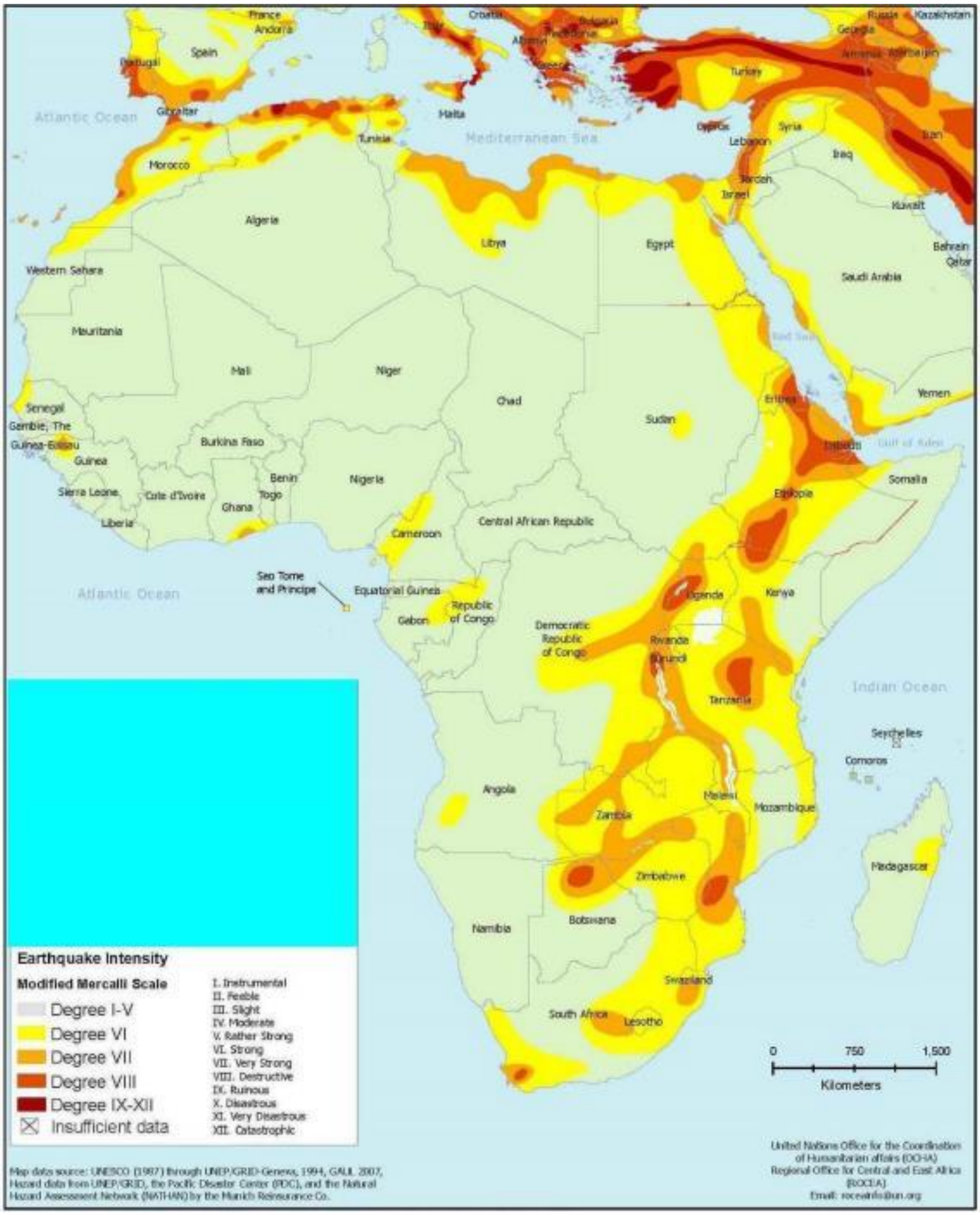

Figure 4 Earthquake Intensity Zones in Africa for a return period of 225 years, in accordance with the 1956 version of Modified Mercalli scale (MM) (OCHA, 2007)

4. Earthquakes in Zambia

4.1 Spatial distribution of Earthquakes in Zambia

The study considered, earthquakes occurring within the area bounded by latitudes -7º to -19º and longitudes 21ºto -35º. The study region extended beyond the boundaries of Zambia into the neighbouring countries to account for seismic effects that extend beyond the vicinity of the epicentre. Figure 5 shows the spatial distribution of earthquakes for the period 1910-2018 in the study area using an earthquake catalogue obtained from the International Seismological Centre (ISC) database.

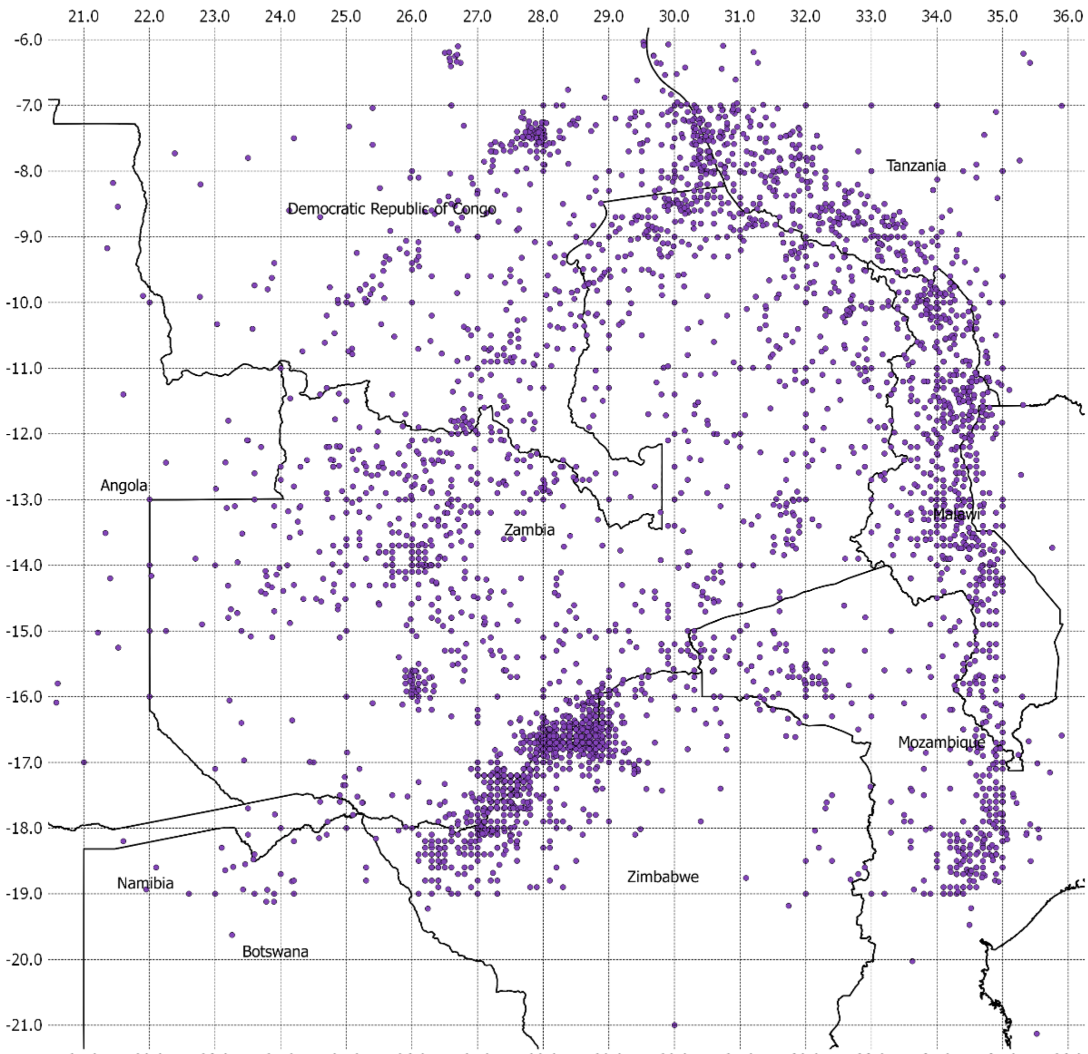

Figure 5 Map showing the distribution of earthquakes in Zambia for the period 1910-2018

A total of 5270 events where obtained from the ISC database to form an earthquake catalogue for the study region representing all available magnitudes. However, magnitudes less than 4 are generally considered to have less significance to structures. Thus 851 events in the catalogue had a magnitude greater or equal 4, as shown in Figure 6. Due to lack of or limited instrumentation prior to 1963, only 50 events were documented prior to 1963.

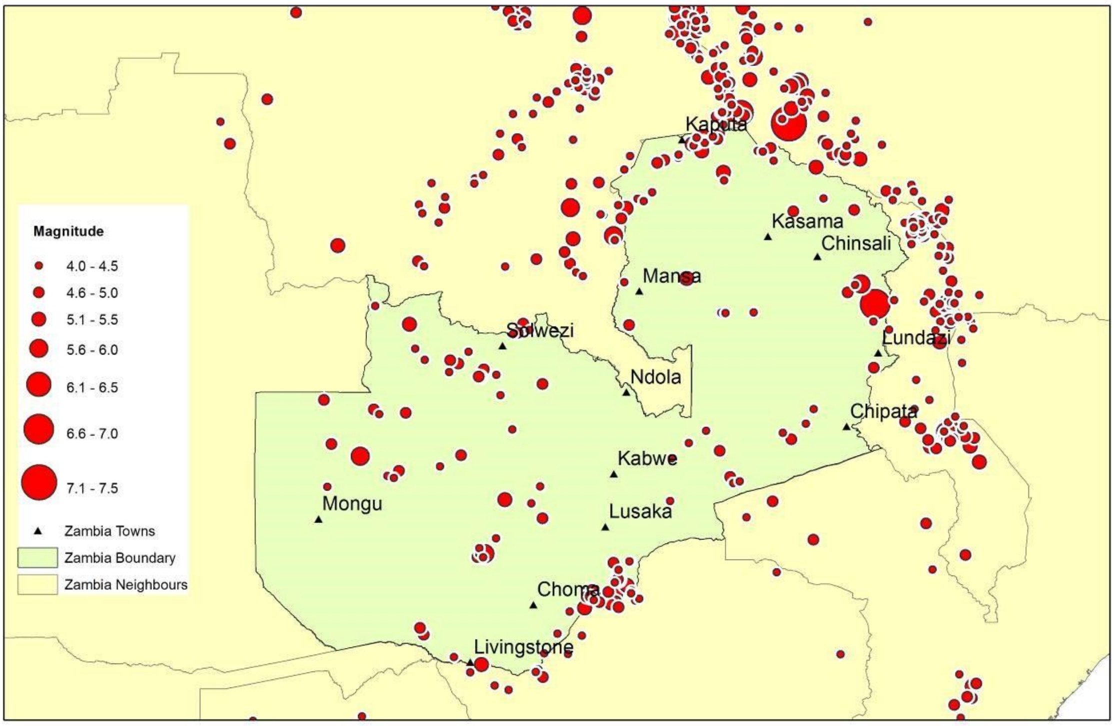

Figure 6 Distribution of earthquakes with magnitude greater than M_w 4 from 1910-2018

The largest earthquake in the data set is a magnitude Mw 7.3 which occurred on December 13, 1910 in Lake Tanganyika Region, about 250km from the border with Zambia. This is reported to have caused significant damage in that region. An Mw 7.2 Earthquake occurred in Tanganyika region on July 8, 1919 at a depth of 15km. This was within 48km of the border with Zambia. The largest earthquake recorded within Zambia was a magnitude Mw 6.7, which occurred south of Chama district on May 1, 1919. Table 1 presents statistics of earthquakes in the study region.

Table 1: Statistics of earthquakes in the study region

Recorded events 1910-2018	5270
Recorded events 1910-1963	50
Recorded events 1963-2018	5220
Recorded events with magnitude > 4	851
Largest recorded event in study region	7.3Mw
Largest recorded event in within Zambia	6.7Mw

4.2 Physical evidence of Earthquake Damage in Zambia

The Zambia Geological Survey Department conducts procedural onsite physical impact assessments after the occurrence of significant earthquakes. The study reviewed reports of three major earthquakes that occurred between January 2016 and June 2017. These were Chirundu (magnitude 4.6), Kaputa (magnitude 5.9) and Lundazi (magnitude 5.2).

The Chirundu Earthquake occurred on 9th January 2016 at 05:25AM at location 16.057˚S, 28.56˚E, 29km from Chirundu town. In the report by Mutamina and Kabele (2016), it was concluded that no physical damage was imposed on structures of the Kariba hydropower plant. However, physical damage was reported on some structures in close proximity to the earthquake epicenter. The damage was in form of cracks on walls of structures.

The Kaputa Earthquake occurred on the 24th of February, 2017, at location 30.0847º E, 8.4695ºS and hypo-central depth of 27km, about 47km east of Kaputa District Administrative Center. According to the report by Kasumba et al (2017), the seismic waves generated by the earthquake were felt in the nearby towns of Northern and Luapula provinces, extending further into the Democratic Republic of Congo and Tanzania, with reports of damage to infrastructure and homes (Figure 7a and b). Fatalities and injuries were also reported.

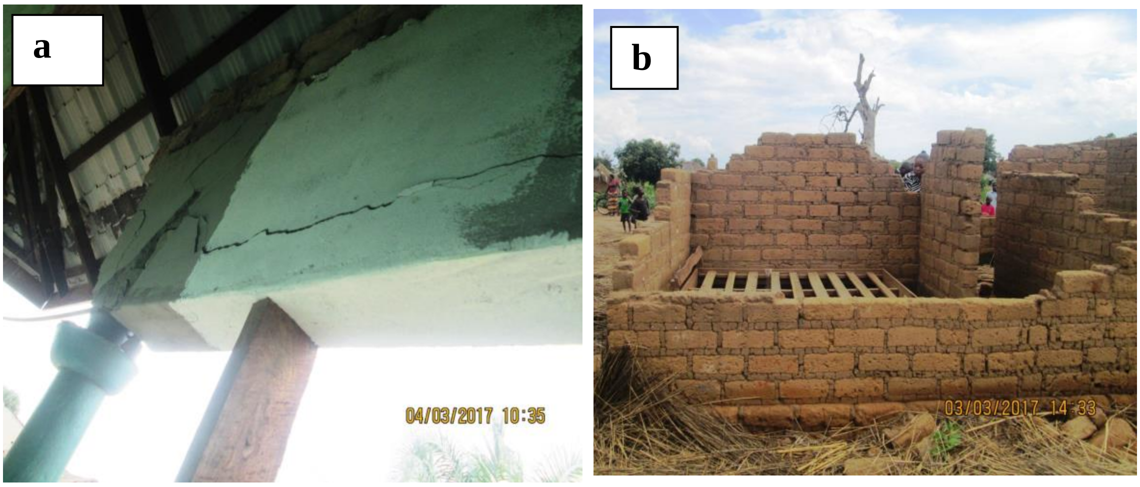

Figure 7a: Structural Damage as a result of Kaputa Earthquake (Kasumba et al, 2017)
Figure 7b: Building collapse as a result of Kaputa Earthquake

The Lundazi Earthquake occurred on 24th June 2017, 12km North East of Chief Kazembe headquarters at location 12.15˚S, 36.65˚E and hypo- central depth of 391km. It was one of three earthquakes of magnitude >5 that occurred in the southern African region within a period of two weeks. The other two occured on 5th July, 2016, of magnitude 5.0, 256km North West of Gaborone and on 24th June, 2017, of magnitude 5.5, 10km North West of Beira in Mozambique. In the report by Mutamina (2017), it was noted that some structures around the vicinity of the epicenter experienced physical damage.

5. Prevalent Seismic Design Practices in Zambia

A questionnaire survey aimed at establishing the prevalent structural design practices relating to seismicity in Zambia was targeted at practicing structural engineers. The engineers were randomly selected from various civil engineering consulting firms, government and quasi government institutions. Current regulation in Zambia does not clearly specify or restrict who among professional registered civil engineers can practice Structural engineering. However, only a limited number of engineers practice structural engineering due to its specialized nature. Therefore, the number of engineers targeted for the study was limited to twenty (20). Precise statistics of engineers practicing structural engineering were not readily available.

The engineers were questioned about their knowledge of seismicity, if they considered seismicity in their design of structures and their sources of seismic data. They were also asked if they believed seismicity was significant to the design of structures in Zambia and whether they believed that structures in Zambia are not adequately designed to withstand seismicity. They were further asked if they had experienced or observed any structural distress (damage) as a result of seismicity in Zambia. In addition, the engineers were asked if they believed there was a need to develop a seismic design and construction guide for Zambia.

Results of the survey show that almost 50% of practicing structural engineers have no knowledge of seismicity, its occurrence and seismic design codes. Forty percent (40%) of the engineers showed limited knowledge while only 10% showed good knowledge of seismicity and seismic design codes. Only 10% of practicing engineers considered seismicity in the design of structures and only limited to critical structures. It was noted that even the few that considered seismicity in the design of structures did not have reliable sources of seismic information. Table 2 summarises the results from the questionnaire survey.

Table 2: Results of questionnaire survey

	None	Limited	Good
Knowledge of seismicity, occurrence, design	50%	40%	10%
Seismic consideration in design and detailing	90%	10%	

A prevalent source of seismic data is a map provided by **United Nations Office for Coordination of Humanitarian Affairs (OCHA, 2007)** showing earthquake intensity zones, for a return period of 225 years, in accordance with the 1956 version of Modified Mercalli scale (MM). According to EN 1998-1, the design seismic action is generally expressed in terms of the seismic action associated with a 10% probability of exceedance in 50 years or a reference return period of 475 years. Therefore, a return period of 225 years, which relates to a 10% exceedance in 24 years, falls short of internationally recognised seismic design practise especially for critical structures. Twenty percent (20%) of the surveyed engineers believed that structures are in Zambia are designed to withstand seismicity, stating that Zambia is a region of low seismicity. Forty percent (40%) believed otherwise. A further 40 percent could not definitively state whether or not structures in Zambia where designed to withstand seismicity. The survey also revealed that over 90% of surveyed engineers did not observe any structural distress or failure resulting from earthquakes in Zambia. Over 90% of surveyed engineers acknowledged the need to develop a seismic design and construction guide.

6. Conclusion

The study highlighted the significance of earthquakes on structural design in Zambia. Zambia lies in the interior of the African plate that is considered relatively aseismic. However, the presence of the East Africa Rift system, with its various sectors, influences seismic activity in the region.

It was revealed in the study that there have been three recent significant earthquake events that have caused serious damage to structures and even loss of lives, recorded in 2017. It is noted that the three earthquake events reviewed in the study occurred in areas that are relatively sparsely populated. Had the incidences been in populated and built up areas, such as the major urban centers of the Copperbelt and Lusaka, the effects would have been more severe.

The study also revealed that limited knowledge existed about seismicity among engineers in Zambia, with almost 50% expressing a lack of knowledge of seismicity, its occurrence and related seismic design codes. It was revealed that seismic consideration in the design of structures was almost non-existent, with only 10% indicating consideration of seismic loading on critical structures. The results of the survey can be attributed to limited guidance on seismic design in Zambia.

While various internationally recognized seismic design codes exist such as the Euro-codes and the Uniform Building Code (UBC), there was limited guidance on the local seismic design parameters required to use the design codes in Zambia. This gap could be closed if seismicity and earthquake design were incorporated in the curricula for engineering students at local training institutions.

It is evident from the study that there is sufficient level of earthquake activity in Zambia to warrant consideration of earthquake actions in the design of structures. Even with the limited history of earthquake event documentation, there are a number of events that should remind engineers of the need to consider seismic loading in the design of structures. According to EN 1998-1, the design seismic action is generally expressed in terms of the seismic action associated with a 10% probability of exceedance in 50 years or a reference return period of 475 years. Although the records of earthquake incidences are only around 100 years, the occurrence of destructive earthquakes anywhere within Zambia, cannot be dismissed.

The periodicities of large earthquakes can be in hundreds of years. For example, investigations into the 1966 Koynanagan earthquake of magnitude 7.0 in the Deccan Plateau of India revealed periodicities of about 200 years for such earthquakes (Brandit, 2011). This particular earthquake occurred in an intraplate region similar to the study region in this study.

7. Recommendations

Based on the findings, it is recommended that **a seismic design and construction guide for Zambia be developed**. This process should be spearheaded by the Engineering Institution of Zambia (EIZ), as an engineering regulating body, in partnership with various engineering research institutions.

The study also recommends the **sensitisation of engineering professionals** on the need to consider seismicity in the design of structures, especially for critical and lifeline structures such as Dams and Bridges. This could be done through workshops, as part of continuous professional development.

It is recommended that **basics of seismicity be incorporated in undergraduate curricula** for all engineering programmes in universities.

Research on earthquakes and seismicity should be supported at research and higher learning institutions, by both Government and industry.

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Review of Engineering Education Curriculum: A University of Zambia Perspective

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Abstract

The University of Zambia's (UNZA) School of Engineering routinely reviews its curriculum to reflect advancements in engineering education world over with special focus to reflect local challenges and solutions. The School embarked on a project to improve graduate employability by looking at different facets of engineering education and whether they are relevant for producing fit-for-purpose graduates. Under the Higher Education Partnership in Sub-Saharan Africa (HEPSSA) program sponsored by the Royal Academy of Engineering (RAE), a structured study was conducted to re-look at the engineering education curriculum. The study was focused on three thematic areas: content; frequency; and quality and challenges related to the curriculum review process. Different stakeholders were consulted via questionnaires and forum discussions at two regional workshops with participation from local industry, four regional universities and one UK advisory university. The responses obtained are well-supported by existing literature. Results from the study showed that the UNZA engineering curriculum needs to be reviewed to reflect a balance between theory and practice and content to focus more on entrepreneurship skills than developing it for monolithic industry. The paper concludes that continuous curriculum review in response to dynamic technological ecosystem along with quality assurance are key to successful engineering curriculum. The paper also presents recommendation on best practices to UNZA. The study also informs two important stakeholders, namely the government and industry, on their important role in helping to deliver relevant and quality engineering education which is tailored to local industry needs as well as driven by the country's developmental agenda stressing that strong synergies of the Triple Helix Model (government, university and industry) are required for mutual benefit.

Keywords: Engineering Education, Curriculum, Curriculum Process, Curriculum Review, Best Practices

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I. Introduction

To produce fit-for-purpose engineering graduates that are relevant to the local industry requires a matching engineering curriculum. Anecdotal evidence suggests local industry is of the view that universities in Zambia are not producing students with the relevant skills. On one hand, local universities also appear to be stuck in their conventional approach to producing graduates with academic skills. The skills provide students with general theoretical knowledge and that help to develop the students' mental ability and problem solving skills. Students learn the engineering working principles and sometimes are involved in real-life examples of the equipment or environment found in industries. On the other hand, is the government, which is perceived to be functionally detached from both university education and industry. Currently, there is debate on the kind of skills that engineering education is required to deliver. Cecilia (2017)) examined key generic competences or skills required for graduates to remain relevant to a growing local industry. The generic skills, which are outside conventional academic skills include communication, teamwork, problem-solving, creativity, time management and moral based attributes. Mahmud et el (2012) argues about what the engineering education curriculum content should be emphasizing and notes that product design and inventing principles should also be key elements of the engineering curriculum. On the other side, industry experts are of the view that students should be equipped with industry specific skills such as use of specific software or standard industrial equipment.

In the research project called Adaptive Education for Employable Engineering Graduates under a Royal Academy of Engineering funded program titled Higher Education Partnership in Sub-Saharan Africa (HEPPSA) undertaken by the School of Engineering at UNZA, four key areas of enhancing engineering education were identified and investigated, namely: industrial training, software training for engineering students, university collaborations within the sub-region, and industry-academia-staff (dual) exchange. At the core of the investigation was curriculum review. The project was thus split into the above work packages to address specific areas of the engineering curriculum. However, a more general outlook involved investigation into the curriculum review process itself. Thus another work package was included with the objective of investigating the curriculum review process. This paper reports on the outcome of this study.

The study was devised to answer the following key questions:

- What should consist the content of the engineering curriculum?
- How frequent should the curriculum review be?
- What challenges are faced during curriculum review and what are the best practices to ensure quality curriculum review process?

II. Engineering Education Curriculum Review – Workshops and Questionnaires

To elicit views from stakeholders on the curriculum review, two regional workshops were planned and held in Lusaka in November 2017 and in Windhoek, Namibia in November 2018. Two questionnaires were also distributed to stakeholders: one to universities; and the another to industries that employ engineering graduates. The responses to the questionnaires were documented and analyzed.

a) Regional Workshops

1. Lusaka Regional Workshop

During stakeholder discussions at the regional workshop in Lusaka, a number of thematic areas were drawn up regarding curriculum review. These were classified as:

- curriculum content review and stakeholder support;
- frequency of curriculum review; and
- quality, challenges and best practices.

Key questions around these themes were developed into questionnaires for both industry and universities. The initial responses from both sectors were interrogated further at the workshop held in Windhoek, Namibia.

2. Windhoek Regional Workshop

During the Windhoek regional workshop, there were extensive discussion on the three themes identified above. Each university gave its perspective on how the curriculum review process works at their university. This is documented under the results section.

b) Questionnaires

The questionnaire survey targeted 8 universities that were part of the hub-spokes interactions within HEPSSA project and 50 industries within Zambia. The closed questions were focused on the above three themes identified in II (a) above.

Twelve (12) academic members of staff from six (6) universities across the region (University of Malawi (UNIMA) from Malawi, Namibia University of Science and Technology from Namibia (NUST), University of Johannesburg (UJ) form South Africa and University of Zambia (UNZA), Copperbelt University (CBU) and Mulungushi University (MU) from Zambia) and one university from United Kingdom (University of Manchester) were consulted and took part in both the workshop and responding to the questionnaire.

Sixty-five (65) questionnaires were returned completed. These were from engineering industries in Zambia and one each from Germany, Namibia and South Africa. The respondents were predominantly engineering graduates from different engineering specializations and across different towns, Johannesburg in South Africa, Mannheim in Germany, Kalumbila, Kitwe, Lusaka, Mazabuka, Mufulira, Mumbwa, Ndola, Solwezi and Kabwe in Zambia. Of the respondents from industries, 7 (10.8%) were female. The responses are discussed in the next section.

III. Engineering Education Curriculum Review Survey Results

a) Regional Workshops

In this section, the current practice of universities with regards to curriculum review and their recommendations for best practice as discussed at the regional workshops are presented.

The School of Engineering at the UNZA has been delivering engineering education since its inception in 1969. Since then much has changed, with the school at the time of this paper producing five engineering disciplines namely, Agricultural, Civil and Environmental, Electrical and Electronic, Geomatics and Mechanical Engineering. Curriculum is reviewed normally every 5 years even though there is no formal procedure to guide the process. Neither is there involvement of central university administration through its recently established Quality Assurance Unit to follow up on quality and procedure of the curriculum review process. The curriculum review process within the university is traditionally school driven but must be approved by the university senate before implementation. The Industry Advisory Board and stakeholder industries are all involved in the review process. Current curriculum is approved by the Zambia Qualification Authority.

In South Africa (SA), and in particular at UJ there is generally a push towards adaptive (continuous) review even though quality checks seem to reduce the efficiency and a general view is that the bureaucracies lengthen the process and should be curtailed. Much of this comes from the fact that the curriculum review is a well-structured process. The process involves University senate approval, Council of Education (government), and is accredited to the Washington Accord. Students also play a role in ensuring that lecturers bring out the set outcomes of courses as set out in the curriculum structure, although lecturers may exercise their freedom to change some content under minor review. Since the current engineering ecosystem is very dynamic, the prolonged curriculum review process means that much of the technology would have changed by the time the curriculum is being implemented. The curriculum review however, focuses on global approach for employability and generally adaptability towards local industry trends. Even with a well-structured and robust review process, developing curriculum that is skill's lead remains a challenge in SA.

In Malawi, at UNIMA the curriculum review process is as bureaucratic as in SA, with some iterations at some stages for quality checks. This ensures quality review with time being traded off. A review consists of major and minor. A policy guides that any program should have a major review every after five years. Minor reviews usually done mid-term are also common and often skips other bureaucratic stages with a seal of approval being done by the vice chancellor. Their major challenge is funding for curriculum review. Without central funding from the government, the faculty staff sources funds elsewhere, often from the industry.

MU is a relatively new university built from UNZA. So it has inherited most practices from its parent university. They equally involve other stakeholders in the review, then followed by university senate approval, before Higher Education Authority (HEA) accreditation. They face similar challenges of funds and resources to manage the review process.

Finally, at NUST, their system is modelled after the South African system because of historical roots. A major curriculum review is over a five-year cycle. The process involves the following: Needs analysis (Questionnaire, interviews), department, program development unit, Quality Assurance, teaching and learning unit, industrial advisory board, Senate, registration on the frame National Qualification Authority (NQA) and National council. The NQA framework bench marks the curriculum against best practices (Engineering Council of South Africa (ECSA), Washington-Sydney Dublin Accords etc.) and assesses the knowledge areas and outcomes. As in SA, a quality assurance unit within the university ensures adherence to quality. This rigorous process focuses not merely on the actual content but more pertinently the outlook of the graduates after graduation. To incentify lecturers to apportion time to do the review, each participant is awarded 5 hours a week for the review. Furthermore, Lecturers have flexibility as regards text books and assessment criteria. A minor review is allowed within 10% of the course content.

b) Questionnaires

The responses to the two questionnaires administered are outlined below. The responses are in line with the three themes identified.

1. Curriculum Content Review and Stakeholder Support

The curriculum review may include a number of components such as theory, practical (that include laboratories, workshops and field work), industrial training or internships, communication skills, and university-industry partnership collaboration in terms of teaching and research. Most universities highly rated the practical aspect as the most important component. Strengthened university-industry partnerships in engineering education was also seen as a key component and was rated high by universities. The survey also showed that industrial training is important for students to get practical skills that prepare them for the world of work. Industry respondents also agree that practical training, university-industry collaboration and industrial training were key to the success of any engineering education curriculum.

Stakeholders' involvement in curriculum review is also important. Most of the respondents from both universities and industry in the survey were of the view that academic and support staff together with industry should be involved in the curriculum review process, as well as the advisory board. Industry stressed the importance of the quality assurance unit within the university to superintend over the curriculum review process. To ensure wide stakeholder participation, various fora should be put in place including advisory board meeting, departmental workshop, school workshop, government and industry workshops and community engagement. Almost all these were present in most universities surveyed. However, industry is of the view that there could be other forums such as alumni networks, conferences and other professional forums to open up wider discussions over the issues surrounding the engineering curriculum.

Half (6) of the respondents from universities in the survey indicated that their curricula meet local industry requirement and expectations, while more than half (52.3% with additional 27.7% not sure) of the total respondents (65) from industries survey disagreed. This is no doubt expected as alluded to earlier, because both sides (university and industry) are at odd as to what is expected of engineering education.

But what though, is the main driver to the review of the curriculum review? According to this survey, the responses to changes in the industry was cited as the main driver. This is because most people think that engineering curriculum should be based on local industries. This in fact is what spurred this project on from inception. To build university and industry relationship so that the school of engineering at UNZA could produce fit-for-purpose graduates. As shown from this survey, improvement in the engineering curriculum may include remolding curriculum so that it is skewed towards local industrial requirements. But are the engineering industries doing much to support curriculum development in universities? The answer was a resounding no. So how can the universities develop programs to meet the industries expectations when industries are not interested in what the universities are doing? There seems to be a war of words going-on on this. Industries think the universities should actively seek industry help and partnership because companies exists to make profit, as long as universities do not prove their worth, they will not be equal partners or rather the partnership would not even work. On the other hand, universities think that some industries do not strive to help build local universities for mutual benefits. May be it's the chicken and egg problem, whatever the case, university-industry collaboration is fundamental to success of engineering education. Worse still, government seems to have little or no impact on improving engineering education. Limited grant supporting mainly salaries means there is literally no funding toward research and there is no government driven university-industry partnership to tackle local engineering problems.

Industrial exposure is also key to ensuring engineering students have relevant industrial training. In the universities surveyed, most students spend between 1 to 3 months doing industrial training. The survey showed that this time is inadequate to allow students to make practical applications of the theoretical concepts they already learnt in the classroom. The current practice where students are attached beginning in third year to fourth year at UNZA is not at variance with what the survey results (universities and industries) suggests. This industrial training program has largely been perceived to be successful in producing graduates with the right exposure to local industry.

2. Frequency of Curriculum Review

A nominal frequency for curriculum review is five years as is the practice with most universities surveyed. A good curriculum review frequency takes into account any rapid changes in the technology but also gives sufficient time for the implementation, monitoring and evaluation of the effectiveness of the previous curriculum. The curriculum review, as many believe (universities and industries), should be necessitated by the changes in industry, but must be driven by the university (e.g. engineering school). However, universities should allow for annual approval of minor reviews without being subjected to much formal university approval process as is normally the case.

3. Quality, Challenges and Best Practices

To ensure quality in the curriculum review process, many universities employ a separate unit called quality assurance unit. Where this is implemented, it sets the standards for the review process, and quality checks in the development and implementation of the curriculum. Even though, as pointed out earlier, many stakeholders are involved in the curriculum review process, the main driver to the curriculum review is the course lecturer.

The engineering curriculum review has its challenges. The major challenges are; cost of comprehensive curriculum review being prohibitive, university bureaucracy and lack of dedicated staff (lecturers) to conduct the curriculum review. Although there are many other minor contributing factors – such as demotivated staff, a gap between what industry perceives to be the immediate challenges they need solutions for and the academia, lack of strong collaboration with industries – it was clear from the survey that unless there is an industry funded and driven curriculum review, the university needs to source its own funding towards curriculum review. Improving the efficiency of the quality assurance checks (bureaucracies), and rather than incentify staff with extra pay for the curriculum review, the university may instead seek dedicated staff to do the major curriculum review who should dedicate quality time and effort.

Most universities ranked theory and practical components of the curriculum as that which need major attention to achieving high quality engineering education. Although there was an indication that industrial training, communication skills, government-university-industry collaboration were also very important, these components may make the difference between producing mere graduates or producing fit-for-purpose graduates.

IV. Discussion and Recommendations

In this section, a discussion of the result of the survey focusing on each theme is discussed along with recommendation for the best practices.

a) Curriculum Review Content and Stakeholder's Support

Curriculum review is one of the key aspect that underpins the competitiveness of any university. It was concluded that curriculum review should strike a good balance between theory and practice. Well it is obvious that the curriculum review should be fit-for-purpose, the issue of employability and entrepreneurship seems to split opinions from both academia and industry. What is clear though is that the curriculum should be relevant to the local industries to make the graduates more employable. However, in view of the current throttled local job market, many researchers think that the curriculum should be skewed more towards entrepreneurship rather than employability (Karim, 2015). Thus they argue that a good engineering curriculum should focus on building entrepreneurship skills. Nevertheless, to develop a good curriculum that is aligned with the local needs of industries (Parashar, 2012), key skills competencies and specific learning outcomes need to be identified first. Then content may be generated to meet the competencies and learning outcomes anticipated.

Furthermore, content should not be developed for monolithic industry. Thus learning institutions ensures that their graduates have adequate knowledge in the field of specialization and have adequately developed their thinking ability and mental prowess to be trainable in any related industry. This ensures that graduates have wider job markets, for examples engineers can work as bankers, sales personnel, economist, accountants etc. as opposed to more conventional engineering roles. The technical perspective engineers bring further enhances these roles. The

A good Engineering Curriculum Should…

- strike a balance between theory and practice
- define specific learning outcomes that students should meet
- focus more on entrepreneurship skills than on employability
- should demonstrate local significance and not developed for monolithic industry
- be anchored on strong synergies of the Triple Helix Model (government-university-industry)
- be reviewed every after 5 years with flexible minor reviews done annually
- be overseen by a good quality assurance unit to ensure monitoring and evaluation of its effectiveness
- be benchmarked against best practices

triple helix model (Abbas, 2019) should be strengthened to make sure that there are strong links to government and industry for mutual benefits (Salleh, 2013). It was observed that the university industry relationship was more of parasitic rather than symbiotic one. Universities were unanimous that the universities should be at the fore front of solving local industry problems and that government policies, regulations and support should be tailored to enhance universities in research and public service. The industry should take up further roles in mentorship, advisory board membership, in-kind grants support to universities, capacity in terms of guest lecturers, feedback via active support in curriculum development and support in terms of equipment. Paramount to these engineering learning institutions is the advisory board. A well formulated board will be influential in all aspect of engineering curriculum development.

b) Frequency of Curriculum Review

The frequency of the engineering curriculum review was re-sounding. Most of the stakeholders in this research, thought that an appropriate cycle should be about 5 years for major review. However, the bureaucratic quality checks should be balanced with the resource availability in terms of lecturers and time required for the review and relevance as the engineering industry is evolving at a rapid rate these days. In general curriculum review should be a continuous process reflected in minor reviews done annually. Others researchers argue that the review process should be part of a job-description to review/moderate the courses annually and should reflect a good balance between the review exercise and lecturer load. While the norm in other universities is that extra time needed to perform the curriculum review is credited to an individual lecturer and accounted for as work load. Irrespective of the different practices, it is recommended that engineering curriculum review should be a continuous process which is reflective of the dynamic engineering ecosystem and reflect a good balance between rigor and rigid.

c) Quality, Challenges and Best Practices

Curriculum review works well with a well-established quality assurance directory/unit (Bais, 2011). Thus most university have a quality assurance (QA) unit that oversees the curriculum review. This was certainly the case at UJ, NUST, UNIMA, UNZA and MU. It was clear that most universities in South Africa, Namibia and United Kingdom have a quality assurance unit. The administration of QA includes peer review and student's evaluation of teaching and transparency. A good QA system also ensures that sufficient time is given to the review process by reducing teaching/research loads for lecturers involved in the review process. It also checks the competences of lecturers and their preparedness to deliver quality engineering education. A good system links lecturer's performance to the appraisal system. A good QA system should incorporate monitoring and evaluation. It should also determine the threshold of whether it is minor or major review of the curriculum. National qualification units within countries also provide country level quality check. (e.g. Zambia Qualification Authority (ZQA), Zambia, National Qualification Authority (NQA), Namibia, Engineering Council of South Africa (ECSA), South Africa). Further quality could be enhanced by accreditation to international boards such as Washington Accords. Quality assurance ensures that the curriculum review delivers quality and relevant engineering curriculum.

d) Recommendations to UNZA School of Engineering:

1. UNZA should revise its engineering curriculum and ensures that content should:
   - focus more on entrepreneurship rather than on employability
   - strike a balance between theory and practice
   - have specific learning outcomes that students should meet
   - demonstrate local significance and not developed for monolithic industry

2. A good engineering ecosystem for any university should be anchored on strong synergies of the Triple Helix Model (government-university-industry). UNZA should build strong relationships with local industries and actively make initiative to engage local industries on solving local engineering challenges. UNZA should also recommend to government on policy directions that enables an excellent technological ecosystem.
3. UNZA should enforce a five-year engineering curriculum review cycle. It is recommended that approval be sort from the university senate to implement annual minor review reflecting about 10% of the curriculum content in each course. Lecturer participating in the major curriculum review should have their teaching and research load reduced or else the School of Engineering should outsource resources to do the curriculum review.
4. The curriculum review should be seen by a quality assurance unit. UNZA should recommend a quality assurance person to be attached to the school to oversee issues of quality. This quality assurance should not only ensure compliance with local standards but also accreditation to international standards. UNZA should accredit its engineering programs to international accords to ensure best practices.

V. Conclusion

It is true from the foregoing that a good engineering curriculum is essential in producing fit-for-purpose engineering graduates. In view of the international forum discussions and the questionnaire results, a number of recommendations were made on the engineering education curriculum. Engineering curriculum should reflect a good balance between theory and practice. However, education providers should be careful not to produce content for monolithic industry but rather a local industry skewed curriculum should focus on developing entrepreneurship skills rather than employability. Additionally, the triple helix model should be strengthened for mutual benefits.

The engineering curriculum should be continuously reviewed in response to the dynamic technology advancement in the engineering ecosystem. A five (5) year major review cycle is recommended with a flexibility for annual minor reviews.

Engineering education curriculum review has many challenges. The major ones are; cost of comprehensive review, bureaucratic quality assurance process, and lack of skilled and sufficient human resource. While it is true that funding could help address these challenges, a well implemented quality assurance process with local and international accreditation is required.

VI. Acknowledgment

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